# TRANSACTIONS

OF THE

# AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

(INCORPORATED)

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## VOL. LXIII

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CONTAINING PAPERS AND DISCUSSIONS ON GEOLOGY, MINING, MILLING AND COAL, PRESENTED AT THE CHICAGO MEETING, SEPTEMBER, 1919, AND AT THE NEW YORK MEETING, FEBRUARY, 1920.

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# PREFACE

In this volume are the papers and discussions on Geology, Mining, Milling and Coal that were presented at the Chicago meeting in September, 1919, and at the New York meeting in February, 1920.

These papers were printed in *Bulletins* 147 to 153, and as Sections of MINING AND METALLURGY in January and February, 1920, but because some of the papers presented at these meetings are reserved for future volumes, only *Bulletins* 147 to 149 are entirely superseded.

The Proceedings of the New York meeting, February, 1920, including reports of Officers and Committees for the year 1919, are printed in this volume

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## Officers and Directors for Year Ending February, 1921

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[1] Until Feb., 1921. [2] Until Feb., 1922 [3] Until Feb., 1923.

# HONORARY MEMBERS

| Year of Election to Honorary Membership | Name | Residence |
|---|---|---|
| 1876. | PROF. RICHARD ÅKERMAN | Stockholm, Sweden. |
| 1913. | DR. FRANK DAWSON ADAMS | Montreal, Canada. |
| 1888. | PROF. HATON DE LA GOUPILLIERE | Paris, France. |
| 1906. | SIR ROBERT A. HADFIELD | London, England. |
| 1917. | HERBERT HOOVER | Stanford University, Cal. |
| 1919. | ROBERT W. HUNT | Chicago, Ill. |
| 1915. | JAMES FURMAN KEMP | New York, N. Y. |
| 1905. | PROF. HENRI LOUIS LE CHATELIER | Paris, France. |
| 1913. | EZEQUIEL ORDOÑEZ | Mexico City, Mexico. |
| 1909. | ALEXANDRE POURCEL | Paris, France. |
| 1911. | PROF. ROBERT H. RICHARDS | Boston, Mass. |
| 1906. | JOHN E. STEAD | Middlesbrough, England. |
| 1902. | PROF. DIMITRY CONSTANTIN TSCHERNOFF | Petrograd, Russia. |
| 1910. | PROF. TSUNASHIRO WADA | Tokyo, Japan. |
| 1907. | CHARLES D. WALCOTT | Washington, D. C. |

## HONORARY MEMBERS (*Deceased*)

| Year of Election | Name | Year of Decease |
|---|---|---|
| 1872. | SIR LOWTHIAN BELL | 1904 |
| 1905. | ANDREW CARNEGIE | 1919 |
| 1892. | A. DEL CASTILLO | 1895 |
| 1902. | MANUEL MARIA CONTRERAS | 1902 |
| 1888. | A. DAUBRÉE | 1896 |
| 1906. | JAMES DOUGLAS | 1918 |
| 1884. | THOMAS M. DROWN | 1904 |
| 1890. | MORITZ GAETZSCHMANN | 1895 |
| 1873. | L. GRUNER | 1883 |
| 1891. | BRUNO KERL | 1905 |
| 1895. | JOSEPH LE CONTE | 1901 |
| 1891. | J. P. LESLEY | 1903 |
| 1899. | FLORIS OSMOND | 1912 |
| 1890. | ADOLPH PATERA | 1890 |
| 1886. | JOHN PERCY | 1889 |
| 1888. | FRANZ POSEPNY | 1895 |
| 1911. | ROSSITER W. RAYMOND | 1918 |
| 1884. | THEODOR RICHTER | 1898 |
| 1899. | W. C. ROBERTS-AUSTEN | 1902 |
| 1890. | ALBERT SERLO | 1898 |
| 1880. | C. WILLIAM SIEMENS | 1883 |
| 1909. | JAMES M. SWANK | 1914 |
| 1872. | DAVID THOMAS | 1882 |
| 1873. | PETER R. VON TUNNER | 1897 |
| 1885. | HERMANN WEDDING | 1908 |

## Proceedings of One Hundred and Twenty-first Meeting, New York City, February, 1920

THE 121st meeting of the Institute held in New York City, Feb.16 to 19, 1920, was a great success despite vicissitudes of weather of unusual severity. On account of snowstorms, only two members of the Canadian contingent arrived, and several members from the west arrived thirty hours late. Two weeks prior to the meeting, the heaviest snowstorm in 24 years wrecked the surface transportation in New York; and it was only by special dispensation that arrangements were effected with the New York Street Cleaning Department to clear away 39th street in front of the Engineering Societies Building so that vehicles could reach the curb. The total registration of members and guests was 1138.

### THE ANNUAL SMOKER

The annual smoker was held on the fifth floor of the Engineering Societies Building, Monday evening, Feb. 17. The attendance, as usual, was between four and five hundred.

### THE LADIES ENTERTAINMENT

The ladies were entertained at the luncheons at Institute headquarters each noon. For Monday afternoon, a trip was planned to various art museums but, because the heavy snow made it impossible for automobiles to travel anywhere except on Fifth Avenue, there was substituted a visit to the Public Library, where a collection of enlargements of photographs of the Signal Corps taken in France was opened especially for the A. I. M. E. ladies. This was followed by a delightful tea at Mrs. Mark L. Requa's apartment on Fifth Avenue. In the evening, the ladies were entertained at the Capitol Theater, the largest theater in the world which shows moving pictures with an ever-changing bill of frivolity review.

Tuesday afternoon, Senator W. A. Clark generously, as he has so many times in the past, opened his art galleries to our members; this was followed by tea.

On Wednesday afternoon, the ladies were entertained at a matinee party to see "Monsieur Beaucaire," which was pronounced one of the most delightful of musical plays.

Many of the ladies braved the unpropitious weather Thursday morning and joined the party visiting the Bush Terminal.

## Annual Banquet at Waldorf

The annual banquet of the Institute was held at the Waldorf Tuesday evening and was preceded as usual by a reception, given by Mr and Mrs. Winchell to Mr. and Mrs. Hoover. There were only three speakers, Mr. Winchell, Mr. Hoover, and Professor Kemp. Lawrence Addicks presided as toastmaster as happily as he did the year before. The dinner was followed by a dance, which was continued until 2 A. M.

## Excursion to Bush Terminal

On Thursday, an excursion was made to the Bush Terminal in Brooklyn. Owing to a fierce snowstorm, which began early Thursday morning, many members who had registered for this trip and who could not foresee that at noon the sun would be shining, were prevented from visiting a most interesting place. Besides visiting the warehouses, where many steamers were loading and unloading interesting and varied cargoes, from Red Cross supplies to crude rubber from the tropics, there was loft building after loft building which revealed the complete manufacturing plant for Lily drinking cups; a complete cigar manufacturing plant of the U. S. Cigar Stores Co.; a complete paper mill with a capacity of 100 tons of newsprint per day, which is owned and operated by the New York *Times* for its own consumption.

## Annual Meeting

At the annual meeting of the American Institute of Mining and Metallurgical Engineers, held at the headquarters of the Institute, on Tuesday, February 17, 1920, at 9.30 A.M., President Horace V. Winchell in the chair, about sixty members were present in person, and over twenty-five hundred members voted by letter ballot.

The minutes of the annual meeting on February 18, 1919, were presented and approved.

The following reports were presented in printed form, and were accepted and ordered filed: Report of the President, Report of The Treasurer, Report of the Secretary.

The Tellers appointed to count the ballots for Officers and Directors presented the following report:

For Director and President, Herbert Hoover, 2274.

For Directors and Vice-presidents, S. W. Mudd, 2208; Frederick Laist, 2195.

For Director, A. S. Dwight, 2202; R. M. Catlin, 2187; W. A. Carlyle, 2183; William M. Corse, 2182; G. H. Clevenger, 2180.

The President thereupon declared these men elected.

The report of the Tellers on the proposed amendment to the Consti-

tution increasing the annual dues made the following report 1727 ballots in favor of the amendment; 361 ballots opposed to the amendment

The President announced that the amendment had carried.

### *Result of Ballot on Simplified Spelling*

The report of the Tellers on the proposed changes in spelling presented by the petitioners was as follows·

First Proposal.—836 ballots opposed to the proposal, 152 ballots in favor of all of the words, 441 ballots in favor of part of the words, with a total of 593. No expression of opinion, 726 ballots.

It was not considered necessary by the Tellers to count the ballots for each individual word because of the preponderance of votes opposed to the change.

Second Proposal.—652 ballots in favor of the proposal, 1081 opposed to the proposal. No expression of opinion, 250 ballots.

Third Proposal.—756 ballots in favor of the proposal, 1090 ballots opposed to the proposal. No expression of opinion, 303 ballots.

Whatever discrepancies may appear in the total number of ballots cast in each of the three proposals are due to mutilated ballots.

The President announced that all three proposals had been defeated.

### *Reports of Committees*

The following reports of standing committees were presented in printed form, and were accepted and ordered filed: Executive Committee, Finance Committee, Membership Committee, Committee on Papers and Publications, Library Committee.

The report of the Committee on Development of the Activities of the Institute was presented in mimeographed form, together with the Joint Conference Committee of the four Founder Societies. The following resolution was thereupon moved by J. W. Richards, seconded by J. Parke Channing, and unanimously carried:

"The American Institute of Mining and Metallurgical Engineers assembled in its annual business meeting, approves the principles of the report of the Joint Conference Committee, and authorizes its representatives on that Committee to act with that Committee in calling a meeting for the organization of the representative engineering body provided for in said report."

## Technical Sessions

The Technical Sessions were very well attended and the papers presented were unusually interesting in value. They were as follows:

*Oil*

Monday, Feb. 16, 10:30 A.M., E. L. De Golyer presiding.

Oil Fields of Persia. By Campbell M. Hunter.

Rise and Decline in Production of Petroleum in Ohio and Indiana By J A. Bownocker.

Petroleum Resources of Kansas. By Raymond C. Moore.

Role of Bed-rock in the Distribution of Hydrocarbons. By M M. Monte-Flores

Water Displacement in Oil and Gas Sands. By Roswell H. Johnson

Tuesday, Feb. 17, 10.30 A.M., R. H. Johnson presiding.

A Foreign Oil Supply for the United States. By George Otis Smith.

Petroleum Resources of Great Britain. By A. C. Veatch.

Genetic Problems Affecting the Search for New Oil Regions. By David White.

Geologic Distillation of Petroleum By Bailey Willis.

Tuesday, Feb. 17, 2 P.M., Ralph Arnold presiding.

Variation in Decline Curves of Various Oil Pools. By Roswell H Johnson.

Appraisal of Oil Properties. By Earl Oliver.

International Aspects of the Petroleum Industry. By Van H. Manning.

The Composition of Petroleum and its Relation to Industrial Use By C F. Mabery.

Geology of the Cement Oil Fields. By F. G. Clapp.

*Oil and Coal*

Monday, Feb. 16, 2 P.M., E. L. De Golyer presiding

Drilling and Production Technique in the Baku Oil Fields. By Arthur Knapp.

Rock Classification from the Oil-driller's Standpoint By Arthur Knapp.

Oil Fields of Kentucky and Tennessee. By L. C. Glenn.

Petroleum in the Philippines. By W. D. Smith.

A Resume of the Pennsylvania-New York Oil Field By R H Johnson and Stirling Huntley.

Petroleum in the Argentine Republic. By Stanley C. Herold.

Low-temperature Carbonization of Coal By S. W. Parr and T E. Layng.

Demonstration Coal Mines By J J. Rutledge

Discussion on Anthracite Mining Costs By E. W. Parker (of paper by R. V. Norris, presented at the last February meeting )

*Non-ferrous Metallurgy, Institute of Metals Division*

Tuesday, Feb. 17, 10.30 A.M., W. M. Corse presiding.

Intercrystalline Brittleness of Lead. By H. S. Rawdon.

A Peculiar Type of Intercrystalline Brittleness of Copper By Henry S. Rawdon and S. C. Langdon.

Relationship of Physical and Chemical Properties of Copper. By Frank L. Antisell.

*Milling and Smelting*

Tuesday, Feb. 17, 2 P. M., E. P. Mathewson presiding.

Volatilization in Assaying. By Frederic P. Dewey.

Reverberatory Furnace for Treating Converter Slag at Anaconda. By Frederick Laist and H. J. Maguire.

Coal-pulverizing Plant at Nevada Consolidated Copper Smelter. By R. E. H Pomeroy.

Milling Plant of the Alaska Gastineau Mining Co. By E. V. Daveler.

*Geology and Mining*

Wednesday, Feb. 18, 10 A.M., L. C. Graton presiding.

Unwatering the Tiro General Mine by Air-lift. By S. F. Shaw.
Earth and Rock Pressures. By H. C Moulton Illustrated by lantern slides.
Rapid Formation of Lead Ore. By H. A Wheeler.
Ore Deposits in the Mogollon District, N M. By David B. Scott.
Examination of Ores and Metals in Polarized Light By Fred. E. Wright.

*Iron and Steel*

Wednesday, Feb. 18, 10 A.M., J. W. Richards presiding.

Blast-furnace Flue Dust. By R. W. H. Atcherson.
Manufacture of Semi-steel for Shells. By Frank E. Hall.
An Experiment in One-piece Gun Construction. By P. W. Bridgman.
Tensile Properties of Boiler Plate at Elevated Temperatures. By H J French.
Graphitization of White Cast Iron. By R. S. Archer.
Application of the Microscope to the Malleable-iron Industry. By Enrique Touceda.

*Iron, Steel, and Ferrous Alloys*

Wednesday, Feb. 18, 2 P M., H. D. Hibbard presiding.

The Coefficient of Expansion of Alloy Steels. By John A. Mathews
Critical Ranges of Some Commercial Nickel Steels. By Howard Scott
Forgeability of Iron-nickel Alloys. By T. D. Yensen.
Microstructure of Iron and Mild Steel at High Temperatures. By H. S. Rawdon and Howard Scott.
Physical Changes in Iron and Steel below the Thermal Critical Range. By Zay Jeffries

*Industrial Organization*

Monday, Feb. 16, 10:30 A.M., T. T. Read presiding.

Report of Progress in Americanization, 1919. By E. E. Bach.
Report of Progress, Cripples in Industry, 1919. By Burr A. Robinson.
Report of Progress, Mining Education, 1919. By E. A. Holbrook.
Report of Progress, Employment, 1919. By W. A. Smith.

Monday, Feb. 16, 2 P.M., Arthur S. Dwight presiding.

Report of Progress in Housing, 1919. By Will L. Clark.
Report of Progress, Mental Hygiene in Industry, 1919. By T. T. Read
Report of Progress, Prevention of Illness, 1919. By H. I. Young.
Report of Progress, Accident Prevention, 1919. By Francis Sinn.
Contract Wage System for Mines. By A. K. Knickerbocker.

*Open Forum on Federal Mine Taxation*

Tuesday, Feb. 17, 10:30 A. M., J. Parke Channing presiding.

*Stabilization of the Bituminous Coal Industry*

The most important sessions were those on the stabilization of the bituminous coal industry, which was a plan evolved by retiring President Horace V. Winchell and President-elect Herbert Hoover for the participation of engineers, especially mining engineers, and all others who could add a ray of light to the problem under a constructive program for the better working of the bituminous coal industry, if not for the complete solution of the bituminous problem.

President Winchell presided at the first session and Herbert Hoover, the first speaker, presented the problem and outlined the scope of the plan in mind which comprises: The causes of intermittency; how and when the irregularities occur; the actual number of days worked by the men at the mine during each season of the year; a possible alteration in the wage basis; actual wages received by workers during each season of the year; the question of storage in each of its phases, and as to all possible places for storage, such as at the mine, at the industrial plants, etc.

The following sessions were held and an open forum after each session gave everyone concerned an opportunity to add to the discussion.

Tuesday, Feb. 17, 2 P.M., Horace V. Winchell presiding.

A Definite Program of Study and Work for the A. I. M. E. and a Constructive Plan for the Better Working of the Bituminous Coal Industry. By Herbert Hoover.

Problems of the Coal Industry. By Van H. Manning. An introductory statement outlining the problem under discussion and indicating its relative significance as compared to other problems of the coal industry.

Fluctuations in Coal Production—Their Extent and Causes. By George Otis Smith and F. G. Tryon. A statistical analysis of the rate of output of a period of years, indicating the relative effect of shortage of transportation, shortage of labor, lack of market, and other factors in producing intermittency in the operation of coal mines.

Wednesday, Feb. 18, 10 A.M., Edwin Ludlow presiding.

Storage of Bituminous Coal. By H. H. Stoek. Storage of bituminous coal; (*a*) at the point of production; (*b*) at centers of distribution; (*c*) by the consumer. Capital and operating cost of storage; breakage; loss in weight; loss in heating value; spontaneous combustion.

Transportation as a Factor in Irregularity of Coal-mine Operation. By S. L. Yerkes. Exact data as to the real effect transportation facilities have on coal production; use of cars for storage; effect of more equipment and its cost to the railroads; effect of lower rates in spring and summer, reduction of cross hauling; long hauls by railroads of their own coal.

Wednesday, Feb. 18, 2 P.M., J. V. W. Reynders presiding.

Stabilizing of Bituminous Coal Industry. By Eugene McAuliffe. Variations in the market demand for coal, possibilities of a sliding scale of prices toward producing regularity of buying. Effect of varying freight rates on the market. Relation between the total demand and the productive capacity. How to provide markets for present productive capacity.

Conservation. By Edwin Ludlow. Conservation was considered in its three phases as follows (a) In the methods of mining; (b) in the utilization of what are now byproducts at the mines and losses; (c) by proper methods of firing in the boiler house.

### Annual Meeting of Woman's Auxiliary

The first session of the Woman's Auxiliary was held on Tuesday, Feb. 17, Mrs. James F. Kemp, the President, presiding. The report of the Tellers showed that the officers elected for the coming year are as follows:

President, Mrs. Arthur S. Dwight.
First Vice-president, Mrs. H. P. Henderson.
Second Vice-president, Mrs. Sidney J. Jennings.
Third Vice-president, Mrs. Mark L. Requa.
Secretary, Mrs. John V. W. Reynders.
Treasurer, Mrs. Karl Eilers.

The report of the Tellers on the proposed changes in the Constitution, was as follows:

| | |
|---|---|
| For change of name | 95 Yes<br>1 No |
| Ballots cast for the proposed election of officers of Central Council to hold office for two years instead of one | 95 Yes<br>1 No |
| Ballots cast in favor of fiscal year beginning March first instead of January first | 89 Yes<br>3 No |

Respectfully submitted,
Madeline P. Stone,
V. Grace Barbour,
*Tellers.*

## The Only Way Out

### Inaugural Address of Herbert Hoover, President A. I. M. E.

I have been greatly honored as your unanimous choice for President of this Institute, with which I have been associated during my entire professional life. It is customary for your new President, on these occasions, to make some observation on matters of general interest from the engineer's standpoint.

The profession of engineering in the United States comprises not alone scientific advisers on industry but, in great majority, is comprised of the men in administrative positions. In such positions, they stand midway between capital and labor. The character of your training and experience leads you to exact and quantitative thought. This basis of training in a great group of Americans furnished a wonderful recruiting ground for service in these last years of tribulation. Many thousands of engi-

neers were called into the army, the navy, and civilian service for the government. Thousands of high offices were discharged by them with credit to the profession and the nation.

We have in this country, probably, one hundred thousand professional engineers. The events of the past few years have greatly stirred their interest in national problems. This has taken practical form in the maintenance of joint committees for discussion of these problems and support to a free advisory bureau in Washington. The engineers want nothing for themselves from Congress. They want efficiency in government, and you contribute to the maintenance of this bureau out of sheer idealism. This organization for consideration of national problems has had many subjects before it, and I propose to touch on some of them this evening.

Even more than before is there necessity for your continued interest in this vast complex of problems that must be met by our government. We are faced with a new orientation of our country to world problems. We face a Europe still at war; still amid social revolutions; some of its peoples are still slacking on production; millions are starving; therefore the safety of its civilization is still hanging by a slender thread. Every wind that blows carries to our shores an infection of social disease from this great ferment; every convulsion there has an economic reaction upon our own people. If we needed further proof of the interdependence of the world, we have it today in the practical blockade of our export market. The world is asking for us to ratify long delayed peace in the hope that such confidence will be restored as will enable her to reconstruct her economic life. We are contemplating the maintenance of an enlarged army and navy in preparedness for further upheavals in the world and are failing to provide some insurance against war by a league to promote peace.

Out of the strain of war, weaknesses have become even more evident in our administrative organization, in our legislative machinery. Our federal government is overcentralized, for we have upon the hands of our government enormous industrial activities that have yet to be demobilized. We are swamped with debt and burdened with taxation. Credit is woefully inflated, speculation and waste are rampant. Our own productivity is decreasing. Our industrial population is crying for remedies for the increasing cost of living and are aspiring to better conditions of life and labor. But, beyond all this, great hopes and aspirations are abroad; great moral and social forces have been stimulated by the war and will not be quieted by the ratification of peace. These are but some part of the problems with which we must deal. I have no fear that our people will not find solutions; but progress is sometimes like the old-fashioned rail fence—some rails are mis-shapen and all seem to point the wrong way; but in the end, the fence progresses.

Your committees, jointly with those of other engineering societies, have had before them and have expressed their views in many matters concerning the handling of the railways, shipping, reorganization of the government engineering work, national budget, and other practical items.

### Demobilization of Railways

The war nationalization of railways and shipping are our greatest problems in government control awaiting demobilization. There are many fundamental objections to the continuation of these experiments in socialism necessitated by the war. They lie chiefly in their destruction of initiative in our people and the dangers of political domination that can grow from governmental operation. Beyond this, the engineers will hold that the successful conduct of great industries is to a transcendant degree dependent on the personal abilities and character of their employees and staff. No scheme of political appointment has yet been devised that will replace competition in its selection of ability and character. Both shipping and railways have today the advantage of many skilled personnel, sifted out in the hard school of competition; and even then the government operation of these enterprises is not proving satisfactory. Therefore, the ultimate inefficiency that would arise from the deadening paralysis of bureaucracy has not had full opportunity for development. Already we can show that no government under pressure of the ever-present political or sectional interests can properly conduct the risks of extension and improvement or can be free from local pressure to conduct unwarranted services in industrial enterprise. On the other hand, our people have long since recognized that we cannot turn monopoly to unrestrained operation for profit nor that the human rights of employees can ever be dominated by dividends.

Our business is handicapped on every side by the failure of our transportation facilities to grow with the country. It is useless to talk about increasing production to meet an increased standard of living in an increasing population without a greatly increased transport equipment. Moreover, there are great social problems underlying our transport system; today their contraction is forcing a congestion of our population around the great cities with all that these over-swollen settlements import. Even such great disturbances as the coal strike have a minor root in our inadequate transportation facilities and their responsibility for intermittent operation of the mines. We are all hoping that Congress will find a solution to this problem that will be an advanced step toward the combined stimulation of the initiative of the owners, the efficiency of operation, the enlistment of the good-will of the employees, and the protection of the public. The problem is easy to state. Its solution is almost overwhelming in complexity. It must develop with experience, step by step, toward a real working partnership of its three elements.

The return of the railways to the owners places predominant private operation upon its final trial. If instant energy, courage, and large vision in the owners should prove lacking in meeting the immediate situation, we will be faced with a reaction that will drive the country to some other form of control. Energetic enlargement of equipment, better service, cooperation with employees, and the least possible advance in rates together with freedom from political interest will be the scales upon which the public will weigh these results.

## National Merchant Marine not a Monopoly

Important phases of our shipping problem that have come before you should receive wider discussion by the country. As the result of war pressure, we will spend over $2,800,000,000 in the completion of a fleet of 1900 ships of a total of 11,000,000 tons—nearly one-quarter of the world's cargo shipping. We are proud of this great expansion of our marine and we wish to retain it under the American flag. Our shipping problem has one large point of departure from the railway problem, for there is no element of natural monopoly. Anyone with a water-tight vehicle can enter upon the seas today, and our government is now engaged upon the conduct of a nationalized industry in competition with our own people and all the world besides. While in the railways government inefficiency could be passed on to the consumer, on the seas we will sooner or later find it translated to the national treasury.

Until the present time, there has been a shortage in the world's shipping, but this is being rapidly overtaken and we shall soon be met with fierce competition of private industry. If the government continues in the shipping business, we shall be disappointed from the point of view of profits, for we shall be faced with the ability of private enterprise to make profits from the margins of higher cost of government operation alone.

Aside from those losses inherent in bureaucracy and political pressure, there are others special to this case. The largest successfully managed cargo fleet in the world comprises about 120 ships and we are attempting to manage 1900 ships at the hands of a government bureau. In normal times the question of profit or loss in a ship is measured by a few hundred tons of coal wasted, a little extravagance in repairs, or the four or five days on a round trip. Beyond this, private shipping has a free hand to set up such give-and-take relationships with merchants all over the world as will provide sufficient cargo for all legs of a voyage. These arrangements of coöperation cannot be created by government employees without charge or danger of favoritism. Lest fault be found, our government officials are unable to enter upon the detailed higgling in fixing rates required by every cargo and charter; therefore, they must take refuge in

rigid regulations and in fixed rates. As a result, their competitors underbid by the smallest margins necessary to get the cargoes. The effect of our large fleet in the world's markets is thus to hold up rates, for so long as this great fleet on one hand holds a fixed rate others will only barely underbid. If we hold up rates, an increasing number of our ships will be idle as the private fleet grows; on the other hand, if we reduce rates we shall be underbid until the government margin of larger operation cost causes us to lose money.

We shall yet be faced with the question of demobilizing a considerable part of this fleet into private hands, or frankly acknowledging that we operate it for other reasons than interest on our investment. In this whole problem, there are the most difficult considerations requiring the best business thought in the country. In the first instance, our national progress requires that we retain a large fleet under our flag to protect our national commercial expansion overseas. Second, we may find it desirable to hold a considerable government fleet to build up trade routes in expansion of our trade, even at some loss in operation. Third, in order to create this fleet, we have built up an enormous ship-building industry. Fifty per cent. of the capacity of our shipyards will more than provide any necessary construction for American account; therefore, there is a need of obtaining foreign orders, or the reduction of capacity, or both. I believe, with most engineers, that, with our skill in repetition manufacture, we can compete with any shipbuilders in the world and maintain our American wage standards; but this repetition manufacture implies a constant flow of orders. It would seem highly desirable, in order to maintain the most efficient yards until they can establish themselves firmly in the world's industrial fabric, that the government should continue to let some ship construction contracts to the lowest bidders. These contracts should supplement private building in such a way as to maintain the continuous operation of the most economical yards and the steady employment of our large number of skilled workers engaged therein.

When we consider giving orders for new ships, we must at the same time consider the sale of ships, as we cannot go on increasing this fleet. When we consider sale, we are confronted with the fact that our present ships were built under expensive conditions of war, costing from three to four times per ton the pre-war amount, and that already any merchant, subject to the long time of delivery, can build a ship for 75 per cent. of their cost. It would seem to be good national policy to sell ships today for the price we can contract for delivery a year or two hence, thus making the government a reservoir for continuous construction. We could thus stabilize the building industry to some degree and also bring the American-owned fleet into better balance, if, as the government sold three or four emergency-constructed cargo vessels, it gave an order for

one ship of a better and faster type. This would make reduction in our shipbuilding steadier and would give the country the type of ships we need.

## National Department of Public Works

Our joint engineering committees have examined, with a great deal of care, into the organization of and expenditure on public works and technical services. These committees have consistently and strongly urged the appalling inefficiency in the government organization of these matters. They report that the annual expenditure on such works and services now amounts to over $250,000,000 per annum, and that they are carried out today in nine governmental departments. They report a great waste by lack of a national policy of coordination, in overlapping with different departments, in competition with each other in the purchase of supplies and materials and the support of many engineering staffs. They recommend the solution that almost every other civilized government has long since adopted; that is, the coördination of these measures into one department under which all such undertakings should be conducted and controlled. As a measure practical to our government, they have advocated that all such bureaus should be transferred to the Interior Department and that all the bureaus not relating to these matters should be transferred from the Interior to other Departments. The committees conclude that no properly organized and directed saving in public works can be made until such a regrouping and consolidation is carried out, and that all of the cheese-paring that normally goes on in the honest effort of Congressional committees to control departmental expenditure is but a tithe of that which could be effected if there were some concentration of administration along the lines long since demonstrated as necessary to the success of private business.

## Need of a National Budget

Another matter of government organization to which our engineers have given adhesion is in the matter of the national budget. To minds charged with the primary necessity of advance planning, coördination, provision of synchronizing parts in organization, the whole notion of our hit-or-miss system is repugnant. A budget system is not the remedy for all administrative ills; it provides a basis of organization that at least does not paralyze administrative efficiency as our system does today. Through it, the coordination of expenditure in government departments, the prevention of waste and overlapping in government bureaus, the exposure of the pork barrel, and the balancing of the relative importance of different national activities in the allocation of our national income can all be greatly promoted. Legislation would also be expedited. No

budget that does not cover all government expenditure is worth enactment. Furthermore, without such reorganization as the grouping of construction departments, the proper formulation of a budget would be hopeless. The budget system in some form is so universal in civilized governments and in competently conducted business enterprise, and has been adopted in thirty of our states, that its absence in our federal government is most extraordinary. It is, however, but a further testimony that it is always a far cry of our citizens from the efficiency in their business to interest in the efficiency of their government.

## Changes in Science of Economics

Another great national problem to which every engineer in the United States is giving earnest thought, and with which he comes in daily contact, is that of the relationship of employer and employee in industry. In this, as in many other national problems today, we are faced with a realization that the science of economics has altered from a science of wealth to a science of human relationships to wealth. We have gone on for many years throwing the greatest of our ingenuity and ability into the improvement of processes and tools of production. We have, until recently, greatly neglected the human factor, which is so large an element in our very productivity. The development of vast repetition in the process of industry has deadened the sense of craftsmanship and the great extension of industry has divorced the employer and his employee from that contact that carried responsibility for the human problem. This neglect of the human factor has accumulated much of the discontent and unrest throughout our great industrial populations and has reacted in a decrease of production. Yet our very standards of living are dependent on a maximum productivity up to the total necessities of our population.

Another economic result is, or will be, a repercussion upon the fundamental industry of the United States, that is, agriculture. The farmer will be unable to maintain his production in the face of a constant increase in the cost of his supplies and labor through shrinkage in productivity in other industries. The penalty of this disparity of effort comes mainly out of the farmer's own earnings.

## The One Remedy

I am daily impressed with the fact that there is but one way out, and that is to reëstablish through organized representation that personal coöperation between employer and employee in production that was a binding force when our industries were smaller of unit and of less specialization. Through this, the sense of craftsmanship and the interest in production can be recreated and the proper establishment of conditions

of labor and its participation in a more skilled administration can be worked out. The attitude of refusal to participate in collective bargaining with representatives of the employees' own choosing is the negation of this bridge to better relationship. On the other hand, a complete sense of obligation to bargains entered upon is fundamental to the process itself. The interests of employee and employer are not necessarily antagonistic; they have a good common ground of mutuality and if we could secure emphasis upon these common interests we would greatly mitigate conflict. Our government can stimulate these forces, but the new relationship of employer and employee must be a matter of deliberate organization within industry itself. I am convinced that the vast majority of American labor fundamentally wishes to coöperate in production and that this basis of good-will can be organized and the vitality of production recreated.

Many of the questions of this industrial relationship involve large engineering problems, as an instance of which I know of no better example than the issue you plan for discussion tomorrow in connection with the soft-coal industry. Broadly, here is an industry functioning badly from an engineering, and consequently from an economic and human, standpoint. Owing to the intermittency of production, seasonal and local, this industry has been equipped to a peak load of 25 or 30 per cent. over the average load. It has been provided with a 25 or 30 per cent. larger labor complement than it would require if continuous operation could be brought about. I hope your discussion will throw some light on the possibilities of remedy. There lies in this intermittency not only a long train of human misery through intermittent employment, but the economic loss to the community of over a hundred thousand workers who could be applied to other production, and the cost of coal could be decreased to the consumer. This intermittency lies at the root of the last strike in the attempt of the employees to secure an equal division among themselves of this partial employment at a wage that could meet their view of a living return on full employment.

These are but a few of the problems that confront us. But in formulating of measures of solution, we need a constant adherence to national ideals and our own social philosophy.

In the discussion of these ideals and this social philosophy, we hear much of radicalism and of reaction. They are, in fact, not an academic state of mind but realize into real groups and real forces influencing the solution of economic problems in this community. In their present-day practical aspects, they represent, on one hand, roughly various degrees of socialism, who would directly or indirectly undermine the principle of private property and personal initiative; and, on the other hand, those exponents who in various degrees desire to dominate the community for profit and privilege. They both represent attempts to introduce or

preserve class privilege, either a moneyed or a bureaucratic aristocracy. We have, however, in American democracy, an ideal and a social philosophy that sympathizes neither with radicalism nor reaction as they are manifested today.

## Social Philosophy of American People

For generations, the American people have been steadily developing a social philosophy as part of their democracy—and in these ideals, it differs from all other democracies. This philosophy has stood this period of test in the fire of common sense; it is, in substance, that there should be an equality of opportunity—an equal chance—to every citizen. This view that every individual should, within his lifetime, not be handicapped in securing that particular niche in the community to which his abilities and character entitle him is itself the negation of class. Human beings are not equal in those qualities. But a society that is based upon a constant flux of individuals in the community, upon the basis of ability and character, is a moving virile mass; it is not a stratification of classes. Its inspiration is individual, initiative. Its stimulus is competition. Its safeguard is education. Its greatest mentor is free speech and voluntary organization for public good. Its expression in legislation is the common sense and common will of the majority. It is the essence of this democracy that progress of the mass must arise from progress of the individual. It does not permit the presence in the community of those who would not give full meed of service.

Its conception of the state is one that, representative of all the citizens, will in the region of economic activities apply itself mainly to the stimulation of knowledge, the undertaking only of works beyond the initiative of the individual or group, the prevention of economic domination of the few over the many, and the least entrance into commerce that government functions necessitate.

The method and measures by which we solve this accumulation of great problems will depend on which of these three conceptions will reach the ascendancy among our people.

If we cling to our national ideals, it will mean the final isolation and the political abandonment of the minor groups who hope for domination of the government, either by "interests" or by radical social theories through the control of our political machinery. I sometimes feel that lawful radicalism in politics is less dangerous than reaction, for radicalism is blatant and displays itself in the open; unlawful radicalism can be handled by the police. Reaction too often fools the people through subtile channels of obstruction and progressive platitudes. There is little danger of radicalism ever controlling a country with so large a farmer population, except in one contingency. That contingency is

from a reflex of continued attempt to control this country by the "interests" and other forms of our domestic reactionaries.

The mighty upheaval following the world war has created turmoil and confusion in our own country no less than in all other lands. If America is to contribute to the advance of civilization, it must first solve its own problems, must first secure and maintain its own strength. The kind of problems that present themselves are more predominantly economic—national as well as international—than at any period in our history. They require quantitative and prospective thinking and a sense of organization. These are the sort of problems that your profession deals with as its daily toil. You have an obligation to continue the fine service you have initiated and to give it your united skill. In their solution, in Lowell's phrase, we cannot "unlock the portals of the Future with the blood-rusty keys of the Past"

## Report of President for the Year 1919

I HAVE the honor to present the following report of the President for the year 1919. In order that this report may be printed and distributed to the members in advance of the February meeting, it is necessary that it end with the calendar year 1919. Any activities during the first six weeks of 1920 will be covered in the Secretary's report for the year 1920.

### Visits to Members

Your President has visited each of the Local Sections of the Institute, with the exception of the ones at Washington, D. C., Tulsa, Okla., and the Nevada Section. These three sections have not arranged meetings at which the President could be present. In addition, your President, accompanied by President Jennings and Secretary Stoughton, in the year 1918, attended meetings of members in Minneapolis and Duluth, Minn., and Houghton, Mich. A dinner and meeting of the members in Pittsburgh, Pa., was also attended, in company with the Secretary, and your President also attended the meeting of Canadian Mining Institute in Montreal, in March, 1919.

### Revamping of Committees

The work of the Institute is carried on by means of Local Sections and committees which, including the six standing committees of the Board of Directors, number nearly thirty technical committees, whose purpose it is to enrich the publications of the Institute and keep them well proportioned and well balanced. There are also nearly a score of miscellaneous committees, many of which are joint committees with other engineering societies. It has been some years since these committees have been thoroughly revised, so the work of revising the membership was

undertaken at the beginning of my term of office, with what result in increased activities along lines of welfare, public policy, and technology, the report of the Secretary will show.

### Increase in Membership

The report of the Committee on Membership will show that the number of members has been greatly increased during the year. This tells only a part of the story, however, since active work along this line did not commence until the month of April, under the direct supervision of Assistant-secretary P. E. Barbour. The gratifying result of this diligent and well-organized effort has been that the applications for membership are 75 per cent. in excess of the largest previous year.

### Committee on Mine Taxation

During the convention of the Institute at Chicago, a request was received from the United States Department of the Treasury for the appointment of a committee to assist the Treasury Department in interpreting the income tax as applied to the wasting industry of mining. A committee of twenty-five members was appointed by telegraph. This committee was composed of some of the most prominent and busiest of the mining engineers of the country. The response to this call is shown by the fact that seventeen immediately responded by wire accepting the appointment. Almost the full committee of twenty-five members, many of them accompanied by their legal advisers, attended the first conference at Washington, where special committees were appointed, and subsequent meetings held at Atlantic City and at New York. Within two months of the original call, the recommendations of the Committee had been made to the Treasury Department, and published in Mining and Metallurgy for December.

### Hoover Dinner

On Sept. 16, 1919, the members of the Institute joined in a complimentary dinner to Herbert Hoover upon his return from Europe in appreciation of his unselfish services in the cause of mankind during the years 1914 to 1919 inclusive. The full report of this dinner is given in the Institute's magazine for the month of October, 1919. Your President traveled from Utah to New York in order to attend the dinner, at which there were 1350 guests.

### Notable Increase in Activities Concerned with Public Affairs and the Welfare of Engineers

The constitution and the charter of the Institute were amended at a special meeting of members held on June 27, in order to permit the Insti-

tute as an organized body to enter more actively into the public affairs of the community and the nation. Through our own Institute, as well as in coöperation with other engineering bodies, important work has been done, especially in connection with the employment of engineers, and more particularly the re-employment of engineers returning from service with the American Expeditionary Force; in connection with the establishment of a national Department of Public Works in place of the present Department of the Interior; in respect to the existing licensing or registering of engineers, etc. By authority of the Board of Directors a committee on the development of the activities of the Institute was appointed, which, in turn, appointed representatives on a joint committee of development with the American Society of Civil Engineers, American Society of Mechanical Engineers, and American Institute of Electrical Engineers. This joint committee has held many long and active meetings in various parts of the country, and has united in a comprehensive report which purposes to effect an improvement in the organization of all engineers for civic and welfare work.

### Local Organizations of Engineers

Your President's visit to different parts of the country has confirmed his belief in the importance of local organizations of engineers at all industrial centers, corresponding in general to the state boundaries of the country, in order that, when necessary, the united influence of the engineers of any state may be brought to bear upon the state or national legislatures thereof for the purpose of urging reforms, or combating new legislation, upon which engineers might be expected to have an opinion of value to the community, or which threatens to do injury to engineers or the profession which they represent.

### Acknowledgments to Directors and Secretaries

Your retiring President desires to express in the most cordial manner his sense of indebtedness to the Directors for their unceasing support and tireless efforts in carrying forward the work of the year and his sincere appreciation of their generous spirit of welcome coöperation and self-sacrificing devotion to the interests of the Institute. Fortunate indeed is an organization which contains so capable and devoted a body of men. It is also a duty and a pleasure to call your attention to the invaluable services rendered by the Secretary, Dr. Bradley Stoughton, and the Assistant Secretary, Major Percy E. Barbour. Whatever success has attended the President's labors for the Institute during 1919 is due in large measure to their intelligent and tireless labors.

There is reason for congratulation in the evident increase of coöperative effort on the part of our members throughout the country; and we

feel confident that under the powers granted by our new charter and the awakening sense of civic obligation our influence will be apparent in much wider fields of public activity during the coming year

Respectfully submitted,

HORACE V. WINCHELL, *President.*

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## Report of Secretary

IN THE President's report and the reports of the standing committees are discussed more in detail the several activities of the Institute, including: Visits of the President and Secretary to Local Sections and to other gatherings of members; the revision of the Institute committees; the increase in membership; coöperation with the U. S. Treasury Department in respect to interpreting the Income Tax law as applied to mining corporations; the dinner to Herbert Hoover; the Library; the publications; etc.

### IMPORTANT INCREASE IN RESEARCH ACTIVITIES

The National Research Council was formed at the request of the President of the United States to assist the Government in preparing for, and prosecuting, the war. It was supported during its infant stages by the funds of Engineering Foundation, a creature of the American Institute of Mining and Metallurgical Engineers, American Society of Civil Engineers, American Society of Mechanical Engineers, and American Institute of Electrical Engineers. During its life it has at all times been actively supported by the Institute officially, and by its members. In this way, the Institute has contributed to the promotion of engineering research in its own field. The relation has become even closer during the year 1919 by the transfer of Engineering Division of National Research Council to the Engineering Societies' Building in New York. It is believed that the Institute can do more to encourage and support research by coöperating with this national body than by any independent activities.

### COÖPERATION WITH FRENCH ENGINEERS

Shortly after the Armistice was signed, an invitation was received from the Société des Ingénieurs Civils de France for a committee of American engineers to visit France and confer with them. The resulting coöperation, which is mentioned in the last Secretary's report, was so successful that the French society requested the maintenance of a permanent committee, which has now been established, with two repre-

sentatives of each of the American engineering societies mentioned in the previous paragraph. A return visit of French engineers is promised for the near future

## National Department of Public Works

The achievements of Engineering Council will be communicated to the members through its own report. Its chief accomplishment has been the formation of the Conference on a National Department of Public Works, in which some 74 societies are participating, representing approximately 105,000 engineers. This subject is familiar to Institute members through the several reports in Mining and Metallurgy

## Affiliated Student Society Changes

Owing to the interruption of regular college work during the activities of the Students' Army Training Corps, our student societies were mostly dormant during the year 1918. An effort was, therefore, made to get into communication with all of them during the first part of 1919, with the result that almost all were found ready to begin active work again upon the resumption of the usual college training. Two new societies have been added, namely: The Mining Society of the Montana School of Mines and The Cross-Hammer Mining Society of the Oklahoma School of Mines.

## Four Meetings in 1919

The Annual Meeting in New York is described in Volume XLI, p. xi. It was characterized chiefly for the coöperation with the Canadian Mining Institute, as well as the American Institute of Electrical Engineers, and for the increase in social activities and good spirits naturally consequent on the cessation of hostilities. It was marked also by the presentation of six papers contributed by National Research Council, and by eleven papers of the Institute of Metals Division, this being the first meeting at which the A.I.M.E. and the former American Institute of Metals joined forces. It was also the first winter meeting held by the A.I.M. and was pronounced by its officers an undoubted success. The total attendance was 1062. (Volume LXII., p. xvii.) The Chicago Meeting was the largest ever held outside of New York City, the registration being 850, which did not include many members who attended but did not register. It was especially a record meeting in respect to the number of technical papers presented, and was unusually brilliant in its social aspects, particularly as to the smoker and banquet and the large number of ladies in attendance. The meeting of the Institute of Metals Division jointly with the American Foundrymen's Association,

in Philadelphia, Sept. 30 to Oct. 2, fully equalled the previous conventions of the same kind. On June 27, 1919, a special business meeting of members was held at headquarters for the purpose of voting on the proposed change in the Certificate of Incorporation, in order to make effective the letter ballot of members for the change in name of the Institute, and for greater participation in public affairs.

## THE INSTITUTE CHANGES ITS NAME

By a vote of 1565, members of the Institute voted in favor of changing its name to the "American Institute of Mining and Metallurgical Engineers." The initials remain "A.I.M.E." as before, by vote of the Board of Directors.

## GREATER PARTICIPATION IN PUBLIC AFFAIRS

Also, by an overwhelming majority the members voted, first by letter ballot and later by proxy at the special business meeting, to change the Certificate of Incorporation so as to remove any possible ambiguity as to the right of the Institute to engage in public work for the welfare of the profession or its members.

## ROBERT W. HUNT PRIZE ESTABLISHED

Upon the occasion of the eightieth birthday of Captain Robert W. Hunt, Dec. 9, 1918, his partners announced that they had given the funds for the establishment of the Robert W. Hunt Prize for the best paper on the subject of iron and steel presented to the Institute for publication. A special committee has been at work upon the rules for this prize, which will be awarded through the Institute's Iron and Steel Committee. Rules have been adopted and a medal is being designed by a distinguished artist, Mr. Emil Fuchs The rules of award and a reproduction of the medal will be published in due time.

## MORE ROOMS FOR OUT-OF-TOWN MEMBERS

On account of the large increase in the number of members who are using the headquarters during their stay in New York, it has been thought desirable to pay for and appropriate another room for this purpose, making three rooms now devoted to the use of the members.

## JAMES DOUGLAS TABLET

A bronze tablet in memory of Dr. James Douglas, the Institute's generous benefactor, has been erected at headquarters.

## Death of Several Prominent Members

The year 1919 was a most unfortunate one in respect to the loss of prominent members by death. Andrew Carnegie, honorary member and donor of Engineering Societies' Building, J. E. Johnson, Jr., member of the Board of Directors and Vice-Chairman of the Iron and Steel Committee, Henry C. Frick, and Samuel T. Wellman were among the number.

## Increase in Cost of Services to Members

The cost of every kind of material or labor entering into the services rendered members has increased an average of at least fifty per cent. in the past few years, without there being any increase in cost for membership. The annual dues of twelve dollars per year is the lowest, by three dollars or more, of any national engineering society which publishes a monthly magazine and annual volumes, while the A.I.M.E. issues more technical material than any engineering society. Paper has increased 100 per cent. and clerical labor the same. In addition to these increases in unit cost, the expenses have gone up because of the greatly added activities of the Institute in respect to public policy, welfare, and employment work.

## American Institute of Metals Tablet

A bronze tablet commemorative of the American Institute of Metals, which was affiliated with the A.I.M.E. in the year 1918, has been erected at headquarters in response to a request from the officers thereof.

## Employment Activities

The A.I.M.E. has long maintained an employment bureau for the purpose of bringing together mining and metallurgical employers and employees. With the object of securing greater efficiency and a larger volume of business, as well as decreased unit cost for publicity, the four engineering societies associated in the Engineering Societies' Building joined their employment bureaus under the direction of the four Secretaries. This move has been an unqualified success and has established what is believed to be the best present medium for bringing together engineers and their employers, and the one accomplishing the maximum results in this field.

## Expulsion of Several Members for Non-professional Conduct

It is unfortunate to have to mention the expulsion of members who are believed to be unworthy the honor or privilege of membership, but

as many members have been expelled this year—namely two—as have been ejected *in toto* during the past six years. If an engineer is unworthy because of his lack of integrity, it is better that he be expelled forthwith, and members are urged for the good of the profession to bring to the attention of the Directors any cases of unprofessional conduct on the part of A.I.M.E. members. We believe that the increase in number of cases reported this year is due to the establishment by the members of a higher qualification for membership, and therefore a greater appreciation of the honor and responsibilities of membership.

### Committee on Development

It would not be right to close this report without mentioning the devoted work of the Committee on Development of the Activities of the Institute, both in its study of the problems of the A.I.M.E. and its work with the corresponding committees of the other engineering societies. The labor taxed the time and the private interests of the members and the grateful thanks of the Institute are due especially to Chairman J. W. Richards and Acting Chairman J. V. W. Reynders.

### Assistant-secretary Percy E. Barbour

The Secretary also wishes to record his appreciation of the work of Assistant-secretary Barbour, who has completed his first year's service in this capacity. He has rendered faithful and efficient service, especially in connection with the improvement in the monthly magazine (which has won spontaneous favorable comment from many members), in the work on increase of membership, and in promoting several matters which make the Institute more valuable to its members.

### Coöperation with Other Bodies

*Canadian Mining Institute.*—At our February Meeting many members of Canadian Mining Institute were welcome guests, and several of our members gladly availed themselves of the invitation to attend the meeting of our sister society in March. A committee on coöperation between the two societies is now maintained, with three members of each body. The members of C. M. I. have been invited to our meeting in February, 1920

*Local Organizations at Engineering Centers.*—Local Sections of the A. I. M. E. or groups of members in engineering localities are actively coöperating in organizations of engineers for the carrying on of public welfare and civic work. Some of these are at St. Louis, Colorado, Minnesota, Spokane, Seattle, Alabama, Georgia, San Francisco, Los Angeles, etc., while additional organizations are being formed at Chicago, Pittsburgh, Washington, D. C., etc.

*Government Departments.*—Our coöperation with the various Government Bureaus and Divisions continues unabated.

*Other Societies in the United Engineering Society.*—The A. I. M. E. continues its coöperation and interest in the several joint activities of the engineering societies with headquarters in New York. Added interest has been taken in the American Engineering Standards Committee and American Welding Society.

*Miscellaneous.*—Many other forms of coöperation have been followed by the Institute during the year 1919, but it is impossible to mention all.

Respectfully submitted,

BRADLEY STOUGHTON,

*Secretary.*

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## Report of Treasurer and Finance Committee

WE HAVE audited the books and accounts of the American Institute of Mining and Metallurgical Engineers, and prepared therefrom a statement of cash receipts and disbursements for the year ended December 31, 1919, and a balance sheet at the latter date. A summary of the cash receipts and disbursements follows:

| | |
|---|---|
| January 1, 1919—Balance in banks and on hand | $ 8,258.37 |
| December 31, 1919—Receipts for the year | 241,44 .25 |
| | $249,701.62 |
| *Deduct:* Disbursements for the year | 246,432.09 |
| December 31,1919—Balance on hand | $ 3,269.53 |

Distributed as follows:

| | |
|---|---|
| National Bank of Commerce | $ 756.47 |
| Brooklyn Trust Co. | 485.15 |
| Fifth Avenue Bank | 480.19 |
| Fifth Avenue Bank (special account) | 446.39 |
| Franklin Trust Co. | 496.17 |
| Bankers Trust Co. | 57.66 |
| Petty cash in office | 500.00 |
| | $3,222 03 |
| Coupons due but not deposited | 47.50 |
| | $3,269 53 |

During the year investment was made in $1000 U. S. Government 4¾ per cent. Victory Loan Bonds due in 1922–1923. The $100 U. S. Government 4¼ per cent. Third Liberty Loan Bonds purchased for employees was taken over by the Institute. This total of $1100 in

U. S Government Bonds is carried as an investment on account of the Life Membership Fund.

During the year the Institute received $100,000 as a bequest under the will of James Douglas for the purpose of maintaining its scientific library. This fund has been invested in the following bonds, in accordance with a resolution of the Board of Directors:

| | |
|---|---|
| $23,000 Idaho Power Co first 5 per cent 1947 | $20,716 81 |
| 20,000 American Telegraph & Telephone Co Collateral Trust 5 per cent 1946. | 18,384 72 |
| 20,000 United Kingdom Great Britain & Ireland 5½ per cent 1937 | 20,409 85 |
| 23,000 Utah Power & Light Co first 5 per cent 1944 | 21,185 55 |
| 20,000 Montreal Tramways first and refunding 5 per cent. 1941 | 17,620 83 |
| | $98,317 76 |

These securities are deposited with the Bankers Trust Co., who, under the above resolution, are to collect the income and credit the account of the American Institute of Mining and Metallurgical Engineers.

The income from the invested funds has not yet been paid over to the United Engineering Society.

At the close of the period covered by our audit there was an uninvested balance in the fund of $1682.24 and a credit balance in the income account of $1875.42. The total cash in hand belonging to the fund, principal and interest, was $57.66, the general funds of the Institute being in debt to the fund to the extent of $3500 which has been replaced. We would suggest that the income be paid over to the United Engineering Society at stated periods, say February and August.

We examined the securities, as set forth in the balance sheet, and found them as there stated. The market values as at December 31, 1919, as quoted by Messrs. Lee, Higginson & Co. are shown below:

| | |
|---|---|
| $1,000 U S. Government Victory 4¾ per cent, 1922–3 | $ 989 40 |
| 750 U. S Government 4¼ per cent. Third Liberty Loan, 1928 | 710 85 |
| 2,000 Interborough Rapid Transit Co first and refunding 5 per cent. 1966. | 1,120 00 |
| 1,000 Chicago, Milwaukee & St Paul R. R. general and refdg. 4½ per cent., 2014 | 775 00 |
| 2,000 Chicago, Milwaukee & St. Paul R. R 25-year 4 per cent., 1934 | 1,405.00 |
| 1,000 Ill. Central & Chicago, St. Louis & New Orleans Joint first refdg., 5 per cent, series A, 1963 | 767.50 |
| | $5,767 75 |

A further payment of $2500 was made during the year on account of the Institute's proportion of the cost of the addition to the Engineering Building, leaving a balance of $2500 still unpaid.

Proper vouchers and cancelled checks were produced for all disbursements. The cash on hand was counted and found correct, and the bank

balances were verified by certificates from depositories. All cash shown by the books as received was deposited. The footings and postings to the general ledger were checked and found correct.

We are pleased to state that we find the books well kept and the records of the Institute in good order. Every facility and courtesy was shown us during our examination

Very truly yours,
BARROW, WADE, GUTHRIE & Co.

ASSETS

| | | |
|---|---|---|
| Cash on hand and in banks | | $ 3,269 53 |
| Investment of "Life Membership" Fund: | | |
| $2,000 Interboro Rapid Transit 5 per cent bonds, 1966 | $ 1,974 31 | |
| $1,000 Illinois Central, Chicago, St Louis and New Orleans 5 per cent. bonds first ref. series A, 1963 | 1,010 86 | |
| $2,000 Chicago, Milwaukee and St Paul R. R. 4 per cent bonds, 1934 | 1,877 63 | |
| $1,000 Chicago, Milwaukee and St Paul R R 4½ per cent. bonds, 2014 | 824 25 | |
| $ 750 U. S. Government 3d Liberty Loan 4¼ per cent bonds, 1928 | 750.00 | |
| $1,000 U. S. Government Victory Loan 4¾ per cent. bonds, 1922–1923 | 1,000 00 | 7,436 75 |
| Investment of James Douglas Fund: | | |
| $23,000 Idaho Power Co first mortgage 30-year 5 per cent. bonds, American series, 1947 | $20,716 81 | |
| $20,000 American Telegraph and Telephone Co. 30-year collateral trust, 5 per cent. bonds, 1946 | 18,384 72 | |
| $20,000 United Kingdom of Great Britain and Ireland, 20-year 5½ per cent, 1937 | 20,408 85 | |
| $23,000 Utah Power and Light Co first mortgage, 30-year, 5 per cent. bonds, American series, 1944 | 21,185 55 | |
| $20,000 Montreal Tramways Co., first and refunding, 30-year, 5 per cent. bonds, series A, 1941 | 17,620 83 | 98,317.76 |
| Interest in United Engineering Building: | | |
| Land and building, 29 West 39th Street, one-fourth of $1,947,171.16 | | 486,792.79 |
| Library: | | |
| Books and periodicals in Library belonging to American Institute of Mining and Metallurgical Engineers | | 40,000 00 |
| | | $635,816 83 |

### Receipts

| | | | |
|---|---|---|---|
| Initiation Fees | | $ 8,449 35 | |
| Annual Dues: | | | |
| Current dues | $73,280 47 | | |
| Arrears . | 2,389 85 | | |
| Advance . | 3,599.50 | 79,269 82 | $ 87,719 17 |
| Receipts from Other Sources: | | | |
| Sale of Transactions | | $ 4,832 02 | |
| Sale of binding | | 9,402 72 | |
| Sale of advertising | | 12,630 48 | |
| Sale of special editions | | 3,415 90 | |
| Sale of Bulletins and Pamphlets | | 6,082 17 | |
| Sale of pins and fobs.. | | 378 70 | |
| Interest on investments and bank deposits | | 991 35 | |
| James Douglas Income Account | | 3,500 00 | |
| Sundry refunds from societies | | 15 00 | |
| Sundry refunds from members | | 3,804 04 | |
| Gift of George D. Barron | | 5,000 00 | |
| Sundry Receipts . . | | 237.53 | 50,289.91 |
| Special Funds: | | | |
| Life memberships. | | $ 1,500 00 | |
| Sale of Dr Williams' Book "The Diamond Mines of South Africa". . | | 45 00 | |
| James Douglas Fund . | | 100,000 00 | |
| James Douglas Fund Income | | 1,889 17 | 103,434 17 |
| Total receipts.. .. | | . | $241,443 25 |
| Cash on Hand January 1, 1919 | | | 8,258 37 |
| | | | $249,701.62 |

### Liabilities

| | | | |
|---|---|---|---|
| Special Funds: | | | |
| Hadfield prize and interest | | $ 1,154 51 | |
| Thayer prize and interest | | 58.14 | |
| James Douglas fund | | 100,000 00 | |
| James Douglas fund income | | 1,875 42 | $103,088.07 |
| "Reserve for Life Membership" Fund | | . .. | 40,000.00 |
| Members Accounts, Deposit account, 1920 volumes | | . | 1,322.00 |
| Life Membership Fund: | | | |
| Balance January 1, 1919 .. | . . | $ 6,600 00 | |
| Additions during year 1919 . . . | | 1,500.00 | 8,100.00 |
| United Engineering Society: | | | |
| Balance due: Additions to building . | .. $12,500 00 | | |
| Paid to January 1, 1919 | 7,500.00 | 5,000 00 | |
| Paid during year 1919. . . . | | 2,500.00 | 2,500.00 |
| Surplus: | | | |
| As at January 1, 1919. .. ... .. . | | $488,545.06 | |
| *Deduct:* Deficit 1919 .. . . . . . | | 7,738.30 | 480,806.76 |
| | | | $635,816 83 |

DISBURSEMENTS

| | | | |
|---|---|---|---|
| General Funds: | | | |
| Bulletin | | $ 49,277.30 | |
| Year Book | | 3,568 89 | |
| Transactions of 1918 | $1,818 70 | | |
| Transactions of 1919 | 5,406 92 | 7,225 62 | |
| Binding Transactions 1918 | $1,772 23 | | |
| Binding Transactions 1919 | 3,304 57 | 5,076 80 | |
| Special editions | | 735 80 | |
| Editorial and Office | | 33,065 87 | |
| Treasurer | | 1,250 05 | |
| Library | | 5,666 68 | |
| Advertising | | 5,767 18 | |
| Meetings | | 4,896.52 | |
| Local Sections | | 2,687 86 | |
| Technical Committees | | 318 17 | |
| Committee on Increase of Membership | | 6,318 06 | |
| Back Volumes 1–56, Binding, etc | | 838 29 | |
| Engineering Council | | 4,000 00 | |
| Engineering Societies' Employment Bureau | | 1,500 00 | |
| American Engineering Standards Committee | | 200 00 | |
| Legal Expenses, Change of Name | | 280 46 | |
| Memorials: | | | |
| James Douglas | | 391 50 | |
| R. W. Raymond | | 1,953 36 | |
| Donation to widow, A L Gresham | | 675 00 | |
| Pins and fobs | | 215 82 | |
| Sundry advances to members | | 3,457.30 | |
| Sundry disbursements | | 1,674 05 | |
| | | $141,040 58 | |
| Addition to Engineering Building: payment on account | | 2,500 00 | $143,540.58 |
| Special Funds: | | | |
| Proceeds sale of Dr Williams' Book donated to American Red Cross Society | | $ 60 00 | |
| Liberty bonds purchased for investment | | 1,000.00 | |
| James Douglas Investment | | 98,317.76 | |
| James Douglas Income | | 3,513.75 | 102,891 51 |
| Total disbursements | | | $246,432 09 |
| Cash on Hand December 31, 1919 | | | 3,269.53 |
| | | | $249,701.62 |

## Report of Committee on Papers and Publications

DURING the past year, 219 papers were submitted to the Committee and 197 were accepted and printed in the monthly magazine. In addition to these, six papers were presented at the joint session with the Canadian Mining Institute and printed with their discussions in the monthly magazine and the TRANSACTIONS.

For the 119th meeting, held in New York in February, 1919, 58 papers were received, of which 50 were accepted and printed. For the 120th meeting, held in Chicago in September, 1919, 146 papers were submitted, 132 of which were accepted and printed. Of these, 60 were in the Pyrometry symposium and 13 in the symposium on Sulfur in Coal.

The papers for the symposium on Pyrometry were secured largely through the efforts of Dr. George K. Burgess, of the National Bureau of Standards and in coöperation with the National Research Council, which latter bore part of the expense of printing. The value of these papers was so great, being considered a mile-stone in technical literature, that it was decided by the Executive Committee on August 11, 1919, by authority vested in it by By-Law XV, to publish these papers in a special Pyrometry volume to be sold to members and not to be included in the TRANSACTIONS.

At the meeting of the Institute of Metals Division, held in connection with the American Foundrymen's Association in Philadelphia, in September, 15 papers were presented, all of which were accepted and printed.

The title of the monthly Bulletin was changed in October to MINING AND METALLURGY for publication and postal reasons. After being very carefully considered by the Papers and Publications Committee, the Finance Committee, and the Executive Committee, the Board of Directors at its September meeting voted to change MINING AND METALLURGY from a 6 in. by 9 in. publication to a 9 in. by 12 in. standard magazine size and to adopt a broader and more progressive editorial policy. The first number of the new magazine was issued in January and has met with unanimous approval.

In addition, Volumes LX and LXI have been printed and sent to the members.

A very large number of unusually high-grade papers were accepted, whose rejection would have meant a distinct loss to the profession. With such a large number of papers, each requiring careful scrutiny, it has been necessary for the Committee to ask many members to give considerable time to the work of the Institute, which has in every case been very freely done. This coöperation has been a great factor in the success attained by our meetings and publications during the year.

A classification of the papers presented is as follows: Coal, 26; copper, 10; geology and ore deposits, 19; gold and silver, 1; industrial, 11; iron

and steel, 39; metallurgy, 17; metallography, 30; milling, 7; mining, 14; lead, 1; nickel, 2; non-metallic minerals, 2; oil and gas, 11; pyrometry, 60; tin, 1; zinc, 5; miscellaneous, 1. A paper that belongs under two headings has been counted under both.

## Report of Committee on Membership

THE total number of applications brought before the Membership Committee during the year 1919 was 1117, the largest in the history of the Institute; the total number of persons who were elected and became members of the Institute during the same period was 986. The total membership of the Institute on Dec. 31, 1919 was 7834 as against 7195 on Dec. 31, 1918. The changes in membership during the year are shown in the accompanying schedule:

| | | |
|---|---|---|
| Total membership, Dec. 31, 1918. | | 7195 |
| Loss by resignation | 82 | |
| Loss by suspension | 241 | |
| Loss by death. | 89 | |
| Expelled | 2 | 414 |
| | | 6781 |
| Elected.. | 986 | |
| Reinstated.. | 39 | |
| By affiliation with American Institute of Metals | 28 | 1053 |
| Membership, Dec. 31, 1919 | | 7834 |

Change of status: Associates to Members, 15; Junior Associates to Associates, 4; Junior Associates to Members, 6.

## Report of Library Committee

IN ACCORDANCE with the requirements of By-law IX, the report of the Library Committee is herewith appended. The total accessions of the Library for the year were as follows: Gifts, 5,707; purchases, 785; total, 6,993. All of this material has been accessioned and is ready for the use of the readers. On Dec. 31, 1919, the permanent Library collection contained

| | |
|---|---|
| Volumes | 115,934 |
| Pamphlets. | 32,818 |
| Maps and plans | 118 |
| Searches | 3,221 |
| Total. | 152,091 |

The recataloguing of the Library has been started and 9892 volumes have been recatalogued under the new system. It is expected that this

work will proceed with increasing rapidity and will be completed some time in 1921.

The attendance of visitors in the Library showed a total of 22,042, or 46 per cent. increase over the year 1918.

The Mining and Metallurgical Index, which was started in the BULLETIN of September, 1918, has been published throughout the year and is being continued as a feature of MINING AND METALLURGY. The classification is continually being improved and amplified and comparison with the only other two similar indexes published in the English language shows as follows:

| | |
|---|---|
| Items published in 1919 by our index | 7431 |
| The best American mining and metallurgical index | 1267 |
| The best foreign mining and metallurgical index | 1864 |

The sale of publications during the year represented the unprecedented total of $14,330.09 as against $8,528.67 for last year. The sale of TRANSACTIONS amounted to $4,832.02, the sale of special editions $3,415.90, and the sale of BULLETINS and pamphlets $6,082.17.

# PAPERS

# Recent Studies of Domestic Manganese Deposits*

BY E. C. HARDER† AND D. F. HEWETT,† WASHINGTON, D C.

(Chicago Meeting, September, 1919)

SINCE early in 1916, when it became apparent that the steel industry of the United States could not depend for the duration of the war on several important foreign sources of manganese and might have to depend entirely on domestic ores, the U. S. Geological Survey, with the assistance of several cooperating organizations, has made a comprehensive investigation of domestic manganese resources. The work has included field studies in a number of states as well as annual, quarterly, and monthly canvasses among operators concerning current and future production. In order that the interested public might have access to the important results of the studies, preliminary reports concerning the field work as well as periodic summaries of production were issued as promptly as possible.

Although numerous summaries of production [12, 13, 14] have been prepared from time to time for the use of the several boards and committees responsible for the conduct of the war, it has been thought for some months that those engaged in the industry, as well as others, might be interested in a preliminary summary of the accumulated geologic and production data. The cessation of hostilities, in November, 1918, probably detracts from the usefulness of much of the material that is here presented, but apart from the historic interest in the performance of an important industry in war time, there is a possibility that some of the conclusions may be helpful to those who continue to exploit domestic deposits, as well as to those who wish to measure the country's independence in mineral supplies.

Appropriate final geologic reports are being prepared by the geologists who did the field work. The present authors, having done but a small part of the recent field work, acknowledge with great appreciation the use of preliminary and unpublished reports and the cordial informal cooperation of the following geologists in the preparation of this paper: F. L Ransome, G. W. Stose, F. C. Schrader, J. T. Pardee, E. L. Jones, Jr., H. D. Miser, Laurence LaForge, J. B. Umpleby, Arthur Keith, F. J. Katz, and E. S. Larsen, of the U. S. Geological Survey. Field work in Virginia was carried out in cooperation with the Virginia Geological Survey, T. L. Watson, State Geologist; in Georgia, with the Georgia

* Published by permission of the Director, U. S. Geol. Survey.

† Geologist, U. S. Geol. Survey.

[12] References are to bibliography at end of paper

Geological Survey, S. W. McCallie, State Geologist; in Tennessee, with the Tennessee Geological Survey, W. A. Nelson, State Geologist; in Oklahoma, with the Oklahoma Geological Survey, C. W. Shannon, director, in California, with the Committee on Scientific Research, State Council of Defense, G. D. Louderback, chairman; in Oregon with the Oregon Bureau of Mines and Geology, H. M. Parks, director; and in Washington, with the Washington Geological Survey, Henry Landes, State Geologist. Mr. E. H. Wells of the New Mexico School of Mines, Socorro, N. M., has kindly submitted results of recent field work in that state to the U. S. Geological Survey, and Mr. J. C. Jones of the Mackay School of Mines, Reno, Nev., has done the same with regard to field work in Nevada during the season of 1918.

## Ore Deposits

### *Ore-forming Minerals*

Most of the specimens that have been collected during recent work have not yet been closely studied by the geologists who collected them, and it will only be necessary to make brief statements at this time concerning the more important minerals that make up the domestic ores. Several rare, and probably new, minerals have been recognized recently, but it would not be appropriate to describe them in advance of study and detailed presentation of data by the respective observers.

*Sulfides.*—Alabandite, sulfide of manganese, was recognized by J. T. Pardee[31] in ores from the Siegel mine, White Pine Co., Nev., and by F. L. Ransome[38] in ores of the Tombstone district, Ariz. Good specimens of the same mineral have been sent to the U. S. Geological Survey by W. K. Morrow, of Rodeo, N. M.

*Oxides.*—Recent work has shown that there is widespread carelessness in the identification of the common manganese oxides, as well as confusion concerning the properties of several of the oxides,* and plans are under consideration for the careful study by modern methods of many of the specimens that have been collected.

Psilomelane is the principal mineral in most of the mines in Virginia, Tennessee, and Georgia, as well as many mines in Arizona, Nevada, and California; but it is commonly associated with more or less pyrolusite, manganite, and wad. Unusually good crystalline pyrolusite, together with some manganite, has been collected recently from deposits in

[31]References are to bibliography at end of paper.

*The recent work by T. L. Watson and E. T. Wherry [Pyrolusite from Virginia, *Jnl.* Wash. Acad. Sci. (1918) 8, 550–560] shows that it is difficult to distinguish pyrolusite from manganite.

Minnesota,[6] Arizona, Utah, and Wyoming.[16] Recent work confirms the early conclusion of Penrose[36] that braunite is the principal mineral of the high-grade ore of the Batesville district.[25] Braunite has been recognized in ore from one locality in Tennessee,[40] and good crystals of the mineral were recognized by one of the authors as an important constituent in material from deposits east and south of Bromide, Okla.[3]

It is widely known that manganese oxides have been recognized that range from manganous oxide (MnO), represented by the mineral manganosite, to manganese dioxide ($MnO_2$), represented by the minerals pyrolusite and polianite. A number of hydrous forms are known also. A preliminary review of the associations of the oxides indicates that, beginning with the minerals that contain a certain per cent. of oxygen, all those that contain more only form in the belt of weathering, whereas those that contain less form under conditions found deeper below the surface. That per cent. which marks the division between these zones appears to be present in braunite ($3Mn_2O_3 . MnSiO_3$).

*Carbonates.*—Until the first shipments of rhodochrosite from the Emma mine, Butte, Mont., were made in December, 1917, the oxides of manganese were practically the only manganese minerals in the marketable materials from domestic manganese mines Since that time, a large quantity of the material has been shipped and a marketable rhodochrosite product has been made by milling material from other mines in that district. Rhodochrosite has also been recognized in many deposits in California,[22] eastern Nevada,[31] and eastern Utah [34]

Argall's conclusion[2] that manganese is characteristically present in siderite of veins and similar deposits related to igneous rocks, whereas it is low or absent from siderite of sedimentary origin, probably has wide application, but sideritic concretions from the Batesville district are striking exceptions.[25] An unusual crystalline carbonate of manganese, calcium, and magnesium has been discovered by A. Keith and G. W. Stose* in a deposit that replaces dolomite along a fault in Sevier Co., Tenn. Analyses of specimens of limestone from several states, such as the Holston marble of eastern Tennessee,[40] indicate that small percentages of manganese are commonly present in such rocks. Study of several small deposits appears to confirm a widespread impression that they may be formed by the concentration of such disseminated manganese during weathering.

*Silicates.*—Although the common silicate of manganese, rhodonite, has been found in a number of new localities, notably in California and eastern Nevada, the most interesting manganese-bearing silicate en-

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[6] References are to bibliography at end of paper.

* Personal communication.

countered during recent work is that recognized by Pardee[35] in the Olympic Mountains, Wash. In appearance the mineral closely resembles a common variety of hornstone or jasper, but it may be distinguished by greater specific gravity. An unweathered specimen from the Black and White group of claims, 18 mi. northwest of Hoodsport, contained manganese, 35.9 per cent.; silica, 23.68 per cent.; ferric oxide, 3.52 per cent.; alumina, 3.48 per cent.; magnesia, 1.31 per cent.; lime, 1 56 per cent.; and loss on ignition, 18.32 per cent.

### *Geographic Distribution*

Although large deposits of a grade comparable with that from several foreign sources of manganese ore appear to be lacking in the United States, small deposits of acceptable material are numerous and widely distributed and large low-grade deposits are found in several places. The principal districts in which high-grade manganese ore is mined, in the order of their importance as recent sources of production, are: Philipsburg district, Mont.; Butte district, Mont.; Southern Nevada region; Central California region; Bisbee district, Ariz.; Appalachian region of Virginia; Batesville district, Ark ; Tintic district, Utah; and Cartersville district, Ga. Other deposits of some importance are sporadically distributed throughout western Arizona, southern California, eastern Tennessee, southern Oregon, and central Utah.

The principal producing districts of ferruginous manganese and manganiferous silver ores, also given in the order of their importance, are: Cuyuna district, Minn.; Leadville district, Colo.; Southern Nevada region; Silver City district, N. M.; Crystal Falls district, Mich.; Appalachian region of Virginia; Cartersville district, Ga.; and Batesville district, Ark. Manganiferous zinc ores, from which the residuum after the zinc has been extracted is used for its manganese content, are mined at Franklin Furnace, N. J. Deposits of manganiferous iron ores with low manganese content are found in the Penokee-Gogebic district, Wis., and in the Crystal Falls district, Mich.

Manganese-bearing ores are thus widely scattered throughout the United States. Commonly the manganese minerals in the deposits are associated with other ores, such as iron ores in the Lake Superior district, in some deposits in Batesville district, and the Appalachian region; silver ore (or more commonly silver-lead ore) in the Philipsburg, Tombstone, Leadville, Pioche, and Tintic districts; zinc ore in the Butte, Red Cliff, and Franklin Furnace districts; and copper ore in the Bisbee district. There are, however, some deposits in which manganese ore occurs alone, as in the principal deposits in the Batesville district, in the

[35] References are to bibliography at end of paper

California deposits, in many deposits in Utah, New Mexico, Arizona, southern Nevada, southern California, and in the Appalachian region of Virginia, Tennessee, and Georgia.

Where manganese minerals are associated with minerals of other metals, they have usually been regarded as a gangue material of little or no value. In many places where associated with iron ore they have even been considered an undesirable impurity. Where associated with ores of precious or semi-precious metals, they are commonly used for flux as in the Leadville, Pioche, Tintic, and Tombstone districts. With the rise in the price of manganese ore during the war, however, many such ores have been mined extensively for the manufacture of manganese-iron alloys, some of them for the first time.

*Physiographic Relations*

Although there are conspicuous exceptions, most deposits of the common manganese oxides are confined to a shallow surface zone that has recently come under the influence of circulating waters of surface origin. The exceptions noted in the foregoing appear to have been formed in shallow bodies of water having continual access to abundant supplies of oyxgen in the air. The first group is widespread in regions that show great diversity of surface forms and that show a wide range in rainfall as well as temperature. It is noteworthy that although some large bodies of manganese oxides are known within the limits of Pleistocene or recent glaciation, the number of important deposits in unglaciated regions is much larger, and these deposits yield most of the world's supply of manganese ore.

As the result of studies in many regions, it is widely recognized that the depth and degree of rock weathering are closely related to climate and surface forms. Although locally, near fissures and similar zones of easy access to surface waters, oxidation and related rock decay may persist to depths ranging from 500 (152 m.) to even 2500 ft. (760 m.) below the surface in regions with high relief, it is uncommon to find rocks completely decayed over large areas more than 250 ft. below the surface. The areas that show complete decay appear to be restricted to regions of low relief or regions of moderate relief that have had unusual physiographic development. It is apparent from the foregoing outline that localized bodies of manganese carbonates and silicates may be deeply weathered to oxides near fissures in regions of high relief, but it appears highly improbable that in such regions large bodies of manganese oxides could be formed by gathering from a large area the manganese that was widely disseminated through the rocks. As a matter of fact, rather meager exploration of most of the manganese oxide bodies in the regions of high relief in the western states shows that they overlie or occur near

bodies of manganiferous carbonates or silicates, commonly rhodochrosite, siderite, or rhodonite, that were the source of the manganese now present as oxides.

Most of the manganese deposits of several eastern states, notably in the Appalachian Valley in Virginia, Tennessee, and Georgia, possess features that are unlike those of the western deposits, in that they yield only oxides of manganese in clay, here and there residual from shale, limestone, or dolomite, but elsewhere apparently sediment in ancient river channels. Extensive explorations in these clays to depths that range from 200 to 312 ft. (60 to 95 m.) have not yet revealed the presence of a carbonate or silicate from which the manganese might have been derived, although the persistent association of the deposits with a zone of sedimentary rocks about 200 ft. thick partly justifies the conclusion that certain layers of the unweathered rocks contained manganiferous carbonate concretions.

As most of the several hundred deposits of that region underlie remnants of upland terraces that coincide with the position of an ancient plain now much dissected, the suggestion has been made[10] that they were formed during the later stages of the development of the plain. Some deposits, such as the Kennedy and Midvale, in Augusta and Rockbridge Counties, Va., occur in transported sands and clays and the local features are such that they could not have received more than insignificant contributions of manganese since this plain was dissected. During the period of the establishment of local plains, there would be an exceptional opportunity for manganese, as well as other substances, to be dissolved by surface waters from the rocks above the level of the plain and carried to a relatively shallow surface zone under the plain. Studies of manganese deposits in regions where the rocks are deeply decayed should include careful consideration of the recent physiographic histories of the regions.

### *Stratigraphic Relations*

Manganese ores are found in the United States in rocks of nearly all geologic periods. There are certain series of rock formations, however, with which manganese deposits are persistently associated over large areas, such as the Shady limestone and adjacent horizons of the Cambrian in the Appalachian region, the Cason shale of the Ordovician in the Batesville district, Ark., the iron-bearing formations of the Huronian in the Lake Superior region, the jasper of the Franciscan formation of the Jurassic(?) in California, the Tertiary lavas and tuffs in the Southwest, and the Carboniferous limestones in various western mining dis-

[10] References are to bibliography at end of paper.

tricts such as Leadville, Bisbee, Tintic, and others. In places, the ore-bodies are enclosed in the fresh rock itself; elsewhere, they are found with decomposition products of the rock.

In the case of the most typical of these occurrences, distinct manganese minerals are found in the original rock either as bedded lenses, local concentrations, or in a disseminated state. Weathering as well as erosional processes have commonly produced further concentration of such manganiferous material into ore deposits. To this class belong the ores of the Appalachian region, of Arkansas, of the Lake Superior region, and of California. There are also certain rock formations, however, such as the Carboniferous limestones of the West, which in widely scattered areas contain important manganese-ore concentrations in places where manganese minerals were not originally constituents of the rocks but have been brought in by outside agencies, particularly by water associated with intrusions of igneous rock. The materials have replaced these particular rocks because of certain lithologic or chemical characteristics such as easy solubility or porosity.

*Stratigraphic Distribution.*—Table 1 shows the stratigraphic distribution of the principal manganese deposits in the United States. The manganese ores of the Piedmont region of Virginia, South Carolina, and Georgia are associated mainly with decomposed residual material derived from Cambrian or pre-Cambrian schist, gneiss, and associated rocks. They occur along certain zones generally parallel to structures such as banding or schistosity in the inclosing rocks. In places, they are found along the contact of two different kinds of rock but elsewhere they occur wholly within one kind of rock. Deposits of this type, though widely scattered, are usually small and have supplied a relatively small part of the total manganese ore produced in the United States.

TABLE 1.—*Stratigraphic Distribution of Manganese Deposits*

Cenozoic
- Recent
  - Bog material near Wickes, Mont. and elsewhere
  - In surface wash in Applachian region.
- Pleistocene and Tertiary
  - In Tertiary rhyolitic tuff near Las Vegas, Nev., and Topoch, Ariz.
  - As fracture fillings and replacements in rhyolitic and other lavas and tuffs in New Mexico, Arizona, Nevada, Utah, and Southeastern California.
  - In basaltic tuff in Lake Creek district of southern Oregon.
  - In monzonite and aplite, in Butte district, Mont.

Mesozoic
- Cretaceous
  - In McElmo formation of Green River region, Utah
  - Associated with Cretaceous limestone near Shumla, southwestern Texas

- Jurassic
  - In jasper of Franciscan formation (Jurassic?) in California
  - With metamorphosed limestone and igneous flows, in Olympic region, Wash.
- Triassic
  - In the Dolores formation, San Miguel Co , Colo
  - In shales of the Newark Group, near Annandale, N J.

Paleozoic

- Carboniferous
  - In Carboniferous limestone in Tintic district, Utah
  - In Carboniferous limestone in Bisbee district, Ariz
  - Veins in Carboniferous limestone in southwestern New Mexico
  - In Carboniferous limestone in Tombstone district, Ariz
  - In Carboniferous limestone in Redcliff district, Colo
  - In Leadville "blue" limestone in Leadville district, Colo.
  - In Carboniferous and other limestones in White Pine Co , Nev.
  - In Fort Payne chert in eastern Tennessee and northeastern Alabama
  - In Kaibab limestone, Coconino Co , Ariz.
  - Associated with chert in Casper formation, Albany Co , Wyo
  - In Calaveras formation, California
- Devonian
  - In Arkansas novaculite in west central Arkansas.
  - At Oriskany horizon along Alleghany front in western part of Virginia.
- Silurian
  - In calcite veins in limestone of Salina age in western part of Virginia
  - Veins in faults near Bromide, Okla
- Ordovician
  - In Ordovician limestones near Silver City, N. M
  - In Cason shale and Fernvale limestone of Batesville district, Ark
  - In Holston marble in southwestern part of Great Valley in Tennessee
  - In Knox dolomite (Ordovician and Cambrian) along Appalachian Valley of Virginia, Tennessee and Georgia
- Cambrian
  - In Cambrian limestone at Pioche, Nev.
  - Near Renova, Madison Co , Mont.
  - In Hasmark limestone of the Philipsburg district, Mont
  - In Shady limestone and adjacent formations along the west slope of Appalachian Mountains, in Virginia, Tennessee, and Georgia.
  - In Hiwassee slate near Sevierville, Tenn.
  - In schist and limestone in Piedmont region, Va.

Proterozoic

- Algonkian
  - Near Cherry and Wigwam Creeks, Madison Co., Mont
  - Manganese-bearing beds of Cuyuna, Crystal Falls, and other Lake Superior ranges.
  - In schist and gneiss of Llano and Mason Co , Tex
- Archean?
  - Some deposits of Piedmont region of Virginia, North Carolina, South Carolina, and Georgia.

The manganese occurrences in the pre-Cambrian area of central Texas are small and unimportant.[28] They consist of lenses parallel to

[28] References are to bibliography at end of paper

the banding in schist and gneiss. The manganese minerals are principally spessartite and piedmontite. They occur interlayered and intermixed with quartz, feldspar, and muscovite, and are superficially oxidized to manganese oxides. Deposits of the type of the central Texas ores are common in Brazil and India and furnish much of the ore imported into the United States from these sources.

The iron-bearing beds of the Lake Superior region[7] are manganiferous in several places, notably in the Cuyuna district,[23, 26, 27] but manganiferous iron ores have also been produced locally in the Mesabi and Crystal Falls districts. In the Cuyuna district the deposits are found along certain beds of the Huronian iron-bearing formation. Generally, the associated rock is ferruginous slate or ferruginous chert, and the manganese oxide both replaces the iron-bearing formation and fills fractures. The ore is richest along the borders of joints and minor fractures and becomes leaner the farther it is from such fracture planes. Here and there small ore blocks bounded by fracture planes have unreplaced centers. Although in their present form the ores are largely replacements, there were present in the same or adjacent beds original manganese minerals that have, through solution, oxidation, and redeposition, yielded the present ores. These were probably mainly manganese carbonate, which is still present in the less altered phases of the rock. The ore of the Cuyuna district consists mainly of manganite and pyrolusite mixed with hydrated iron oxides. Psilomelane is also widespread, however.

In the Madison Valley near Cherry and Wigwam creeks, Mont.,[30] there are several occurrences of manganese ore associated with marbleized limestone belonging to the Cherry Creek (Algonkian) formations consisting of mica schist, gneiss, and marble. The ores occur not far below the base of the Flathead quartzite (Cambrian) and are in the form of lens-like bodies of mixed psilomelane, manganite, and wad, associated with some calcium carbonate and iron oxides as impurities. The limestone is locally discolored due to small amounts of iron and manganese minerals disseminated through it. It is believed that the bodies recently explored were formed during the Tertiary erosion period by the solution of these minerals and the redeposition (in openings) of their contained manganese and iron.

*Manganese Ores in Cambrian Rocks.*—Of the manganese ores in Cambrian rocks, those in the Appalachian region and those in the Philipsburg district are most important, the Philipsburg district alone having supplied more than one-half of all the high-grade manganese ore mined in the United States in 1917 and 1918. The manganese orebodies at Philipsburg[4, 45] are found in the Hasmark magnesian limestone and most of them have the form of tabular bodies or pockets. These bodies

[7] References are to bibliography at end of paper.

are associated with silver-bearing quartz veins and, like them, are related to intrusions of Tertiary granodiorite. There is, thus, no age relationship between the ores and the inclosing Cambrian limestone. Some of the orebodies consist of nodules of manganese oxides in a clayey matrix, others consist of soft powdery manganese oxides; in both cases the ore is probably a mixture of manganite and pyrolusite. The ores mined are manganese oxides, but in the lower levels of some of the mines rhodochrosite has been encountered; this is probably the original manganese mineral. The principal gangue mineral associated with the manganese oxides is quartz.

The ores occurring along the west slope of the Appalachian Mountains,[41] from Pennsylvania to Georgia, although not sedimentary in their present form, are so persistently associated with residual material derived from a certain stratigraphic horizon as to make it almost certain that the deposits are derived by concentration of manganese in minerals disseminated through strata at or near this horizon. The deposits consist of small fragments, nodules, or larger bodies of manganese oxides in residual clay or sand formed by weathering from the Shady limestone or locally, from the lower part of the overlying Watauga shale. These clays are manganese-bearing in many places but only locally are the particles and fragments of manganese oxide in them abundant enough to form ore deposits. The principal manganese minerals are psilomelane, pyrolusite, and manganite. All three may occur in the same fragment. Wad is found in many places. Transitional sandy and shaly beds occurring between the Shady limestone and the underlying Erwin quartzite are locally manganese and iron-bearing and may represent the source from which some of the ore now occurring in the residual clay is derived. The principal localities in which ores associated with the Shady limestone are found are the Blue Ridge region, Va.,[11, 41, 50] the northeastern Tennessee region,[40] and the Cartersville district, Ga.[21, 49, 51]

The other deposits of manganese minerals in Cambrian rocks are of minor importance and little or no ore has been produced from them. Unique manganese-carbonate deposits associated with the Hiwassee slate have been found at one locality, viz., in the Great Smoky Mountains, 13 mi. southeast of Sevierville, Tenn.[40] The orebodies occur as nearly vertical lenses along the contact of slate and dolomite and consist of an uncommon carbonate of manganese, calcium, and magnesium which replaces dolomite. The carbonate ore is oxidized to psilomelane and other oxides to a depth of 6 to 10 ft. (1.8 to 3 m.). The lenses are as much as 5 ft. (1.5 m.) thick.

Manganese ores associated with Cambrian rocks in Montana are situated in the Jefferson Valley, near Renova, Madison Co.[30] A bed

[41] References are to bibliography at end of paper.

of iron ore, consisting mainly of hematite with some limonite, is found a short distance above the top of the Flathead quartzite (Cambrian). It is as much as 15 ft. (4.5 m.) in thickness and is usually underlain by shale and overlain by limestone, both of Cambrian age. Locally, the middle portion of the iron-ore bed is rich in manganese oxides, mainly psilomelane and pyrolusite and constitutes a manganiferous-iron ore. The bed of oxides is believed to be of sedimentary origin.

The Pioche district, Nev.,[44] has important deposits of silver-bearing ferruginous-manganese ore but this ore has been mined thus far only for fluxing purposes. As at Philipsburg, the manganese of the deposits at Pioche was not deposited with the sediments in which the orebodies are found. It has been brought in by later ore-bearing solutions that have replaced certain beds of limestone and, to a less extent, calcareous shale. Such replacement deposits have been found at five or six horizons in the Pioche district, all of Cambrian age, separated by layers of barren rock. The ores are oxides, 503 ft. (156 m.) below the surface, the present lower limit of explorations.

*Manganese Deposits in Ordovician Rocks.*—The most important manganese deposits in rocks of Ordovician age are those of the Batesville district in northern Arkansas[25, 36] These ores occur in or near a definite stratigraphic zone known as the Cason shale, to which they are genetically related. The unweathered Cason shale is a greenish sandy shale which contains many flat concretions of manganiferous siderite, and is locally phosphatic. By weathering, the nodules are set free and enriched with additional manganese. The shale forms the uppermost member of the Ordovician. It is usually unconformably underlain by the Fernvale limestone, also of Ordovician age, and overlain by the St. Clair limestone (Silurian); or, where the limestone is absent, by the Boone chert (Carboniferous).

The ores of the Batesville district are of two kinds: (1) stratified beds of iron and manganese-carbonate material that forms part of the Cason shale, and (2) richer manganese ores occurring in residual clay derived from the weathering of the Cason shale and Fernvale limestone and probably originating by solution and redeposition of manganiferous material that at one time was in the Cason shale. The ore beds in the Cason shale consist of small flattened buttons of a manganiferous carbonate imbedded in a ferruginous sandy or shaly matrix, the whole forming a low-grade ore. The buttons are oxidized only near the surface. The ore in the residual clay and weathered shale derived from the Fernvale and Cason beds, respectively, is in the form of nodules and irregular fragments and bodies of all sizes from a walnut to a barrel. Most of it is hard and consists of psilomelane and braunite, but locally soft oxides

[44] References are to bibliography at end of paper.

are found as well. The ore in residual clay is an enriched product derived during the weathering of the lean stratified ore of the Cason shale.

Similar to the Batesville deposits, in relation to the inclosing rocks, are the manganese deposits associated with the Holston marble in southeastern Tennessee.[40] There is believed to have been interruption in sedimentation between the deposition of the Holston marble and that of the overlying Tellico sandstone, as there is an irregular contact between the two formations. Along this contact, manganese and iron oxides occur in the basal beds of the Tellico sandstone, extending downward into irregular openings in the upper part of the Holston marble. Enriched bodies of oxides, largely very pure pyrolusite and manganite, are found in residual clays derived from the weathering of these rocks.

Manganese oxides are, in many places, associated with chert and clay derived from the weathering of the Knox dolomite.[40] The chert masses have been fractured and then recemented and partly replaced by manganese oxides. Fragments and disseminated particles of manganese oxides occur also in the associated clay. Such occurrences of ore are widely distributed through the Appalachian valley, but only locally have the oxides been sufficiently concentrated to form deposits. The manganese has been gradually segregated during the weathering of the Knox dolomite, in which it appears to have been present originally in minute quantities widely disseminated.

The ferruginous-manganese orebodies at Silver City, N. M.,[47, 52] are associated with upper limestone members of the Ordovician, in which they occur as irregular lenticular replacements more or less parallel to the bedding. Some of them occur near quartz veins but there is no regularity in this association. Although granodiorite is found in the immediate vicinity of the ore-bearing area, the deposits are believed not to be related to it but to have been formed by the leaching out of manganese from overlying shale beds. The ores are mixed iron and manganese oxides. Exploration work is shallow and the character and extent of the deeper ore is not known.

*Manganese in Silurian Rocks.*—Rocks of Silurian age appear to be deficient in manganese ores in the United States. A few occurrences are known in the western part of Virginia,[42] where manganese oxides are found in calcite veins in limestone of Salina age. This ore may have been introduced from manganese deposits in overlying rocks of Oriskany age, which are discussed later.

Near Bromide, Johnson Co., Okla.,[3] several deposits of manganese ore are found along faults in rocks of Silurian age. The most important deposits form a group of lenses near Springbrook (Viola) and consist of mixtures of braunite and a manganiferous carbonate, which, in places,

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[40] References are to bibliography at end of paper.

fill an extensive northwest trending fault fissure between the Arbuckle limestone (Cambrian and Ordovician) on the south and the Viola limestone (Ordovician) on the north. Another deposit east of Bromide, consisting of similar minerals, occurs along a northwest trending fault in the "Hunton limestone" (Silurian and Devonian). The deposits near Springbrook are as much as 6 ft (1.8 m ) in width and extend along the fault about 800 ft. (243 m.).

Locally, in western United States, in areas where the principal manganese deposits are associated with Carboniferous limestones or with Tertiary lavas, minor occurrences of ore, such as small veins or replacements, may be found in early Paleozoic rocks, including the Silurian, where these rocks are present.

*Manganese in Devonian Rocks.*—Ores in Devonian rocks that have been exploited to a considerable extent are found in two regions in the United States, in west-central Arkansas[24] and in the western part of Virginia.[42] In west-central Arkansas and eastern Oklahoma, manganese oxides are associated with the Arkansas novaculite. The Arkansas novaculite consists of three divisions: (1) a lower white novaculite at the top of which there are sporadic manganese deposits that may indicate a period of non-deposition of sediments; (2) a middle dark novaculite with interlayered shale having a conglomerate at the base; and (3) an upper massive calcareous novaculite also carrying manganese oxides. The manganese minerals are found along bedding planes, in joint cracks, veins, or as cement in novaculite breccia. They consist principally of mixtures of the manganese oxides, psilomelane, pyrolusite, manganite, and wad, associated in places with iron oxides. The deposits are small and are scattered through the hard novaculite, so that their exploitation has not been successful commercially.

The manganese ores along the Alleghany front, in the western part of Virginia, occur in the lower part of the Oriskany sandstone. They are associated with the Oriskany iron-ore deposits, which are of widespread occurrence along this horizon, and many of them are ferruginous because of this association. Most of the ores are in the form of breccia fillings and replacements in sandstone of the Oriskany formation and occur on benches as well as on mountain slopes. Certain of the more important orebodies of this group are situated where the crests of anticlines are exposed in areas that represent residuals of the Cretaceous peneplain that was once widespread in the Appalachian region. Some high-grade deposits also occur in synclines as brecciated cave fillings along calcareous layers between sandstone beds. Not all the ores occur in bedrock, however; some are found in residual soil derived from the weath-

[24] References are to bibliography at end of paper.

ering of these formations and occur as nodular masses in sandy clays. The ore is mainly psilomelane, but pyrolusite also occurs in abundance.

*Manganese in Carboniferous Rocks.*—Manganese ores are associated with Carboniferous rocks in many places in the Rocky Mountain and Great Basin regions in the western part of the United States; but the only group of deposits associated with rocks of this age known in the east is found in the Fort Payne chert in southeastern Tennessee[40] and northeastern Alabama. The ore in the Fort Payne chert is found as fillings of fractures and as replacements of the chert. As the chert masses weather, they break up into fragments and the inclosed ore is either left free or is associated with more or less chert in residual soil. The ore is mainly psilomelane.

Of the deposits associated with Carboniferous rocks in the west, those in the Leadville, Red Cliff, Tintic, Bisbee, and Tombstone districts, and in parts of southwestern New Mexico, show many similar features. In all these localities, the manganese ores are generally more or less closely associated with ores of more valuable metals, such as silver, zinc, lead, or copper. In all these localities, also, the ores are associated with Carboniferous limestones, not because of any original manganese content in these rocks, but because ore-bearing solutions, derived probably from igneous intrusions, found these rocks readily replaceable. Limestones of Carboniferous age are widespread in the west and the common association of ores with them is easily accounted for by their prevalence.

In the Bisbee district,[1, 38] the manganese orebodies are generally superficial although in one mine a body of manganese oxides has been found at a depth of 1300 ft. (396 m.). They are found not only in the copper-bearing area immediately adjacent to Bisbee but are scattered over a wide outlying area. Although they occur in the same limestones (Naco and Escabrosa) as the copper orebodies and are associated with the same north-and-south fissure zones, the manganese orebodies are thought to be more or less distinct from the copper orebodies. Manganese ores have been found at the surface in places where copper ore occurs at depth, but the connection of the two occurrences has not been demonstrated. However, most of the manganese ores contain a small percentage of copper and, like the copper ores, probably owe their origin to solutions accompanying the intrusion of the granite porphyry. The ore is almost entirely psilomelane associated with calcite and barite. It forms replacements and fracture fillings in Carboniferous limestone along fissure zones.

In the Tombstone district,[1, 38] the manganese ores occur at intervals along fissure zones in Carboniferous limestone, usually not far from intrusive granodiorite. Most of the ore replaces limestone but, to

[40] References are to bibliography at end of paper.

some extent, fracture fillings occur also in the limestone. The ore is mainly psilomelane. Manganiferous calcite is found in some of the mines, and it may have been the source of much of the manganese in the ore. Alabandite also has been found in small quantities in the district At Tombstone the manganese oxide is part of the gangue of silver minerals which have greater value than the manganese. For this reason, the raw ore is treated in a concentrating plant which yields a high-grade manganese concentrate and a silver-bearing residue that is most valuable as a flux in smelting. The percentage of iron in the Tombstone manganese ores is small.

The ores at Leadville[2, 46] contain both manganese and iron oxides. Most of them are found in the Leadville or "Blue" limestone below the white porphyry sill and represent weathered replacements of the limestone. Fissuring has probably had much to do with their localization. The ferruginous-manganese ores occur mainly in old lead workings, but some ores have been found in old mines from which very little lead or zinc has been produced. Most of the Leadville ores are argentiferous and the manganese and iron oxides have been useful as fluxes in smelting, as has been the case at Tombstone. The silver is probably present in the oxidized ores as chloride and the lead as carbonate. The unoxidized ores contain pyrite, galena, zinc blende, and a small amount of silver chloride and the gangue minerals are quartz, barite, calcite, and manganiferous siderite. The latter is the source of the manganese oxides in the oxidized ore.

The deposits at Red Cliff,[48] like those at Leadville, yield ferruginous-manganese ore. The orebodies are in the form of great lenses parallel to the bedding of the inclosing Carboniferous limestone, which dips 15° NE. Synclines trending at right angles to the strike of the bedding appear to have influenced the localization of the orebodies. The unoxidized ore consists of pyrite and sphalerite and some galena in a manganiferous siderite gangue, whereas the oxidized ore which extends to a depth varying from a few hundred feet to over 800 ft. (243 m.) below the surface, consists of iron and manganese oxides with some lead carbonate.

The Tintic district[32] produces silver- and lead-bearing manganese and ferruginous-manganese ores that, until recent years, were used mainly for flux in smelting silver ores. The orebodies are irregular branching pockets, lenses, and pipes that occur along the upper parts of jaspery quartz lodes in limestone, shale, or quartzite. The oxidized ore consists of a mixture of pyrolusite and wad and most of it is soft and crumbly. At water level, depth about 100 ft. (30 m.), these minerals have in some bodies been found to give way abruptly to rhodonite.

[2] References are to bibliography at end of paper

Jaspery quartz is generally associated with the manganese oxides, and more or less silver and lead occur in all the ore. The unweathered minerals are thought to have been formed by solutions of deep-seated origin.

At Lake Valley, Kingston, and other localities in southwestern New Mexico,[20, 52] manganese and ferruginous-manganese ores occur in Carboniferous and perhaps other Paleozoic limestones. The ores are mainly replacements and fracture fillings along fissure zones and were formed by ore-bearing solutions originating from igneous intrusions. All contain silver, lead, or zinc. The oxidized ores consist of various manganese oxides, manganite, pyrolusite, or wad, associated with more or less iron oxide, quartz, and calcite. Below water level, these minerals usually give place to rhodonite, rhodochrosite, or ankerite with more or less quartz, calcite, argentiferous galena, sphalerite, pyrite, and other sulfides.

The ores in the Ely and neighboring districts in White Pine Co., Nev.,[31] are mainly irregular replacements in limestone and associated bodies of jaspery quartz. Granodiorite intrusions are generally found in the neighborhood of the orebodies. The bulk of the ore is in the form of soft oxides, pyrolusite, and wad; but psilomelane and manganite are also widespread and large masses of a hard compact oxide believed to be braunite have been found in several mines. Jaspery quartz, calcite, and iron oxides are associated with the manganese oxides. In one mine (Siegel mine) water level was reached at a depth of 200 ft. (60 m ) and below it rhodonite, rhodochrosite, and alabandite were found associated with pyrite and galena. Cerusite and massicot have been found in the oxidized ore. All the ore contains more or less silver.

The ore deposits described are of the silver, lead, zinc, or copper-bearing type. All occur in the same group of rocks in widely separated areas but in every case the manganese they contain was derived from deep-seated sources and was not a part of the rocks in which the deposits are now found. There are a few groups of deposits in the west, however, in which the ores are more closely related genetically to the associated rocks; these are described in the following paragraphs.

The manganese ore of Albany Co., Wyo.,[19] is associated with a bed of chert several feet thick interlayed in a formation consisting of pink and maroon limestone and some sandstone. The limestone rests on pre-Cambrian granite. The manganese minerals, pyrolusite and manganite, form fernlike aggregates in chert.

The manganese ores of Coconino Co., Ariz.,[17] are situated on Mogollon Mesa. They are in the form of superficial replacements and breccia fillings in flat-lying limestone and associated sandstone. The replace-

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[20] References are to bibliography at end of paper.

ments tend to follow bedding planes and are found within a few feet of the surface, whereas breccia fillings extend to greater depths. Within several feet from the surface, the rock is decomposed and in this material manganese oxide occurs as nodules and masses. The ore is mainly psilomelane. Some is fairly pure but where sandstone has been replaced abundant sand grains remain. Brown iron oxide is associated with the manganese oxide.

Ore in Triassic sediments, somewhat resembling that found in Cretaceous rocks in the Green River region of Utah, to be described later, is found in San Miguel Co., in western Colorado.[18] The deposit forms a thin layer interbedded with sediments of the Dolores formation (Triassic). Above it is red clay and below it red sandstone. The ore is medium soft and consists of pyrolusite. Calcite is commonly associated with it and locally barite has been found. Exploration from the outcrop, however, shows that pyrolusite gives place to slightly manganiferous calcite. For this reason, it is believed that manganese has been derived from overlying rocks and has partly replaced the calcite. Near Annandale, N. J., manganese oxides locally replace red shale of the Newark group (Triassic) in the vicinity of quartz veins. The shale is nearly horizontal and the group of narrow quartz veins, which are vertical, may be traced about 600 ft. (182 m.).

*Manganese in Jurassic Rocks.*—Jurassic rocks contain important manganese deposits in central California.[22] These ores are found in beds and lenses of jasper occurring at various horizons in the Franciscan shale and sandstone formation, of probable Jurassic age. The manganese ores, which are oxides near the surface, replace the chert and fill fractures in it. Most of the orebodies form pockets or irregular beds; but in a few places they occupy fault planes and fissures bounded by chert walls. Some of the surface ore is hard and massive, but some is porous and crumbly. As replacement of the chert is rarely complete, much of the ore is very siliceous. As explorations in many localities have not extended below the zone of oxidation, the source of the manganese is obscure but the fact that recently large quantities of rhodochrosite have been found in some of the California deposits indicates that, locally at least, the present ores are derived from the oxidation of manganese carbonate.

The manganese-silicate deposits of the Olympic Mountains[35] are among the interesting new discoveries of manganese occurrences. Deposits are found at several widely separated areas but the geologic occurrence in all places is similar. The orebodies are found in regionally metamorphosed greenstone, argillite or slate, and limestone of probable Jurassic age. Most of the bodies are closely associated with the lime-

[18] References are to bibliography at end of paper

stone, which is strongly discolored by manganese and iron oxides. They are lens-like in form and consist mainly of an uncommon brown hydrous manganese silicate associated with some red jaspery quartz, hematite, and rhodonite. The hydrous manganese silicate is pale brown in color and is fine-textured and flinty in appearance. For several feet from the surface, the manganese minerals are weathered to black oxides. At one locality, a hard, heavy, compact, dark green oxide, that may be manganosite, was found forming from the oxidation of the manganese silicates.

*Manganese in Cretaceous Rocks.*—In the Green River region in central Utah,[34] manganese oxides appear to form lenticular bodies over an extensive area at a definite horizon in calcareous, gypsiferous, red sandstone of the McElmo formation (Cretaceous?). As the rocks are nearly flat, the outcrop of the ore-bearing zone forms a sinuous line around hills and ridges. The maximum thickness of any lens is 3 ft. (0.9 m.). The zone extends over many square miles. Below the ore zone, manganese oxides form veins and nodular concretions along joints in the sandstone; where the manganese ore zone has been weathered and disintegrated many fragments of manganese oxide occur in residual soil. Near the surface the ore consists almost entirely of pyrolusite. Rhodochrosite has been encountered in tunnels of two mines 100 ft. or more from the outcrop. The gangue materials associated with the ore are silica and calcite. Locally gypsum is found and in one mine celestite occurs as small lenses.

The Green River manganese deposits probably represent bedded lenses of manganiferous carbonates including rhodochrosite, deposited immediately after the underlying sandstone.

At Shumla, Tex.,[39] manganese ore is found in association with Lower Cretaceous limestones. The ore consists of wad and pyrolusite associated with clay in irregular depressions and cavities on the surface of the Buda and Edwards limestones (Comanchean). The localization of the ore is believed to be due to the solution of the limestones along joints and fractures, and the filling of these open spaces by clays and associated manganese oxides. In places, the accumulation has taken place in caverns or other large underground-solution cavities where joints were abundant and where underground-water circulation was active. The manganese is believed to have been concentrated from a disseminated condition in the overlying rocks. Iron oxide is abundantly associated with the manganese oxides.

*Manganese in Tertiary Rocks.*—Deposits associated with Tertiary volcanic and intrusive rocks are common in many places in western United States. By far the most widely distributed are those occurring

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[34] References are to bibliography at end of paper

as veins and replacements in rhyolitic, andesitic, basaltic, and other lavas and tuffs of Tertiary and, probably in a few places, of Quaternary age. The most important, commercially, of the Tertiary ores are vein deposits associated with intrusive monzonite, aplite, and quartz porphyry at Butte, Mont

Manganese ores are found in Butte[29] in groups of veins situated north, west, and south of the main copper-bearing area. These are the veins that were formerly worked for silver and some of which recently have yielded much zinc. Some of the veins near the copper-bearing area also contain considerable copper. The principal vein systems in which manganese ore is found are the Rainbow lode north of the copper-bearing area and the Black Chief lode southwest of this area. The oxidized portions of the veins extend to depths of from 20 to 200 ft. (6 to 60 m.) and consist of a mixture of various manganese oxides, such as pyrolusite, psilomelane, braunite, and wad, associated with quartz. In the unoxidized parts of the veins, rhodochrosite, rhodonite, siderite, quartz, and various sulfides, sphalerite, galena, pyrite, and others, are found. In most of the veins rhodonite and rhodochrosite are intermixed and the material is of no value as an ore of manganese but locally, as in the Emma mine on the Black Chief lode, rhodochrosite is abundant and pure. The ores were deposited by manganese-bearing solutions that originated from deep-seated sources and followed the intrusion of the inclosing rocks. The manganiferous silver-zinc-bearing lodes appear to have been formed at the same time as the copper-bearing lodes, but probably under slightly different conditions of temperature.

In the Lake Creek district, Ore.,[33] and at Las Vegas, Nev.,[17] and Topock, Ariz., manganese oxides are found in bedded replacement deposits in tuff. In each district, there are interlayered beds of lavas and tuffs and the manganese deposits are generally found along a certain definite horizon in tuff beds. Whereas, in their present form, the ores are undoubtedly secondary and constitute replacements in the tuff, their distribution through a definite bed, parallel to the layering in the associated volcanic rocks, strongly suggest that the manganese was derived from that bed. Ores of this type are usually low grade and contain much unreplaced rock material.

In the Lake Creek district,[33] manganese oxide is associated mainly with a bed of brick-red basaltic tuff that is underlain and overlain by platy basalt flows. To a less extent, it is also found in gray andesitic tuff higher in the series. The oxide in the red tuff appears to be localized in many places at the top of the bed directly under the basalt. By solution and redeposition, however, manganese oxides have been formed lower down in the tuff, especially in areas where the overlying basalt

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[29] References are to bibliography at end of paper

has been removed by erosion. Near the surface, the manganese oxides are soft and crumbly; but below the surface, hard manganite is the principal mineral forming the deposits. Besides the silicate minerals present in the tuff, some hematite and limonite and a little gypsum, a zeolite and barite are found with the manganese oxides.

At Las Vegas,[17] manganese oxide is in beds of clay or tuff associated with rhyolite and volcanic breccia. The ore forms a flat-lying bed, varying in thickness up to 25 ft. (7 m.), which overlies greenish sand or tuff containing rhyolite fragments and is overlain by a bed of angular volcanic debris. The ore is soft, brownish black porous oxides, mainly wad. Calcite and fibrous gypsum are present in places. The oxides clearly replace tuffaceous material but the bedded form of the deposit suggests that manganese minerals may have been formed at this horizon at the time of the deposition of the associated volcanic rocks.

Near Topock, Ariz.,[17] deposits of the same type occur in tuff associated with rhyolite and volcanic breccia.

Veins, fissure fillings, and breccia deposits of manganese oxides are found in Tertiary lavas in many parts of the Southwest. Among the more important districts in which these ores have been exploited are the Aguila, Williams, Bouse, Topock, and Parker districts in western Arizona,[15] Greenlee and Graham counties, eastern Arizona,[15] the Ironwood Maria and Chocolate Mountains,[17] southern California,[17] and Socorro County and elsewhere in New Mexico.[20] The deposits of this group are valuable for their manganese content only; they do not contain precious or semi-precious metals, as in case of the veins and replacement deposits associated with Paleozic limestones near intrusive contacts, which are so common throughout the west. While occurring principally in Tertiary lavas, veins and breccia fillings are found also in other rocks, both igneous or sedimentary, that are associated with the Tertiary lavas. They have been found to a small extent in sedimentary and igneous rocks ranging in age from pre-Cambrian to Recent and in Quaternary basalt flows. Although most of the veins and breccia deposits are associated with rhyolite, deposits in andesite and basalt are also of common occurrence. The ores are simple vein and breccia fillings but fillings with associated replacement of inclosing rocks are also known. Shear zones consisting of ramifying veins are common. The ore shoots vary in size from mere veinlets to bodies many feet in width and of considerable longitudinal extent.

The manganese minerals include psilomelane, pyrolusite, and manganite. The common gangue minerals are calcite, quartz, iron oxides, and barite.

Recent bog deposits of manganese ore and mechanical accumulations

[17] References are to bibliography at end of paper.

of manganese-ore fragments and particles in stream gravels and surface wash, although common in many places in association with manganese-ore bodies of other types, rarely form important deposits by themselves.

In summarizing, it may be said that although manganese deposits occur in rocks that show a great range in age, many important deposits of the United States are distinctly related to certain geological horizons, notably the pre-Cambrian, Cambrian, Ordovician, and Jurassic. The deposits that show such associations contain: (1) Manganese minerals of sedimentary origin deposited as distinct beds or local lenses during periods of erosion in nearby lands; (2) manganese minerals formed by concentration in residual material of manganese originally disseminated through certain rock strata. The remaining deposits include: (1) Those formed in easily replaceable rocks such as limestone, as well as calcareous shale and sandstone, through fortuitous association of such rocks with igneous rocks that were a source of manganese-bearing solutions; (2) veins and breccia deposits, especially abundant in certain volcanic rocks, the manganese of which was derived from obscure sources by circulations set up following their appearance on the surface.

### *Structural Relations*

The common manganese minerals are deposited from solution in water or by interaction with rock materials, and the movement of the waters through the rocks is largely determined by their structural features. Therefore, the location, extent, and purity of the manganese minerals that make up deposits are largely determined by the rock types and their structural features. In any consideration of most deposits of manganese oxides, it is necessary to distinguish, on the one hand, between the relations of the manganese sulfides, carbonates, silicates, the parent minerals from which the oxides may have been derived, to the inclosing rocks, and, on the other hand, the relations of the oxides to the parent minerals as well as to the inclosing rocks.

As the structural relations of oxide orebodies are dependent upon those of the bodies of parent minerals it will be sufficient to mention briefly the textural and structural relations of the parent manganese minerals to the inclosing rocks. Although the extent and relations of manganese-bearing carbonates and silicates to the inclosing rocks are well known in several western mining districts, such as Leadville,[2] Coeur d'Alene,[37] and Butte,[29] many districts have yielded large quantities of oxides of manganese, the mineral source of which is still obscure. The work of Argall[2] has shown that, in the Leadville district, limestone and dolomite may be largely replaced by carbonates of iron and manganese

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[2] References are to bibliography at end of paper.

near faults or igneous contacts in a region where there has been extensive igneous activity. Similarly, in the Coeur d'Alene district, Ransome[37] has shown that shale and quartzite may be extensively replaced along fissures by manganiferous siderite. Probably similar replacement of carbonate and siliceous rocks by obscure carbonates of iron and manganese, as well as magnesium and calcium, will be shown by future study to be widespread, and offer an explanation of the source of manganese oxides that have long been known.

The occurrence of rhodochrosite and rhodonite in veins in regions affected by igneous rocks is widespread. Recent work, by Pardee,[35] in the Olympic Mountains, Wash., has shown the existence, in a large area, of lodes of an uncommon silicate of manganese that are either interbedded with or replace metamorphosed limestone and basic igneous flows. At present, there is reason for suspecting that many minor deposits of manganese oxides in regions of schistose rocks, such as eastern Virginia and South Carolina, have been derived from lenses of rhodonite in the schist.

Although bedded deposits of the manganese oxides that appear to have been laid down between successive sediments have been exploited in the Caucasus and Nicopol regions, Russia, and several districts in Chile, none definitely considered to belong to this type are known in the United States. Recent work by Jones[17] has shown that a number of deposits in western Arizona and southeastern Nevada possess striking resemblances to this type, but it appears that practically all the deposits of manganese oxides in the United States, except bog ores, have been deposited by circulating surface waters moving along pervious zones that are determined by differences in rock texture, joints, fissures, faults, or breccia zones.

Recent studies, by several Survey geologists,[11, 41] of ninety-nine deposits in the Valley of Virginia, in which manganese oxides are conspicuous, permitted the recognition of five structural types. In the commonest type, manganese oxides form irregular masses in the clay residual from dolomite where underlying gently dipping impervious quartzite meets a piedmont plain. In these deposits, the manganese probably originally disseminated through certain zones in the dolomite appears to have been deposited as oxides where the waters on meeting the plain were temporarily stagnant. The second type, though less common, yields the largest bodies of ore, such as that worked in the Crimora mine. The oxides are concentrated near the end of a gently dipping trough of impervious quartzite where it emerges from a piedmont plain. In two other types manganese oxides are deposited in clay along an extensive fault and in the quartzite breccia along minor displacements. These types are not

[37] References are to bibliography at end of paper

only common in Virginia and other eastern states, but are the most common in several western states, such as Arizona[17] and New Mexico.[20] Where the manganese has been derived from masses of manganese carbonates, or silicates localized along faults or fissures, the oxides appear to remain in the fissure; but, locally, solutions percolate along fractures into the neighboring wall rocks and by replacing the rock minerals, convert it to a material rich enough in manganese to ship. Evidence of rather complete replacement of many lithologic varieties of rock by manganese oxides has been obtained during the progress of recent work. Limestone and dolomite, both pure and siliceous, quartzite, sandstone, shale, and chert, as well as the clays derived from them and their common metamorphic equivalents, many varieties of acidic and basic igneous rocks and their tuffaceous phases are replaceable by manganese oxides in solution in surface waters.

The fifth structural type in which manganese oxides are deposited in stream sediments or alluvial material, is known in Virginia, and elsewhere in the Appalachian region, as well as in Nevada. In contradistinction to the bedded deposits, the manganese oxides in this type appear to have been introduced after the sediments were in place. The deposits of manganese oxides in bogs may be regarded as intermediate between these two types.

### *Genesis of Deposits*

Studies of the genesis of manganese deposits, in contrast with those of many other common metals, encounter unusual problems that can best be handled by recognizing two types of deposits, primary and secondary. Among those of the first type may be included all those bodies of sulfide, carbonate, silicate, and probably phosphate minerals, the processes and sites of deposition of which are related to zones below the surface of the earth, whereas in the second type may be included those bodies of oxides that are formed where solutions containing manganese have access to free oxygen and are therefore formed in the surficial zone. Doubtless, small quantities of manganiferous carbonates may be deposited under peculiar local conditions with those of the second type. It will be apparent that secondary deposits commonly develop near primary deposits when these are brought near the surface by erosion. Confusion may arise, however, if the features of the nearby or superposed types are not clearly separated. Further, by contrast, many secondary deposits are known and others have been extensively explored without yielding satisfactory evidence concerning the minerals and localities from which the oxides were derived.

Most manganese deposits in metamorphic rocks present problems that are peculiar to those rocks that have been deeply buried. Both

[17] References are to bibliography at end of paper

primary and secondary deposits may be deeply buried after they have assumed definite form, and here and there it is difficult to distinguish the features that should be attributed to the successive stages in their development. Although manganese deposits occur in metamorphic rocks in California, Washington, Texas, and the Piedmont region of the Atlantic coast, only in the first-named state have they been the scene of extensive exploration.

*Primary Deposits.*—The minerals that commonly make up the primary deposits include the following, arranged approximately in the order of their importance: rhodochrosite, siderite, and the related group of carbonates of manganese, iron, calcium, and magnesium; rhodonite; spessartite; tephroite and other silicates; and alabandite, the sulfide. Several of the lower oxides, including hausmannite ($Mn_3O_4$) and manganosite ($MnO$), are known to occur with manganiferous carbonates in primary deposits. Franklinite, the unusual manganese, iron, zinc oxide, as well as manganosite, occurs with rhodonite in the calcite gangue of the Franklin Furnace, N. J., deposits. Accessory minerals in such deposits include quartz and many metallic sulfides and sulfarsenides.

Although rhodochrosite, manganiferous siderite, and the other manganese carbonates are deposited in fissures in many western districts, notably the Butte district, Mont., and the San Juan region, Colo., they commonly replace the wall rock adjacent to fissures, faults, and igneous contacts. The parent mineral of the oxide masses at Philipsburg, Mont., is rhodochrosite, which generally replaces magnesian limestone At Leadville, and Redcliff, Colo., the parent mineral of the manganese and iron oxides is manganiferous siderite, which replaces limestone. In the Coeur d'Alene region, Ida., manganiferous siderite replaces slate and quartzite, but important bodies of oxides have not been formed by weathering. Although alabandite and rhodochrosite are known in the Tombstone district, Ariz., the quantity appears to be inadequate to account for the large bodies of oxide ore, so that other obscure manganese minerals may be present. The manganese of all of these deposits and many others having similar minerals and structural associations appears to have been a constituent of waters of deep-seated origin, that are given off during periods of igneous activity.

The silicates, spessartite and tephroite, are rare in the United States; the former is recorded from pegmatite dikes in Amelia Co., Va., and both are found in layers in banded gneiss in Llano and Mason Co., Tex. Tephroite occurs in the deposits of Franklin Furnace, N. J.

The layers of manganiferous carbonates and silicates that are interbedded with sediments present interesting problems. In some regions, notably Newfoundland,* both pure and impure persistent layers are

* N. C Dale: The Cambrian Manganese Deposits of Conception and Trinity Bays, Newfoundland. *Proc* Am. Phil Soc. (1915) **54**, 371–456.

interbedded with marine sediments. The cherty iron carbonate of the Cuyuna Range, Minn.,[7,26] forms lenses in beds whose origin is obscure, and the Cason shale, which contains the carbonate concretions at Batesville, Ark.,[25] is probably a marine sediment. As the Cason shale has not suffered much metamorphism and it is known from studies of the Indian manganese deposits that manganese oxides survive considerable metamorphism, it would appear that the manganese carbonate was deposited contemporaneously with other carbonates and clay. Investigation of the rhodochrosite and rhodonite lenses of California and the silicate lenses of Washington has not proceeded to the point where satisfactory conclusions concerning their origin can be stated.

*Secondary Deposits.*—Secondary deposits of manganese commonly contain psilomelane, manganite, pyrolusite, braunite, and wad. Both psilomelane and wad commonly contain iron, and an isomorphous series that ranges from hydrous oxide of manganese to hydrous oxide of iron appears to exist. Locally, distinct hydrous oxides of each metal are intimately mixed

The weathering of most primary manganese deposits yields superposed bodies of manganese oxides, but locally the oxides migrate into the wall rocks and nearby surface debris and more rarely even into the sediments of nearby streams, and apparently to the sea, for at several localities the oxides of manganese are deposited simultaneously with common marine sediments and chemical precipitates. The nodules of manganese and iron oxides found in the depths of the sea remote from existing lands appear to present a unique problem. It should be stated here, however, that the manganese oxides of many deposits that appear to be interbedded with sediments or volcanic flows have been found with further exploration and study to be the weathered outcrops of lenses or beds of carbonate minerals or to have been formed by replacement of pervious zones of bedded materials. Recent work indicates that manganese oxides only rarely are laid down as sediments.

The scope of this paper does not permit an exhaustive review of the chemical problems of the solution and deposition of manganese minerals. Attention should be called, however, to a recent paper by V. Vincent* that bears closely on an important problem in the formation of secondary manganese deposits. The manganese of most of the manganese minerals in primary deposits is in the state of manganous oxide (MnO), although several minerals contain manganese sesquioxide ($Mn_2O_3$). It is known that manganese goes into solution and is redeposited in a higher state of oxidation. If the higher oxides are relatively soluble in the solutions commonly found in the zone of weathering it would be possible

[7] References are to bibliography at end of paper

* V. Vincent. *Comptes Rendus* (1916) **162**, 259–261

to have successive solution and redeposition of manganese oxides that would lead to a zone of secondary enrichment, similar to that found in copper and silver deposits. If the higher oxides are relatively insoluble in these solutions, however, they should tend to accumulate near the surface as the nearby weathered rock products are removed. According to Vincent, at 12° C., 62.5 mg. of manganese in the form of manganese carbonate are dissolved by a liter of water saturated by carbonic acid The solubilities of the oxides in a saturated solution of carbonic acid at 13° C. represented by milligrams of manganese per liter follow: manganous oxide, 29.1 mg.; manganese sesquioxide, 4.6 mg.; manganese dioxide, 1.5 mg. The order of solubility is therefore

$$MnCO_3 : MnO : Mn_2O_3 : MnO_2 = 42 \cdot 20 : 3 : 1$$

Further, manganese carbonate and manganous oxide are soluble in humus but manganese sesquioxide and dioxide are not appreciably soluble. The data indicate that the solubility of the higher oxides of manganese ranges from 1.5 to 4.6 parts per million and that they are therefore highly insoluble. There is reason for suspecting that unless strong acids, such as sulfuric, or materials capable of reducing the oxides to a lower state of oxidation, are available the higher oxides of manganese will tend to persist in the mantle of weathered rock and not be carried downward to form enriched zones. In the eastern states, float containing manganese minerals is commonly found in areas underlain by deposits. It is also the impression of the geologists who have examined most of the western deposits that although some deposits in carbonate rocks do not outcrop, the outcrops that do exist give a dependable impression of the horizontal extent of the deposits and yield material (oxides) that is as good as or better than that which underlies it.

With reference to the mode of deposition of the manganese oxides, it may be tentatively stated that only rarely is there oxidation *in situ*, and therefore, pseudomorphous replacement of the parent mineral. It is most common in deposits of silicate minerals, where there is loss of silica in solution. In most places, and most conspicuously where there are carbonate minerals in limestone and dolomite, the oxides migrate locally and are either deposited in open spaces or replace clay or the nearby wall rocks. The character of manganese-bearing solutions appears to be such that no rock type is immune to replacement by oxides, for specimens show more or less complete replacement of limestone, shale, and sandstone, as well as acidic and basic igneous rocks and their fragmental equivalents. Abundant evidence is available from deposits in the eastern states to indicate that although oxides are locally deposited in open spaces, most of the nodules of oxides are formed by replacement of clay in which they are imbedded. The Batesville district yields excellent manganese oxide pseudomorphs of brachiopods.

There is lack of uniformity in the conclusions reached in several districts concerning the tendency toward segregation of iron and manganese oxides that have been derived from the same or nearby minerals. Van Hise and Leith* state that in the Lake Superior districts the upper parts of the iron deposits contain more manganese than the lower. In the eastern states, however, there are several instances, notably the Mt. Torry mine, Augusta Co., Va., and the Kendall and Flick mine, Elkton, Rockingham Co., Va., where material mined near the surface was slightly manganiferous limonite whereas deeper exploration, 100 to 250 ft. below the surface, yielded highly manganiferous limonite or slightly ferruginous manganese ore. Similar instances are reported in Tennessee.[40]

The present state of investigations concerning the relation of the oxides to water level does not permit a statement of conclusions. In many districts, notably those in the eastern states, where water level commonly ranges from 10 to 80 ft. below the surface, large bodies of oxides are mined 180 to 220 ft. below average water level. In these, as well as several western states where oxides locally persist several hundred feet below water level, close examination of accumulated information concerning the recent physiographic history, as well as further study, may be necessary to determine whether the oxides were actually deposited below water level or there have been oscillations of water level since the oxides were deposited.

### *Suggestions Regarding Examination and Exploitation of Manganese Deposits*

Those who seek to exploit ore deposits generally have two questions in mind. first, where are the favorable areas for deposits; second, what features should govern the effective exploitation of the deposit when it is found. Concerning the first question, it is only possible to make most general statements. Certain large areas in the United States that are underlain by similar groups of rocks and have been affected by similar erosional processes, such as the belt of valleys and ridges east of the Alleghany front in Virginia, Tennessee, and Georgia,[5] contain many deposits. Likewise, the belts of outcrops of certain rocks, such as the Franciscan cherts in California, contain many deposits. Little help is to be obtained from such conclusions, however, for such belts extend for hundreds of miles and most deposits of manganese, as well as other ores, are measured by acres and fractions thereof. Recent investigations of deposits in Virginia[11] warranted the conclusion that, whereas most of the deposits occurred along the narrow belt where the remnants of an

* C. R Van Hise and C. K. Leith: Geology of the Lake Superior Region U. S Geol Survey *Monograph* 52 (1911) 488, 560.

[40] References are to bibliography at end of paper.

extensive plain merge with the Blue Ridge, the largest deposits occur where the underlying rocks form troughs under this belt. It can be stated with considerable assurance that in estimating the merit of areas in this region, engineers and geologists would do well to examine closely the structure of the nearby rocks and the position of areas with respect to the plain. Since few of the deposits in this region show conspicuous outcrops and yield significant float, this conclusion may aid in selecting areas for prospecting.

In general, it may be stated that the concentrated masses of certain manganese minerals that are called ore deposits, in some places occur in certain definite stratigraphic zones, elsewhere with distinct structural associations, such as in troughs, faults, or related breccia zones, as well as with fairly definite relations to erosion surfaces. Only when unexplored areas have been carefully examined with respect to the possible significance of these three branches of geologic research can the most dependable conclusions be reached.

In some respects, it is fortunate that most manganese deposits outcrop at the surface, and that, as a rule, the outcrop material is as good as or better than any that will be encountered below the surface. Thus, the operator may quickly determine with considerable assurance whether he can ship acceptable material at a profit. Exceptions to this rule are found in regions where the manganese minerals are associated with residual clays, for although locally float material may be found, it may be misleading or absent. Several large manganese deposits in Virginia are covered by barren wash, and the ferruginous manganese deposits of Minnesota are covered by a thick blanket of glacial drift. Also, many deposits of manganese ore in the Valley of Virginia underlie deposits of iron ore, the iron oxides having been relatively more insoluble in the underground waters than the manganese oxides.

Where the manganese oxides that make up the marketable material of a deposit have been derived from localized masses of the various carbonate and silicate minerals that contain less manganese, the persistence of the oxides bears a fairly definite relation to the position of the groundwater table. Careful study of some regions warrants the conclusion that the water table is higher than it has been in times past and that therefore oxides and other minerals, which normally only persist to water level, are found much deeper than the present level. It may be neccessary to consider the possible bearing of disturbances of the water table due to vertical oscillations of the land surface as well as to prolonged pumping coincident with deep exploration.

In some arid regions with considerable surface relief, such as parts of California, the ground-water table may be deep and the unaltered carbonate and silicates may remain near the surface. In the Tombstone district, manganese oxides persist 630 ft. (192 m.) below the surface and

an unknown short distance below water level; whereas in the Bisbee district,[38] they persist 1300 ft. (396 m.) below the surface, also slightly below water level. At Leadville, conditions are not simple due to complicated faulting, a number of shafts are dry and yield oxidized ores 400 to 500 ft. (121 to 152 m.) below the surface; but in the Penrose mine, oxides persist 810 ft. (246 m.) below the surface and about 600 ft. (182 m.) below the recent water level. Explorations at only three mines in the Philipsburg district have extended below water level, which appears to range from 150 to 250 ft. (45 to 76 m.) below the surface at the outcrop, and oxides persist below this depth.

In the eastern states, where there is moderate rainfall and low surface relief and most of the manganese ores are in residual clay, the water table ranges from 10 to 80 ft. (3 to 24 m.) below the surface. Only a few explorations on manganese deposits extend more than 300 ft. (91 m.) below the surface but oxide ores have been mined at several places in Virginia from 240 to 260 ft. (73 to 79 m.) below the surface and 180 to 220 ft. (54 to 67 m.) level. These data may serve as a guide in exploration.

It is highly essential that the mineralogy and texture of the ores of most deposits, and especially of the oxides, be carefully studied as close observation of these features sooner or later will permit conclusions as to the mode of introduction of the minerals into their present positions. As the manganese oxides are generally the only minerals that may be marketed, it is important that evidence of the minerals from which the oxides are derived should be obtained as promptly as possible, during the progress of explorations. Chemical analyses of the material shipped will show the quantity of silica and other oxides present, but do not show whether the silica is present as quartz grains or veinlets or as the minerals rhodonite, spessartite, or other silicates. Thus, according to Pardee,[29] it appears to be possible by a study of the texture and mineralogy of the oxide ores at Butte to distinguish those that were derived from rhodonite from those derived from mixtures of rhodochrosite and quartz. Since rhodochrosite may be separated from quartz by milling processes and marketed to advantage and rhodonite has no value, the observation may serve a good purpose.

Summarizing, it is safe to infer the extent of a deposit of manganese oxides in proportion as it is possible to demonstrate either, on the one hand, an adequate or concentrated nearby source of manganese in the form of sulfides, carbonates, or silicates that might be altered or further concentrated by ordinary processes, or on the other hand, a rather inadequate or disseminated source of manganese concentrated by a combination of extraordinarily effective processes. If a concentrated source of

[38] References are to bibliography at end of paper.

manganese carbonates or silicates can be demonstrated and processes have been favorable, a large deposit may be expected. If, however, only a disseminated source of such minerals can be proved or the source is obscure, one should be cautious in assuming that a deposit of oxides is extensive, unless there is good reason for believing that unusually effective processes of concentration have been in operation. The study of the "border zones" between oxides of manganese and waste or wall rocks has the highest importance.

## Production

*Classes of Ore.*—In the preparation of statistical reports concerning the production of manganiferous materials, the U. S. Geological survey now recognizes five classes according to composition: Manganese ore, ferruginous-manganese ore, manganiferous-iron ore, manganiferous-silver ore, manganiferous-zinc residuum.

Manganese ores are those that contain 35 or more per cent. manganese. Until 1918, only material with 40 per cent. or more manganese was regarded as manganese ore,[13] on the assumption that 80 per cent. ferromanganese could not be made from lower grade material unmixed with that of higher grades. Even though the reduction of the manganese content in ferromanganese from 80 to 70 and 60 per cent. was a war expedient to provide a market for larger quantities of domestic ore, there is basis for believing that, in favored localities, material ranging from 35 to 40 per cent. manganese will continue to find a market with the makers of ferromanganese. Although it is possible for material with 35 per cent. manganese to contain 40 per cent. silica, even recently very little ore with more than 25 per cent. silica has been marketed and the average manganese content of all domestic ores produced during 1918 was about 39 per cent. manganese and 13 per cent. silica. So far as is now known, only a few deposits, such as several in the Leadville district, yield material with more than 35 per cent. manganese and more than 10 per cent. iron and it seems appropriate to recognize this percentage as the dividing line between ferruginous-manganese ore and manganese ore.

Ferruginous-manganese ores are those that, in the natural state, contain from 10 to 35 per cent. manganese. Although during the shortage of manganese ores in 1918, small quantities of material with this range of manganese and 20 to 30 per cent. silica and 5 to 15 per cent. iron were marketed, most of this ore contained from 30 to 38 per cent. iron and 10 to 20 per cent. silica. The product of most of the Leadville mines contains from 20 to 30 per cent. manganese, 10 to 15 per cent. silica, and 10 to 20 per cent. iron. The change of the lower limit of manganese in

[13] References are to bibliography at end of paper.

ferruginous-manganese ore to 10 from 15 per cent., as used in recent publications by the United States Geological survey, is based on the conclusion that spiegeleisen with 16 per cent. manganese can be made from ore with as low as 10 per cent. manganese, and that this alloy will continue to be made and used in the steel industry.

Manganiferous-iron ores are considered to be those that, in the natural state, contain from 5 to 10 per cent. manganese and more than 35 per cent. iron.

Manganiferous-silver ores are those that contain more than 5 per cent manganese, generally more than 10 per cent. iron, and sufficient silver to make it more profitable to use them as a flux in lead or copper smelting than as a source from which to recover metallic manganese or iron. Such ores are only produced by mines in Arizona, Colorado, and Nevada at the present time

Manganiferous-zinc residuum is the product obtained from franklinite ore from New Jersey by calcining it with coal. A large part of the annual production is used in making spiegeleisen.

In Table 2, a summary of the domestic production of manganese and manganiferous ores during recent years is presented. It contains detailed statistical data for only manganese ore, ferruginous-manganese ore, and manganiferous-iron ore; in addition, recent annual production of zinc residuum is presented.

The most impressive feature of the statement of production given in Table 2 is the extraordinary increase in rate of shipment as well as the total quantity of ore shipped within a short period from new sources in the western states in contrast with that from the previously well known sources in the eastern states. This is largely due to the fact that the ores of most of the eastern districts are distributed through residual clay and other waste, which must be washed away from the ore in washers and jigs. As a result, the rate of shipment is practically determined by the capacity of the mills at the respective deposits. There can be scarcely any doubt that the part of the investments in the domestic industry, during 1917 and 1918, used in installing milling equipment was larger than that used in making explorations. Thus, aside from seasonal variations due to the local climate, the shipments from Virginia, Tennessee, Georgia, and Arkansas during 1916, 1917, and 1918 were much less than many would have expected from the records of previous production.

The extraordinary output of the new deposits in the Philipsburg district,[45] has been the principal contribution of the domestic industry. The first shipments were made, in 1916, from several deposits that were known from the earlier period of activity when deposits of gold and silver were exploited, but the search quickly became intensive. By

[45] References are to bibliography at end of paper.

TABLE 2.—*Domestic Production of Manganese Ores*

| Name of State | 1913, 12 Months | 1914, 12 Months | 1915, 12 Months | 1916, 12 Months | 1917 6 Months | 1917 Third Quarter | 1917 Fourth Quarter | 1917 12 Months | 1918 First Quarter | 1918 Second Quarter | 1918 Third Quarter |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Alabama | | | | | | | | | | | |
| 35+ per cent ... | | | 200 | [a] | 40 | 50 | 174 | 264 | | 53 | 358 |
| 10–35 per cent. .... | | | | | | | | | | 85 | 222 |
| Arizona | | | | | | | | | | | |
| 35+ per cent. ... | | | 339 | 3,060 | 1,735 | 7,734 | 5,333 | 14,802 | 4,375 | 5,497 | 4,086 |
| 10–35 per cent . | | | 1,452 | 7,392 | 5,400 | 8,108 | 5,545 | 19,053 | 37 | | 1,096 |
| 5–10 per cent. . . | | | | | | 9,429 | 5,967 | 15,396 | | | |
| Arkansas | | | | | | | | | | | |
| 35+ per cent. . | | | 1,288 | 6,318 | 4,361 | 875 | 4,904 | 10,140 | 1,686 | 2,403 | 2,776 |
| 10–35 per cent | 9,650 | 1,970 | 2,655 | 3,682 | 3,994 | 3,704 | 1,402 | 9,100 | 1,440 | 2,094 | 4,214 |
| California | | | | | | | | | | | |
| 35+ per cent . . | | 501 | 2,563 | 6,136 | 3,092 | 3,470 | 7,634 | 14,196 | 3,137 | 7,464 | 6,919 |
| 10–35 per cent. . | | | | 139 | 50 | 40 | 62 | 152 | 31 | | |
| | | | | | | 60[b] | | | | | |
| Colorado | | | | | | | | | | | |
| 35+ per cent. | | | 150 | 110 | [b] | | [b] | 114 | 33[b] | 34[b] | 1,150[b] |
| 10–35 per cent . | 41,862 | 33,282 | 26,755 | 98,255 | 57,109[b] | 16,557[b] | 30,849[b] | 118,481 | 31,000[b] | 29,804[b] | 19,859[b] |
| 5–10 per cent . . | 7,891 | 6,599 | 4,166 | 8,858 | [b] | [b] | [b] | 6,380 | [b] | [b] | [b] |
| Georgia | | | | | | | | | | | |
| 35+ per cent... . | | | 3,168 | [a] | 56 | 2,174 | 1,384 | 3,614 | 450 | 1,513 | 3,119 |
| 10–35 per cent. | | | 676 | [a] | | 9,640 | 3,641 | 13,281 | 3,840 | 1,311 | 2,403 |
| 5–10 per cent.. | | | | | | | | | | 300 | 784 |
| Michigan | | | | | | | | | | | |
| 10–35 per cent.. .. | | | | | 19,000 | 10,866 | 22,332 | 52,198 | | 5,534 | 11,406 |
| Minnesota | | | | | | | | | | | |
| 10–35 per cent .. .. | 26,200 | 55,192 | 42,973 | 193,257 | 216,131 | 97,190 | 46,252 | 359,573 | 1,431 | 221,901 | 229,425 |
| 5–10 per cent .. . | | | | 47,146 | 2,710 | 47,790 | 41,090 | 91,590 | | 82,309 | 99,991 |
| Montana | | | | | | | | | | | |
| 35+ per cent... | | | | 6,418 | 14,386 | 27,649 | 19,074 | 61,109 | 36,525 | 50,813 | 60,145 |
| 10–35 per cent... . | | | | | 873 | | 695 | 1,568 | | | |
| Nevada | | | | | | | | | | | |
| 35+ per cent... ... | | | | [a] | | 1,000 | 2,450 | 3,450 | 3,653 | 8,933 | 4,888 |
| 10–35 per cent..... | | | 104,498 | 122,429 | 60 | 86,516 | 36,296 | 122,872 | [b] | 27[b] | 141[b] |

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| New Jersey[c] | 102,239 | 100,198 | 159,318 | 198,817 | | | | 155,332 | | | |
| New Mexico | | | | | | | | | | | |
| 35+per cent. | | | | [a] | | 658 | 1,945 | 2,603 | 1,396 | 645 | 230 |
| 10–35 per cent | | | | 16,068 | 12,000 | 3,100 | 800 | 15,900 | 2,405 | 4,573 | 11,157 |
| North Carolina | | | | | | | | | | | |
| 35+per cent. | | | | | | | 102 | 102 | 76 | 168 | 71 |
| Tennessee | | | | | | | | | | | |
| 35+per cent | | | 150 | 429 | 506 | 657 | 833 | 1,996 | 451 | 389 | 1,536 |
| 10–35 per cent | | | | | | 28 | 55 | 83 | 231 | 728 | 839 |
| Texas | | | | | | | | | | | |
| 35+per cent . | | | [a] | [a] | | 25 | | 25 | | 207 | 101 |
| 10–35 per cent | | | | | | 110 | 25 | 135 | | | |
| Utah | | | | | | | | | | | |
| 35+per cent. | | | [a] | 1,282 | 690 | 1,490 | 2,015 | 4,195 | 2,011 | 1,690 | 938 |
| 10–35 per cent. | | | | [a] | 100 | | 36 | 136 | | | |
| Virginia | | | | | | | | | | | |
| 35+per cent . | 4,048 | 1,724 | 1,620 | 4,417 | 3,975 | 5,140 | 3,245 | 12,360 | 1,758 | 2,522 | 4,331 |
| 10–35 per cent | | 1,222 | 1,944 | 7,213 | 2,260 | 7,208 | 8,109 | 17,577 | 1,682 | 3,524 | 2,418 |
| 5–10 per cent | | | | 30,487 | | | 16,819 | 16,819 | 1,534 | 2,090 | 1,178 |
| Wisconsin | | | | | | | | | | | |
| 5–10 per cent | | | 10,616 | 3,982 | [d] | [d] | [d] | [d] | [d] | [d] | [d] |
| Total, U. S A[e] | | | | | | | | | | | |
| 35+per cent.. | 4,048 | 2,635 | 9,613 | 31,474 | 28,841[f] | 51,282[f] | 49,228[f] | 129,405 | 55,682[g] | 82,481[g] | 90,738[g] |
| 10–35 per cent | 51,512 | 91,666 | 180,953 | 453,853 | 316,977[f] | 243,067[f] | 156,749[f] | 730,759 | 42,947[g] | 269,581[g] | 283,180[g] |
| 5–10 per cent.. | 7,891 | 6,599 | 14,782 | 90,473 | 2,710[f] | 57,219[f] | 63,876[f] | 130,185 | 1,534[g] | 84,699[g] | 101,953[g] |

[a] U. S Geological Survey not at liberty to publish this tonnage, but it is included in total for United States.
[b] Not including ore shipped for fluxing which was not reported quarterly.
[c] Zinc residuum.
[d] For 1917 and 1918, shipments contained less than 5 per cent. manganese, natural basis.
[e] Including shipments of ore from New Jersey, Oregon, Pennsylvania, South Carolina, South Dakota, Washington, and Wyoming, not shown above.
[f] Not including ore shipped from Colorado for fluxing which was not reported quarterly.
[g] Not including ore shipped from Colorado and Nevada for fluxing.

August, 1917, 17 deposits of considerable size were explored and in September, 1918, 29 deposits with more than 500 tons reserve were developed. The marketing was temporarily hindered by the unwillingness of consumers to accept the ores with 15 to 20 per cent. silica, but this disappeared in 1917 and 1918 when the shortage of better material became acute. The ease with which the material may be mined makes the reserves of the Philipsburg district the most important source in the country in the event of an emergency.

Other western mining districts that had been previously explored for other metals have made important contributions of high-grade manganese ore; Bisbee and Tombstone, Ariz.,[38] Lake Valley, N. M.,[20] and Tintic and Eureka, Utah.[32] Although considerable ore probably remains in the Bisbee district, the acceptable material from the other districts is apparently nearly exhausted. The Leadville district, which for many years has regularly shipped large quantities of silver-bearing manganese and iron oxide ores for use as a flux in lead smelting, was vigorously explored and shipments during 1916, 1917, and 1918 were more than twice those during previous recent years.

One of the most interesting new sources of ore during 1917 and 1918 is the Emma mine in the Butte district,[29] Mont., which has yielded about 56,000 tons of rhodochrosite, an ore that, so far as can be found, was never before smelted to ferromanganese in the United States. The veins of rhodonite and rhodochrosite of the Butte district have been well known for many years; and although the Emma mine has recently been exploited for zinc ore, the bodies of rhodochrosite were found to be pure enough to permit shipment to eastern makers of ferromanganese. The other known bodies of pure rhodochrosite are not large but, as recent milling experiments in the district have shown that it may be possible to separate rhodochrosite from the mixtures with quartz, a large quantity of carbonate material may be recoverable from the district. Between March and September, 1918, the Anaconda Copper Mining Co., having surplus electric power at Great Falls, erected five electric furnaces to reduce the carbonate ore from the Emma mine. The plant was operated for only a month when the surplus of ferromanganese and prospective decline in price made operations hazardous and it was closed.

Several newly discovered deposits in regions that, before 1916, were not known to contain manganese deposits, however, have made important contributions but not as large as many who think the West is still scarcely scratched might have expected. The most conspicuous new deposit is the Three Kids, 15 mi. east of Las Vegas, Clark Co., Nev.,[16] which was discovered in 1917 and until December, 1918, had yielded 15,000 tons of material with 35 to 42 per cent. manganese. Other new deposits from

[38] References are to bibliography at end of paper.

which more than 500 tons each had been shipped, by the end of 1918, include several groups in eastern Riverside and Imperial Co., Cal.; several groups in Maricopa, and Santa Cruz Co., Ariz.; the Myers mine near Magdalena, Socorro Co., N. M. and several near Ely, Nev. Several large deposits of manganese-bearing siliceous materials have been found in the Williams River district, Ariz., and of manganese-bearing calcareous material near Shumla, Tex., but only trial shipments have been made.

It is a coincidence that the low-grade deposits of the Cuyuna Range, Minn.,[7] were explored several years before the outbreak of the European war and were available as an important source of material when the shortage in manganese alloys began to develop. Since 1914, the district has been systematically explored for manganiferous ores, and shipments have increased at a rapid rate, although there is a seasonal variation on account of the effect of the climate on transportation. Until 1916, most of the shipments were material with less than 0.125 per cent. phosphorus and more than 12 per cent. silica and were used in making spiegeleisen; but shipments of material with more than 0.125 per cent. phosphorus have steadily increased and there is a prospect that a permanent market may have been established for such material.

Remoteness from all except a single small outlet, that of Pueblo, has hindered the exploitation of the ferruginous-manganese ore deposits of the Silver City district, N. M.[47] If the material were near the steel-producing centers of the east, it would be extensively used. The decline in shipments through 1917 was caused by excessive accumulation of stocks at Pueblo and the increase during 1918 to shipments to Chicago, and other eastern markets.

The shortage of ships, which began in 1914 and was most acute early in 1918, caused a rise in price of desirable grades of manganese ore (more than 45 per cent manganese) to figures that were about five times those prevailing in 1913 and 1914. In the same period the price of ferromanganese rose to a maximum of ten times the price just preceding the war and then became rather stationary at about seven times the old price. As prices of ore rose, the shipments of desirable material from domestic sources increased only slightly, and there was therefore created by enterprising alloy makers in the latter part of 1916 a market for less desirable manganese ore (35 to 45 per cent. manganese, and 8 to 20 per cent. silica). The extraordinary increase in rate of production shown in Table 2 began when alloy makers were willing to accept the less desirable material that most of the domestic deposits were only capable of supplying.

The increase in rate of production of ferruginous-manganese ore has been large but scarcely comparable to that of high-grade ore. Any great increase in consumption of such ore has depended on the capacity of steel works to enlarge the use of spiegeleisen, which might be made

[7] References are to bibliography at end of paper.

TABLE 3.—*Composition of Manganese Ores (35 Per Cent.+ Manganese), Natural Basis*

| State | Remarks | Quantity | Manganese Per Cent | Iron, Per Cent. | Silica, Per Cent | Phosphorus, Per Cent | Water, Per Cent | Authority |
|---|---|---|---|---|---|---|---|---|
| Alabama | | | | | | | | |
| Near Altoona, Etowah Co | Average | [a]518 tons | 45.75 | 1.55 | 7.68 | 0.37 | 10.56 | Southern Manganese Corpn |
| Arizona | | | | | | | | |
| Bisbee dist, C & A Mining Co | Best car 1916 | 50 tons | 53.99 | 0.61 | 14.57 | 0.018 | 1.32 | Miami Metals Co |
| Bisbee dist., C. & A Mining Co | Best car 1917 | 49 tons | 49.99 | 1.63 | 12.35 | 0.046 | 0.35 | Miami Metals Co |
| Bisbee district, Higgins mine | Best car 1916 | 38 tons | 47.57 | 2.39 | 17.35 | 0.053 | 3.99 | Miami Metals Co |
| Bisbee dist, Higgins mine | Best car 1917 | 45 tons | 47.18 | 7.96 | 12.44 | 0.055 | 0.67 | Miami Metals Co |
| Bisbee dist, Copper Queen | Best car 1917 | 43 tons | 47.68 | 1.84 | 8.35 | 0.040 | 1.23 | Miami Metals Co |
| Bisbee dist, Shattuck-Arizona | Best car 1917 | 29 tons | 51.53 | 2.56 | 7.71 | 0.045 | 1.24 | Miami Metals Co |
| Bisbee dist . . . . . | Average 1917 | [b]8590 tons | 43.80 | 4.9 | 13.80 | | | Reports to U S G S by shippers |
| Various districts, largely Bisbee | Average of stocks, July, 1918 | [a]6600 tons | 41.20 | 6.31 | 13.09 | 0.059 | 1.96 | War Industries Board |
| Arkansas | | | | | | | | |
| Batesville district . | Average of stocks, July, 1918 | [a]1688 tons | 44.40 | 5.2 | 9.8 | 0.091 | 5.72 | War Industries Board |
| California | | | | | | | | |
| All districts, largely Livermore | Average 1917 | [b]12,146 tons | 45.40 | 0.9 | 13.1 | | | Reports to U S G S by shippers |
| All districts, largely Livermore | Average of stocks, July, 1918 | [a]6,221 tons | 41.60 | 1.51 | 13.6 | 0.059 | 4.6 | War Industries Board |
| Georgia | | | | | | | | |
| Various districts, largely Cartersville . . . . . . . . . . | Average of stocks, July, 1918 | [a]1,818 tons | 40.00 | 5.84 | 9.52 | 0.206 | 5.18 | War Industries Board |
| Montana | | | | | | | | |
| Butte district, Emma mine | Average | 16,376 tons | 36.5 | 2.13 | 7.05 | 0.048 | 2.3 | Miami Metals Co, Southern Mn Corpn |
| Philipsburg dist, Jackknife mine | Best car 1917 | 14 cars | 40.51 | 6.64 | 13.23 | 0.045 | 6.55 | Miami Metals Co |
| Philipsburg dist, Sharktown mine . . . . . . . . . | Best car 1917 | 118 cars | 46.68 | 1.43 | 13.66 | 0.053 | 8.79 | Miami Metals Co |
| Philipsburg dist, Coyle mine | Best car 1917 | 53 cars | 46.90 | 1.53 | 13.55 | 0.116 | 8.44 | Miami Metals Co |
| Philipsburg dist, Little Gem Elmore, Wingert No 2. . | Best car 1917 | 23 cars | 48.00 | 1.43 | 12.98 | 0.041 | 9.57 | Miami Metals Co |
| Philipsburg dist, True Fissure & Algonquin. . . . . . . . | Average | 193 cars | 44.72 | [c]2.00 | 15–16 | 0.023 | | Philipsburg Mining Co |

| | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| Philipsburg dist , All mines .. | Average 1917 | [b]56,780 tons | 42 8 | 2 00 | 16 80 | | | Reports to U S G S |
| Philipsburg dist , All mines .. | Average of stocks, July, 1918 | [a]45,834 tons | 39.55 | 1 82 | 17 12 | 0 068 | 8 77 | War Industries Board |
| Nevada | | | | | | | | |
| Various districts . | Average of stocks, July, 1918 | [a]2,343 tons | 39 1 | 1 86 | 10 6 | 0 022 | 5 13 | War Industries Board |
| Las Vegas Dist , Three Kids mine ........ | Average | 1,564 tons | 34 53 | 1 02 | 14 4 | 0.047 | 8.35 | Miami Metals Co |
| Tennessee | | | | | | | | |
| Various districts .. ... | Average of stocks, July, 1918 | [a]828 tons | 42 65 | 5.73 | 8 06 | 0 148 | 4 86 | War Industries Board |
| Utah | | | | | | | | |
| Various districts, largely Tintic and Green River .... .. | Average of stocks, July, 1918 | [a]456 tons | 40 00 | 1 90 | 8 60 | 0 083 | 5 04 | War Industries Board |
| Virginia | | | | | | | | |
| Various districts.... . .. | Average of stocks, July, 1918 | [a]3,189 tons | 40 00 | 5 42 | 14.10 | 0 132 | 7.06 | War Industries Board |
| United States | | | | | | | | |
| Various districts . .. | Average of stocks, July, 1918 | [a]91,149 tons | 39 4 | 2 47 | 13.41 | 0 067 | 6.15 | War Industries Board |
| Brazil ........ | Average of stocks, July, 1918 | [a]179,009 tons | 43 0 | 4 44 | 4 03 | 0 083 | 7 32 | War Industries Board |
| India ........ | Average of stocks, July, 1918 | 21,721 tons | 50.65 | 6 46 | 6 67 | 0 099 | 1 05 | War Industries Board |
| Cuba ........ | Average of stocks, July, 1918 | 8,786 tons | 37 5 | 4 38 | 15 15 | 0 059 | 7 22 | War Industries Board |
| Costa Rica ........ | Average of stocks, July, 1918 | 1,086 tons | 44.60 | 4 15 | 20 70 | 0 032 | 2 78 | War Industries Board |

[a] Manganese 35 per cent +. [b] Manganese 40 per cent +. [c] Approximate.

from them, or on the development of variations of present methods of steel making whereby such ore might in part replace the alloys now used. That both tendencies have been in operation is shown by the destination of shipments of ore, as well as the increase in manganese metal used as spiegeleisen, from 9 per cent. of the total prior to 1913, to 15 per cent. of the total in 1917. Recent studies have shown that the metallic manganese in alloys made from domestic ores has risen from about 8 per cent. of the total to over 30 per cent. in 1918.

The increase in production of manganese ores, from 1915 to 1918, considered with the available estimates of reserves of ore, give considerable reassurance concerning the possibility of maintaining the present rate of steel production for a year or two in the event that no foreign ore or alloy were available.

## Composition of Domestic Ores

An important element in measuring the extent to which domestic sources of ore meet domestic requirements is the composition of the domestic material. Makers of alloys are particularly interested in the percentages of manganese, iron, silica, phosphorus, and water. The discovery that domestic ores from important mines in several states, notably Georgia and Virginia, contain from 10 to 15 per cent. barium oxide has demanded close examination of manganese ores for this element, however.

Tables 3 and 4 have been prepared on the basis of data reported by various producers of ore as well as the Miami Metals Co., Chicago, and the Southern Manganese Corpn., Anniston, Ala., to the United States Geological Survey, but the most valuable data are based on compilations made from monthly reports submitted by makers of ferro-alloys to the War Industries Board. Mr. H. W. Sanford of the Ferro-alloys Division of that board has kindly permitted the presentation of these data. The weighted analyses of the stocks of Brazilian, Indian, Cuban, and Costa Rican ores on hand July 31, 1918, are given to permit a comparison with domestic ores.

Although there can be no doubt that many of the new, as well as old, domestic deposits can produce some material with as much manganese and no more iron or silica than the best foreign ore, when the mines exploiting known deposits attempt to produce a considerable part of the requirement under the stress of war conditions and at prices four to five times normal, they yield a material less desirable for making ferromanganese than most of the imported ore. The mines of several districts, such as Batesville, Ark., produce material that compares favorably with most of the foreign ores, but most of the districts, and especially those from which during 1917 and 1918 a large part of the domestic

TABLE 4.—*Composition of Ferruginous Manganese Ores (10 to 35 Per Cent. Manganese), Natural Basis*

| State | Remarks | Quantity, Tons | Manganese, Per Cent. | Iron, Per Cent | Silica, Per Cent. | Phosphorus, Per Cent. | Water, Per Cent | Authority |
|---|---|---|---|---|---|---|---|---|
| Colorado: | | | | | | | | |
| Leadville district | Average stocks July, 1918 | 83,436 | 21.15 | 19.3 | 9 37 | 0.033 | 11 25 | War Industries Board. |
| Minnesota: | | | | | | | | |
| Cuyuna district | Average stocks July, 1918 | 121,235 | 15 20 | 32.10 | 14.89 | 0.119 | 9.42 | War Industries Board. |
| Cuyuna. Phos less than 0 12 per cent | Average of 1917 shipments | 290,344 | 14.98 | 32 99 | 19 12 | ? | ? | Reports to U. S G. S. |
| Cuyuna. Phos. less than 0.12 per cent | Reserves, 1917 | ^c^2,254,000 | 15.69 | 34.99 | 15.62 | 0.083 | 11 35 | J. F. Murphy. |
| Cuyuna. Phos. more than 0.12 per cent | Average of 1917 shipments | 156,608 | 10 29 | 43.36 | 6 84 | ? | ? | Reports to U. S G S |
| Cuyuna. Phos. more than 0.12 per cent | Reserves, 1917 | ^c^11,374,000 | 9 98 | 42 91 | ? | 0.260 | 16.96 | J. F. Murphy. |
| New Mexico: | | | | | | | | |
| Silver City district | Average stocks July, 1918 | 2,181 | 15 19 | 29 35 | 9.7 | 0.018 | 2.32 | War Industries Board. |
| Silver City district, Legal Tender mine | Best of 131 cars | 24 | 20.98 | 35 62 | 8 78 | 0.013 | | Miami Metals Co. |
| Silver City district, Legal Tender mine | Car with low Mn | 35 | 12.25 | 41 22 | 9 91 | 0 020 | 2 90 | Miami Metals Co. |
| United States: | | | | | | | | |
| Various districts | Average stocks July, 1918 | 216,162 | 18.17 | 25.85 | 12.70 | 0.1125 | 9.34 | War Industries Board |

[c] Dry basis.

production was shipped, yield ores that contain a little less manganese, a little more iron and water, and considerably more silica than those from the principal foreign sources. The composition of the ores from the Philipsburg district, which made up nearly one-half the 1917 production, is noteworthy on account of the low percentage of iron and high percentage of silica. Practically all the ores from west of the Rocky Mountains contain less than 0.06 per cent. phosphorus, which is less than that present in most foreign ores. On the other hand, the material shipped from the mines in Alabama, Arkansas, Georgia, Tennessee, and Virginia, where the ores occur in residual clays that result from long-continued weathering contain two to three or more times as much.

One of the most interesting recent events in the domestic manganese industry has been the exploitation of the carbonate ore of the Emma mine at Butte. Although large quantities of such material had been used to make ferro-alloys in Europe, never had it been used in the United States. Approximately 56,000 tons of ore of the indicated composition were shipped from the Emma mine during the year ending Dec. 1, 1918, and the material has been used to advantage in the blast furnace, when mixed with oxide ores, and in electric furnaces, unmixed, in making 70 to 80 per cent. ferromanganese.

Had the shortage of ships that was acute in 1918 caused continued restriction of imports of foreign ore, a much wider market for low-grade manganese ore would have been created. In order to make up the deficit caused by the restriction, most makers of ferromanganese had the choice of maintaining the grade of ferromanganese at 80 per cent. manganese, thereby utilizing rather siliceous material from many domestic sources, or reducing the grade of ferromanganese to 70 or 60 per cent. manganese and thereby utilizing ferruginous manganese ores obtainable in large quantities from a few sources. In response to the appeal of the American Iron and Steel Institute in April, most makers reduced the grade of the alloy and made large purchases of ore in the Cuyuna district, Minn., and Leadville district, Colo. Most of the Cuyuna range material contains more than 0.125 per cent. phosphorus; and although the silica is rather uniformly more than 10 per cent., large quantities of standard grades of spiegeleisen have been made of the ore from one mine without adding high-grade ore. It is unfortunate that the large reserves of Leadville and Silver City districts, which are highly desirable material, are so remote from points of consumption.

## Reserves

Among the purposes of the field investigations of manganese deposits by the United States Geological Survey during the past 3 years, the attempt to estimate reserves has been fundamental. This part of the work was approached with a certain apprehension, for it was recognized

that for most districts neither the extent of explorations nor time available for the work would permit the order of accuracy that most mining companies require as guides in operating. However, since the domestic steel industry, until 1917, depended on overseas imports for about 90 per cent. of the total manganese metal needed, and there was a possibility that a serious shortage of ships might occur during 1918 and 1919, it was thought necessary for the government to have access to dependable estimates that might be the basis of important decisions in matters of national policy.

The geologists engaged in the work have not been ignorant of the problems that were involved in the attempt to assign dependable estimates to the quantities of ore of each grade in deposits that showed every degree of exploration. Early in the investigations it was recognized that important geologic features of several types of deposits were obscure and that estimates of quantity and grades might only indicate the order of magnitude. On the other hand, it could not be denied that such estimates by a skilled observer had definite value. It is now believed that, in addition to the original purpose, the point of view has stimulated closer scrutiny of the features that determine the dimensions of the deposits and that better geological data have been obtained than would otherwise have been the case.

Since a number of geologists have aided in the work and many types of deposits have been examined, complete uniformity of method and the same degree of accuracy in the estimates have not been attainable throughout. In presenting the estimates, it was considered advisable to show two types of reserves. The estimates in the first column under each grade are based on fairly satisfactory data concerning horizontal extent and either definite data concerning persistence in depth or geologic evidence that permitted assigning vertical limits. In only a few deposits, such as several in the Philipsburg and Butte district, Mont., have data necessary to calculate "ore developed" or "ore being developed" as defined by Philip Argall* been obtainable. Conditions in the industry since 1916 have encouraged maximum rate of removal of visible material and few mines have undergone the systematic development demanded by conservative mining. In several states, such as Virginia, Georgia, and Tennessee, where the material marketed is commonly disseminated through masses of clay, which is rarely explored far in advance of mining, the estimates represent little more than the order of magnitude of minimum recoverable quantities. The second column under each grade includes estimates of quantities that will probably be added to those in the first column by work in progress.

* T. A. Rickard and others: "The Sampling and Estimation of Ore in a Mine," 76 to 81. New York, 1907.

TABLE 5.—*Reserves of Manganese and Manganiferous*

| State District | Authority | Number Deposits Examined 1916–1918 | Work Concluded | Number Deposits Examined Less Than 50 Tons | Manganese 35 Per Cent. + | | |
|---|---|---|---|---|---|---|---|
| | | | | | Number of Deposits | Reserves, Tons | Additional Reserve in Prospect |
| Arizona . | F L. Ransome<br>E L Jones, Jr | 76 | Mar., 1918<br>June, 1918 | 12 | 40 | 82,000 | ? |
| Arkansas, western . | H D. Miser | 37 | June, 1916 | 30 | 5 | 600 | |
| Arkansas, Batesville dist . | H D Miser | 181 | June, 1918 | 26 | 110 | 100,000 | 160,000 |
| California . | G D Louderback & associates | 202[a] | Sept, 1918 | 97 | 59 | 45,000 | ? |
| Colorado, Leadville dist.. | J B. Umpleby | 13 | Aug, 1917 | | | | |
| Colorado, other districts | J B Umpleby<br>E L Jones, Jr | 5 | July, 1917 | 4 | 2 | 1,000 | |
| Georgia | J P. D Hull<br>F C Schrader | 53 | July, 1918 | 1 | 25 | 100,000 | 200,000 |
| Minnesota, Cuyuna Range | J F Murphy | 10 | 1917 | | | | |
| Cuyuna Range | J F. Murphy | 15 | 1917 | | | | |
| Montana, Butte district . | J T. Pardee | 62 | Aug, 1917 | | 2 | 2,800 | |
| Montana, Philipsburg dist | J. T. Pardee | 25 | Oct, 1918 | | 25 | 130,000 | 350,000 |
| Montana, other districts . | J T. Pardee | 4 | 1917 | 1 | 1 | 100 | |
| Nevada . . | J T. Pardee<br>J C. Jones<br>E L Jones, Jr. | 35 | July, 1918 | 2 | 18 | 21,000 | ? |
| New Mexico .. | E L. Jones, Jr<br>J B. Umpleby<br>E. H Wells | 22 | July, 1918 | 6 | 7 | 3,500 | |
| North Carolina | F. C Schrader | 17 | Nov, 1918 | 14 | 1 | 50 | |
| Oklahoma ..... | D. F. Hewett | 3 | Oct, 1917 | | 3 | 1,000 | |
| Oregon ....... | J. T Pardee<br>E S Larsen | 23 | Aug, 1918 | 15 | 6 | 13,000 | |
| Tennessee . .. | G W. Stose<br>F. C. Schrader | 120 | Oct, 1918 | 78 | 42 | 45,000 | 20,000 |
| Utah .... | V. C. Heikes<br>J. T. Pardee | 45 | June, 1918 | 25 | 15 | 11,000 | ? |
| Virginia, east side of valley | G. W. Stose<br>H D. Miser<br>F. J. Katz<br>D F Hewett | 110 | Sept, 1917 | 58 | 30 | 100,000 | 350,000 |
| Virginia, west side of valley | G. W. Stose<br>H D. Miser | 84 | Sept., 1918 | 45 | 25 | 42,000 | 50,000 |
| Washington .. | J. T. Pardee | 32 | Aug, 1918 | 2 | 1 | 500 | |
| Alabama.. ..<br>Idaho .. ...<br>South Carolina<br>Texas .. ...<br>Wyoming. .... | G. W. Stose<br>L. G Westgate<br>F C Schrader<br>J B. Umpleby<br>E. L. Jones, Jr | 7 | 1917 & 1918 | 2 | 4 | 1,200 | |
| Total .. ...... | | 1181 | | 428 | 421 | 699,750 | 1,130,000 |

Note.—The interrogation mark indicates that further exploration will probably reveal additional

[a] Incomplete

*Ores in the United States, December, 1918*

| Manganese 5–35 Per Cent., Largely More Than 20 Per Cent. Silica, Less Than 30 Per Cent Iron | | | Manganese 5–35 Per Cent., Largely More Than 30 Per Cent. Iron and Less Than 20 Per Cent. Silica | | | Remarks |
|---|---|---|---|---|---|---|
| No of Deposits | Reserves, Tons | Additional Reserve in Prospect | Number of Deposits | Reserves, Tons | Additional Reserve in Prospect | |
| 23 | 235,000 | ? | 1 | 5,000 | | Practically all recorded deposits have been examined |
| 2 | 200 | | | | | |
| 45 | 160,000 | ? | | | | |
| 43 | 54,000 | ? | 3 | 6,000 | | Practically all recorded deposits have been examined in cooperation with Cal Council of Defense |
| | | | 13 | 700,000 | 800,000 | Most important deposits examined, material commonly ranges from 20–30 per cent. iron |
| 1 | 200 | | 2 | 750,000 | 250,000 | Only a few deposits in Chaffee, Eagle, Fremont & San Miguel Co. examined |
| 17 | 50,000 | 230,000 | | | | All deposits in Bartow; several in Fannin, Murray & Polk Co, examined in cooperation Georgia Geological Survey. |
| | | | 10 | 2,254,000 | ? | Cuyuna range, less than 0 125 per cent phosphorus. |
| | | | 15 | 11,374,000 | ? | Cuyuna range, more than 0 125 per cent phosphorus. |
| 60 | 400,000 | | | | | All recorded deposits of oxide ores examined, estimate does not include large deposits of carbonate ore, 35–38 per cent manganese |
| 19 | 56,000 | 230,000 | | | | |
| 2 | 700 | | 1 | 1,800 | | Madison County |
| 15 | 6,000 | | 4 | 552,000 | 500,000 | All recorded deposits have been examined. |
| 6 | 133,000 | | 3 | 500,000 | 500,000 | All recorded deposits have been examined |
| 2 | 200 | | | | | Almost all recorded deposits have been examined |
| 2 | 1,700 | | | | | Johnson Co only; examined in cooperation with Oklahoma Geological Survey. |
| 2 | 101,000 | | | | | All recorded deposits in Baker, Jackson, and Josephine Co. have been examined |
| | 25,000 | 20,000 | 15 | 10,000 | | All recorded deposits have been examined in cooperation with Tennessee Geol Survey |
| | 4,000 | | | | | All recorded deposits have been examined. |
| | | | 22 | 5,000 | 100,000 | All recorded deposits have been examined except Piedmont region, east of Blue Ridge |
| | | | 14 | | ? | Examined in cooperation with Virginia Geological Survey |
| 30 | 86,000 | ? | | | | All recorded deposits, except several in Gray's Harbor Co, examined; reserves range 30–35 per cent manganese. |
| 1 | 10,000 | ? | | | | Some important deposits not examined |
| 275 | 1,323,000 | 480,000 | 103 | 16,157,800 | 2,150,000 | |

reserves.

*Classification of Reserves.*—The classification used in Table 5 aims to indicate the use to which the material can probably be put. Although material with more than 35 per cent. manganese can contain about 40 per cent. silica, in which case it would not be acceptable for making ferromanganese, most of the material under the heading "Manganese 35 per cent." can readily be used to make ferromanganese with 70 to 80 per cent. manganese. By judicious mixing, all of it can be so used. Where the material of a deposit is not homogeneous throughout, only that part with more than 35 per cent. manganese that may readily be selected under existing conditions is considered as the reserve. The average manganese content of this class is about 40 per cent.

In the second class is placed all material with from 5 to 35 per cent. manganese, and generally less than 30 per cent. iron, that is not now readily marketable on account of high silica or lime content. With improvements in the processes of beneficiation, it would become available as a source of manganese. The average manganese content is about 25 per cent.

The third class contains all material in which manganese is equal to or less than the iron and may either be used to make spiegeleisen or, if the phosphorus is too high, may be used in making high-manganese pig iron. The average manganese content of the natural material is about 16 per cent.

*High-grade Ore.*—The scope of this paper does not permit a detailed review of the reserves in each state. It is clear, however, that the largest reserves of high-grade ore are in the Philipsburg district, Mont. The nature of the deposits, as well as the state of mining explorations, makes the ore of this district the most readily available for the alloy industry in the event of great need. Large reserves of high-grade ore undoubtedly exist in Virginia, Georgia, and Tennessee; but explorations are meager and, since the rate of production depends on elaborate milling equipment, they would not contribute largely to an urgent demand. This is clearly shown by records of production during 1917 and 1918. The known reserves of Arizona contain less manganese and more silica than the ores of Philipsburg. If it is assumed that about 850,000 tons of high-grade ore are needed to make the 42,000,000 tons of steel produced in 1918, domestic reserves can be depended on for about 1 year's supply. Since only a part of this is readily available it is improbable that known deposits could within 1 year supply more than one-half the needed ore.

*Low-grade Ore.*—The Butte district, Mont., and Arizona contain the largest reserves of low-grade siliceous ore, very little of which has been amenable to concentration by known processes even at the high prices prevailing in 1917 and 1918. If these prices were assured over a reasonable period, the material would offer an attractive field for research and appreciably add to the dependable reserves. The silicate

ores of the Olympic Mountains, Wash., which are found near large undeveloped sources of water power, would also be especially attractive.

The ferruginous-manganese ores of the Cuyuna Range, Minn., appear to contain more than three-fourths of the reserves of that grade in known deposits. For several years after they had been extensively explored, there was much discussion concerning the use to which the product might be put. Although there may be problems in marketing some of the material when prices are nearer those prevailing prior to 1914, these deposits must be considered the most important single source of manganiferous ore in time of great need Since they are nearest the centers of iron and steel manufacturing, most of the output will be more readily available to that industry than higher grade, but more remote material, at Leadville and Silver City. Only unusually high prices would permit marketing the ores of the Pioche districts in the steel-producing region east of the Mississippi. The utilization of the low-grade ores of practically all districts depends largely on either the increased use of spiegeleisen, into which they can be converted, or on new or unperfected processes of steel making. There can be no doubt, however, that they are the most important domestic source of manganese-bearing material.

## CONCLUSION

As the result of high prices and the widespread appeal from many sources, the country has been closely examined for manganese deposits. In this search, prospectors, miners, mining engineers, geologists, as well as the friendless brokers, have played a part. Obviously, the search has been most intensive in those regions in which deposits had been previously exploited. It is rather interesting, however, that the contribution from the well known deposits in Virginia, Georgia, Arkansas, and California has been much less than that from districts such as Philipsburg, Mont., and Bisbee, Ariz., which, although extensively explored for other metals prior to 1915, were not known to contain large bodies of manganese oxides. Although it would not be wise to assume that further search will not show the presence of many additional deposits that could contribute in time of need, it appears that almost every locality of promise has been examined.

Although preliminary reports describing most of the districts have been published, only the salient geological problems have been indicated. At present, there is good basis for thinking that these recent studies of domestic deposits have yielded data that will offer important contributions to the knowledge of manganese deposits.

There can be no doubt that many competent observers, including the consumers of manganese ore, have been surprised to observe the increase of the domestic output from less than 2 per cent. of requirements

in 1916 to 35 per cent. in 1918. If some consumers have been reluctant to use the available high-grade, as well as low-grade domestic ores, other consumers, advantageously situated or with commendable enterprise, have proved conclusively that the problems of utilization of the domestic material are not as insuperable as at first was thought to be the case. There is even good basis for the assumption that, under stress, the steel and related industries could be maintained at the 1918 rate of production for a short period, at least, entirely on the products of domestic mines.

From the beginning of the campaign of the examination of deposits by the U. S. Geological Survey, the attempt was made to estimate the character and extent of reserves of manganiferous materials. Making proper allowance for the probable accuracy of the estimates that have been made, it can be stated that the known deposits of high-grade material are large enough to maintain the steel industry at the 1918 rate of production for 1 year. The extent of explorations, considered with geological observations, holds out the hope that 2 years' supply may exist in these deposits, but it seems certain that 3 or 4 years' supply is not by any chance available.

The widespread utilization of low-grade, in place of high-grade, material undoubtedly presents imposing metallurgical problems. To the optimistic observer, not competent to consider these problems in detail, the progress made to this end during 1917 and 1918 offers considerable encouragement. The large reserves of the low-grade material can probably be depended on, under stress, to double the probable life of the high-grade ore.

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# Manganese-ore Deposits in Cuba*

BY ERNEST F. BURCHARD,† M. S., WASHINGTON, D. C.

(Chicago Meeting, September, 1919)

A RECONNAISSANCE of the manganese- and chrome-ore deposits of Cuba was made by the writer, as a representative of the U. S. Geological Survey, in company with Mr. Albert Burch of the Bureau of Mines, under instructions of the Secretary of the Interior, in the spring of 1918. The Cuban Government courteously detailed Sr. E. I. Montoulieu, a mining engineer connected with the Treasury Department, to act as escort and associate throughout the work on the island. The object of the study was to obtain authentic information for use of the United States Government as to the location, character, quantity, and availability of the manganese- and chrome-ore deposits in Cuba. Accordingly, the work comprised examination of such deposits as seemed to be of promise, without regard to stage of development, with a view to a determination of the quantity and quality of ore likely to become available for shipment during 1918 and 1919, and of the tonnage of ore in reserve. While it was possible to visit most of the deposits that seemed to be of promise, there was not sufficient time to visit all that were brought to our attention, some of which may have merit. Moreover, the necessity for paying particular attention to the ore deposits and the mines precluded the possibility of making more general studies of the geology of the region, and of making satisfactory collections of country rock and of fossils, and so for other details as to the geography and geology of Cuba the reader is referred to the report by Hayes, Vaughan, and Spencer [1]

Mr. Burch returned to the United States about 10 days in advance of the writer, who continued, in company with Sr. Montoulieu, investigations of reported deposits in Pinar del Rio, Matanzas, and Santa Clara Provinces. On his arrival in Washington, Mr. Burch submitted to the Director of the Bureau of Mines a summary of the results of the work to date. Later the writer published, as *Press Bulletin* No. 380 of the U. S. Geological Survey, a statement concerning the chrome and man-

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[1] C. W. Hayes, T. W. Vaughan, and A. C. Spencer: Report on a Geological Reconnaissance of Cuba, made under direction of General Leonard Wood, Military Governor (1901), U. S. Geol. Survey *Bull.* 213.

ganese ores of Cuba, and in that bulletin, as well as in this paper, the engineering data contained in the unpublished report of Mr. Burch has been freely used.

A few months prior to the field work of the writer, G. Sherburne Rogers had examined a manganese deposit near Trinidad and Max Roesler, a deposit near Mendoza. The field work of these geologists was not duplicated, and the writer has taken, from their reports, essential facts concerning the deposits they described.

Grateful acknowledgments are due Sr. Montoulieu; his knowledge of the language and customs, general acquaintance with the regions visited, and his professional training made his service of inestimable value. Much valuable assistance and information were given by Mr. George Reno, Chief of the Bureau of Information in the Cuban Department of Agriculture, Commerce, and Labor; by Sr. Pablo Ortega, Director of the Bureau of Mines of Cuba; by Major Thomas F. Van Natta, Jr., Military Attache to the United States Legation; and by Captain Varona of the Cuban War Department. Especial thanks are due Messrs. Chas. F. Rand and E. G. Spilsbury, of New York City; George E. Starr, of Philadelphia; Pedro Aguilera, Eugenio Aguilera, Wm. Pitt, and Frederico Lopez-Adazabal, of Santiago de Cuba; A. J. Trumbo, Howard Trumbo, and Capt. W. F. Smith, of Havana; and the many mining officials and operators throughout the Island, through whose assistance the party was enabled to cover the territory to the best advantage.

In the study of material for the preparation of the present paper, petrographic determinations have been made by E. S. Larsen, W. T. Schaller, D. F. Hewett, and Miss E. F. Bliss, chemical analyses by George Steiger and W. T. Schaller, and paleontologic determinations by T. Wayland Vaughan and Joseph A. Cushman, all of the U. S. Geological Survey, besides paleontologic determinations by J. W. Gidley of the U. S. National Museum.

## Distribution of Deposits

Manganese ore is found in Cuba in Oriente, Santa Clara, and Pinar del Rio Provinces, but in Oriente Province only does it occur in large commercial quantities. There, the deposits are in three areas: one north of Santiago de Cuba, one south of Bayamo and Baire, and one on the Caribbean coast between Torquino Peak and Portillo. In Santa Clara Province, a little ore has been found near the Caribbean coast west of Trinidad; and in Pinar del Rio Province, manganiferous material occurs north of the city of Pinar del Rio and farther west near Mendoza. All these deposits were examined, but only the deposits in Oriente Province, near Santiago and Bayamo, give promise of considerable production. A map of the island showing the location of the deposits is given in Fig. 1.

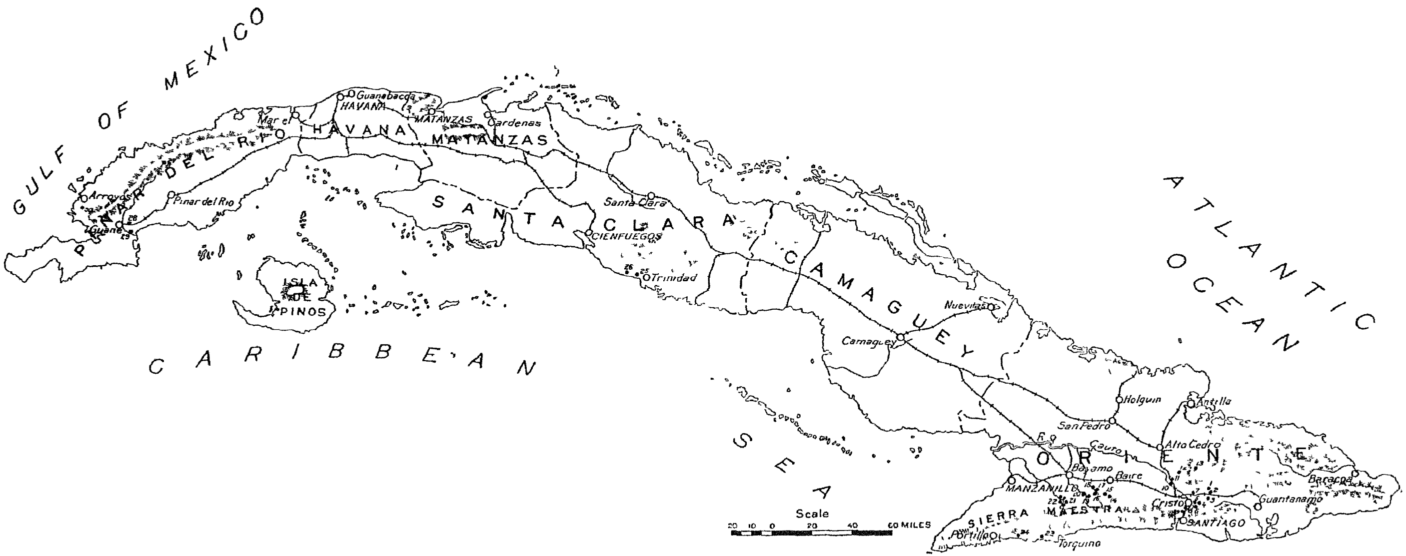


FIG. 1.—MAP OF CUBA SHOWING LOCATION OF PRINCIPAL MANGANESE MINES AND PROSPECTS. THE MANGANESE MINES AND PROSPECTS ARE INDICATED BY NUMBERS, AS FOLLOWS:

1 Laura mine
2 Dolores mine
3 Ponupo group
4 Botsford mine
5 Ysabelita mine
6 Boston mine; Pilar mine
7 San Andreas mine
8 Agosta Luis mine
9 Caridad prospect
10 Abundancia (Cauto) mine
11 Llave mine
12 Gloria mine
13 Valle prospect
14 San Antonio mine
15 Adriana mine
16 Charco Redondo mine
17 Llego prospect
18 Guisa prospect
19 Cadiz mine
20 Francisco mine
21 Costa group
22 Manuel mine
23 Magdalena prospect
24 Camaroncito group
25 Trinidad mine
26 Dario prospect
27 Dolores prospect
28 Turrialba prospect
29 Maria Cristina prospect
30 Mantua manganiferous iron-ore deposits

## General Geologic and Topographic Features

The manganese ores of Cuba occur in the oxidized zone, mainly near the surface, but in places extend to depths of at least 100 ft. and considerably below groundwater level. The deposits are found principally in sedimentary rocks, such as limestone, sandstone, and shale, that are in places metamorphosed, and in beds that originally may have been water-laid tuff but are now partly replaced by manganese oxide, zeolites, calcite, and other minerals. In the most heavily mineralized localities the deposits are in and about masses of siliceous rock, locally termed "jasper" and "bayate," that are associated with the country rock. At one locality south of Bayamo, the manganese and its siliceous associates are in igneous rocks, such as latite-porphyry and latite.

The sedimentary rocks with which the manganese deposits are associated are in some places nearly horizontal but generally show dips ranging from a few degrees to 45° or more. These inclined beds usually represent portions of local folds. Some faulting is shown in the vicinity of certain manganese deposits and may have influenced the localization of the deposits

The area north of Santiago and that south of Bayamo are north of the mountain range known as the Sierra Maestra, but that on the coast east of Portillo is at the southern base of this range. The area north of Santiago is, broadly speaking, in the basin formed by the Sierra Maestra on the south and the Sierras de Nipe and del Cristal on the north, the greater part of which is drained westward by Rio Cauto and its tributaries, and small parts of it by Rio San Juan and Rio Guantanamo to the south and east. The deposits of manganese ore are found on both sides of the basin. The deposits in the area south of Bayamo are in the northern foothills of the Sierra Maestra drained by Bucy, Bayamo, Yao, and Cautillo rivers.

The deposits in the two areas north of the Sierra Maestra show an interesting concordance in altitude. They are from 500 to 1200 ft. (152 to 365 m.) above sea level, and most of them are at altitudes of nearly 600 to 700 ft. (182 to 213 m.), suggesting a relation between the deposition of the manganese and a certain stage in the physiographic development of the region Most of the manganese-ore deposits are above drainage level on the slopes of hills of moderate height, the maximum relief in the immediate vicinity of the deposits seldom exceeding 500 feet.

Spencer,[2] who has studied the geologic section across the island through the area near Santiago, has pointed out that the geologic structure between Guantanamo on the east and Manzanillo on the west is

[2] A C Spencer: Manganese Deposits of Santiago, Cuba. U. S. Geol. Survey *Bull.* 213 (1902) 251–252.

that of a broad synclinal fold, with an east-west axis; the stratified rocks that compose the northern slopes of the Sierra Maestra dip at angles of from 10° to 20° toward the depressed area of the interior occupied by Rios Cauto, Guaninicum, and Guantanamo, while upon the north side of these drainage basins the strata rise toward the mountains that border the north coast. He states that the rocks exposed along the crest of the Sierra Maestra are coarse, well-stratified, volcanic breccias; and that upon the northern slope these pass beneath strata showing an alternation of marine sediments and fine-grained volcanic tuffs, which are in turn covered by flows of basalt and still other fragmental volcanic deposits, finally grading into and giving place to limestones and other purely marine sediments; see Fig. 2.

Some of the manganese-ore deposits have been observed to lie on the crest and slopes of anticlinal hills, but these anticlines are but local puckerings of the strata forming the broad synclinal basin just mentioned. The foraminiferal limestones and greensand marls associated with the manganese ores in the Santiago district, and also probably along the northern foot of the Sierra Maestra south of Baire, are regarded by Cushman as of the upper Eocene age. Eocene time is considered to have been characterized in the area of Oriente Province chiefly by subsidence, with active volcanoes, causing interbedding of volcanic and sedimentary rock.[3]

## Types of Deposits

The deposits of manganese ore examined in Cuba consist chiefly of mixtures of some or all of the oxides pyrolusite, psilomelane, manganite, and wad, and there are probably others present, but as a rule the individual minerals are not readily distinguished, and they have not been

[3] Hayes, Vaughan, and Spencer: *Op. cit.*, 31–33. See also reprint of the chapter on physiography and general geology, partly revised by Pablo Ortega, Director of the Bureau of Mines, Havana, 1918.

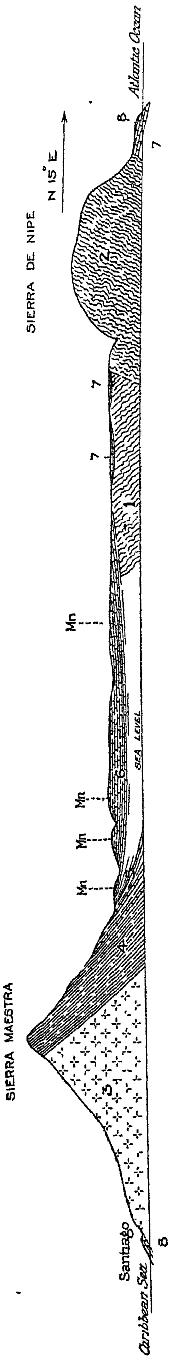


FIG 2.—SECTION ACROSS ORIENTE PROVINCE FROM SANTIAGO TO ANTILLA LENGTH 65 MILES, VERTICAL SCALE EXAGGERATED (*After Hayes, Vaughan, and Spencer.*) 1, METAMORPHIC ROCKS; 2, SERPENTINE; 3, INTRUSIVES; 4, VOLCANIC FLOWS AND TUFFS, 5, VOLCANIC BRECCIAS AND AGGLOMERATE, WITH SOME INTERCALATED LIMESTONE; 6, LIMESTONE, GLAUCONITIC SANDSTONE, AND SHALE, WITH A FEW THIN BEDS OF TUFF; 7, MASSIVE LIMESTONE; 8, COASTAL LIMESTONE.

studied in detail. Braunite, an oxide containing manganese silicate, was determined in a specimen from the Camaroncito district. With this exception no manganese silicate was found in the Oriente ores, nor was any manganese carbonate noted, but small quantities of ankerite and rhodonite were observed by Rogers in manganese-bearing metamorphosed limestone and calcareous schist near Trinidad, Santa Clara Province. The deposits in Oriente vary in form and associations but they may be grouped into three general types according to their associations: deposits in bedded rocks, deposits in irregular masses of siliceous rock (jasper, or bayate), and deposits of nodules and fragments in clay. (The term jasper is more commonly used in the Santiago district, and bayate in the Bayamo district.) The "bedded" deposits comprise several varieties, one of the most common consisting of poorly consolidated, tuffaceous material, granules of pink clay, zeolite minerals, and manganese oxides. Such deposits are of at least two generations; the earlier deposits (Figs. 23, 26), occur as thin tabular or lenticular bodies interbedded with Eocene limestone strata, and the later ones fill depressions in the surface and solution cavities within the limestone (Figs. 9, 10, 19). These latter deposits may in places be formed by slumping down of beds due to caving of underlying rocks; they are then apt to be confused with deposits of the residual clay type. Other bedded deposits are replacements of limestone, sandstone, and conglomerate beds, a good example of fossil bog deposit was found (Figs 24 and 25). The masses of jasper are irregular in outline (Figs. 7, 10, 12, 22). They cut across beds and also follow the general planes of bedding in places. Manganese oxides in a nearly pure state are found in pockets; they are also intricately disseminated through the jasper.

The extent and quality of the deposits varies considerably. Most of the largest and richest orebodies are associated with jasper, those that have replaced limestone thoroughly are also rich but small. The tuffaceous ore where interbedded with limestone is mostly in thin beds; where it is detrital it generally occurs in deposits of workable size but of low grade and must be washed in order to separate the granular gangue material from the manganese oxide Deposits in residual clay yield good ore after picking, screening, or washing, but are not numerous or extensive.

## Deposition of the Manganese Ores

The deposits of manganese oxides associated with masses of jasper or bayate appear to have been one of the earliest of the several types of ore to be formed. The field relations of the jasper bodies indicate that the jasper has been deposited in the limestone and glauconitic beds by solutions that have enlarged openings along joints and crevices and locally replaced the enclosing rocks by silica and manganese minerals.

The relations of the manganese minerals to the jasper as observed in thin sections suggest that the manganese and silica have been deposited very nearly contemporaneously. If not wholly so, then some of the manganese probably came in with a secondary silicification. There has been more or less replacement of silica by manganese, also leaching of manganese minerals from the jasper and redeposition in other rocks. As a result of a careful examination of the deposits at Ponupo, Spencer[4] came to the conclusion that bayate (jasper), and oxides of manganese had been deposited as replacements of limestone and of calcareous sandstone. He considered the relation of the siliceous materials to the manganese oxides to be so intimate that, in some instances, the ore appeared to have been deposited subsequent to the bayate and had in part replaced the siliceous material; but in other instances the silica appears to have been the replacing substance and to have been brought in after the formation of the ore.

The nature of the regularly bedded deposits, which consist of granular to fragmental material containing specks and grains of pink clay, the whole more or less cemented by oxides of manganese, is of considerable interest. The original material of the beds, without much question, is volcanic tuff laid down in an Eocene sea, between deposits of foraminiferal limestone. (At the Charco Redondo mine, sharks' teeth were found in a thin layer interbedded with massive limestone). The manganese minerals appear to have been introduced later in waters that readily moved along the porous beds, the materials of which they partly replaced. Some irregular masses appear to have been deposited by streams that transported material from other beds of tuff that had disintegrated rapidly, the detrital material having been cemented by manganese oxide.

Little direct evidence could be obtained during the rapid reconnaissance as to the source of the manganese prior to its deposition with the jasper and in the lenses of tuff, but the proximity of volcanic rocks to the manganese-bearing areas in Oriente Province, and the broader structural relations of the region (Figs. 2 and 20) suggest the possibility that the manganese was derived from volcanic rocks of the Sierra Maestra, transported by artesian waters and deposited, together with silica, as masses of manganiferous jasper in joints, fissures, and cavities in the limestone and other rocks in the local anticlines of the basin, as well as along the beds of the more porous tuff. The jasper and bedded tuff in weathering may have contributed the manganese that is now found in the adjacent rocks and in the detrital deposits.

Likewise, the manganese near Trinidad may have come from volcanic or other igneous rocks in the Sierra de Trinidad, structural conditions there being favorable for the flow of underground waters toward the south coast.

[4] Hayes, Vaughan, and Spencer: *Op. cit.*, 64–65.

*Examination of Siliceous Ores by E. F. Bliss*

Thin sections of some of the typical ore associated with jasper and other siliceous rock were examined by Miss E. F. Bliss of the U. S. Geological Survey, who reports as follows.

*Specimen of Manganiferous Jasper, or Bayate, from Balkanes Opening, Ponupo Group*—A dark-colored rock consisting of an aggregate of brilliant steel-gray to black pyrolusite crystals associated with a reddish-brown ferruginous chert. Crystalline quartz cuts the pyrolusite in veins and irregular areas. The rock is full of small cavities lined with pyrolusite crystals. Under the microscope, the rock is seen to consist of an opaque brown ferruginous chert containing numerous small rounded areas of prismatic quartz that crystallize in a radiating aggregate, so that under crossed Nicols these quartz crystals show most beautiful spherulitic crosses. These spherulites are stained brownish yellow and in some cases show a concentric zonal structure, due apparently to changes in saturation of iron oxide at the time of crystallization of quartz. The whole rock is cut by veins and irregular replacement of crystalline quartz usually in a fine-grained mosaic. In the wider veins the center is filled by a coarsely crystalline aggregate of interlocking quartz. Scattered throughout the slide there are small areas of dark-colored manganese oxide, which can be distinguished from opaque brown chert by the fact that in incident light they are a steel gray while the iron oxide is reddish brown.

It would appear that the rock was an original ferruginous chert in which a secondary crystallization has been established, thereby forming the spherulitic quartz aggregates at the expense of the original material. The relation of the quartz filling to the spherulitic crystallization is not perfectly definite; but in the majority of cases the quartz appears to cut the spherulites, eating into them and rendering their outline indistinct. That the quartz has replaced the original chert is evident from the fact that within the quartz vein are faint wavy brown lines, which lie on the border of the chert and are apparently contemporaneous with the formation of the chert. These lines, which indicate a depositional change in the original material, antedate the vein crystallization because the prismatic quartz crystals cut across the chert and the spherulitic quartz appears to have been brought in by the siliceous water and deposited throughout the rock more or less at random. In the cases where the spherulites are the best preserved, the manganese is the least abundant, always forming a vein around the outer cherty border and never occurring in the center of the spherulites. Where the spherulites are more thoroughly impregnated by the secondary quartz, the manganese is quite abundant. In one case a quartz vein appears to be directly continuous with a quartz spherulite as if the vein formation was contemporaneous with the spherulitic crystallization; but in the majority

of cases it would seem that the impregnation of the rock by siliceous water carrying manganese was secondary both to the chert and to the spherulitic quartz.

*Specimen of Manganiferous Bayate from Oviedo Claim, Costa Group.*—An aggregate of beautifully radiating areas of soft glistening steel-gray pyrolusite apparently surrounds areas of brown chert; the rock is filled with considerable secondary crystalline quartz. One slide shows an area of opaque brown chert grading into an area of spherulitic quartz crystallization similar to the ore at Ponupo. The chert is abundantly replaced by black manganese oxide, which extends into the spherulitic quartz forming complete rings around each spherulite. Both chert and spherulites are cut by narrow veins of clear fine-grained quartz. The other slide shows, on one side, an opaque dark-colored area in which dark brown chert is almost completely replaced by a massive steel-gray to black manganese mineral. Adjoining this opaque area is a border of clear, fine-grained, radial, quartz crystallization in which manganese oxide forms cores and concentric rings. This fine-grained clear quartz grades into an area of brown cherty looking material containing minute grains of manganese oxide scattered throughout the mass. Under crossed Nicols, this opaque light brown area shows a beautiful spherulitic crystallization that is coarser grained than the adjoining clear quartz area but apparently is directly continuous with it. The manganese in both the clear and the opaque areas cut across the spherulitic crystallization.

*Specimen of Manganiferous Banded Chert from Manuel Claim.*—A dark-colored rock of fairly high specific gravity showing a fine banding of black hard cherty material and steel-gray soft pyrolusite. Under the microscope, the rock appears to be an original ferruginous chert and cryptocrystalline spherulitic quartz associated with bands of granular quartz. The whole rock has been penetrated and replaced by a dense black manganese oxide that forms prismatic crystals. It cuts across the bands of granular quartz in fine stringers and appears to be the last material formed.

*Specimen of Manganiferous Siliceous Limestone from Camaroncito.*—This is a hard black rock of high specific gravity chiefly composed of psilomelane; thin section shows a brown chert similar to the ferruginous chert at Ponupo. The chert is thoroughly interpenetrated and replaced by a dark brown to black manganese mineral, which forms an intricate network throughout the rock; in some places, it is replaced by dense steel-gray to black aggregates. The interstices of the rocks are filled by crystalline calcite, which often shows sharp outlines against the chert and appears to be secondary to the chert. In some places the limestone encloses black needles of a manganese mineral which appear to have crystallized out with the calcite. It looks as if the manganese and calcite had been deposited together and are both secondary to the chert.

Miss Bliss summarizes her observations as follows: "The manganese is evidently a replacement of an original ferruginous chert or jasper. The chert has apparently undergone a secondary spherulitic crystallization, perhaps under the influence of impregnating siliceous waters. This spherulitic crystallization was accompanied and followed by a vein filling of quartz, which was doubtless the medium through which the manganese minerals were deposited"

### *Examination of Clay by Larsen, Hewett, and Steiger*

An incomplete examination of the powder from some of the pink clay and other gangue material from a specimen of ore from the Abundancia mine was made under the microscope by E. S. Larsen of the U. S. Geological Survey, who comments as follows: "The dull pinkish bodies are made up mostly of rather well-formed to poorly formed crystals of kaolinite (?) with much zeolite and some calcite. Veinlets of zeolite are also present and are clearly later than the manganese oxide that they cut. The zeolites are analcite, heulandite (?), laumontite (?), and probably others, and they make up much of the gangue of the ore. Some parts of the specimen that look like the dull pinkish bodies but carry abundant well-formed fresh crystals of zoned andesite feldspar, green augite, and brown hornblende are an ordinary, fresh, sandy, andesitic tuff. It seems probable that this andesitic tuff represents remnants of the original bed and that the manganese ore and the zeolite bodies have replaced it."

A little of this granular, pink, clay-like material from which most of the black manganese oxide had been separated was analyzed by George Steiger with the following results: Insoluble in HCl, 58.39 per cent; $(AlFe)_2O_3$, 13.76 per cent.; CaO, 3.60; MgO, 2.60, Mn, 5.79 per cent.

A green and pink clay that occurs in streaks in the bedded deposits at the Laura mine possesses considerable absorptive power and, when immersed in water, swells and then breaks down to a light powder. The residue from the washed powder, examined under the microscope by D. F. Hewett and E. S. Larsen, was found to contain about 60 per cent. plagioclase in clear sharp grains. Some magnetite is also present. An analysis by George Steiger of the pink clay showed: Soluble in HCl, 53.16 per cent.; $(AlFe)_2O_3$, 14.74 per cent.; Mn, 0.50 per cent.; CaO, 3.40 per cent.; MgO, 4.00 per cent.; loss on ignition, 24.76 per cent. The material is suggestive of volcanic ash and resembles bentonite, the volcanic origin of which has been suggested by Hewett[5] and by Wherry.[6]

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[5] D. F. Hewett: Origin of Bentonite and Geologic Range of Related Materials in the Bighorn Basin, Wyoming. *Jnl.* Wash. Acad. Sci. (Apr. 4, 1917) **7**, 196–198.

[6] E. T. Wherry: Clay Derived from Volcanic Dust in the Pierre in South Dakota. *Jnl.* Wash. Acad. Sci. (Nov. 19, 1917) **7**, 576–583.

The evidence therefore points to a volcanic origin of these bedded materials, probably to the deposition of the volcanic materials directly into shallow water; in some places these mingled with unconsolidated calcareous sediments, and in others they overlay limestone and other sedimentary beds already formed and in turn were overlain by other terrestrial sediments.

## Deposits in Oriente Province

### *Representative Mines and Prospects in District Near Santiago*

The deposits of manganese ore examined in the Santiago district comprise the Ponupo group, the Ysabelita, Botsford, Boston, Pilar, Dolores, Laura, San Andreas, Cauto (Abundancia), Llave, and Gloria mines, and the Caridad and Valle prospects. All these properties, except the two prospects, were producing ore. A few small producing mines in the

Fig. 3.—Sabanilla, Cuba, Dolores mine. Folded tuffaceous sandy manganiferous strata overlying masses of jasper. Drift in pocket of manganese ore in crevice in jasper.

district were not visited. The Ponupo, Ysabelita, and Boston mines were opened many years ago and have produced a large quantity of ore. The Ponupo and Ysabelita are still relatively large producers, though the grade of ore is not so high as in the earlier days.

*Dolores Mine.*—The Dolores mine is situated at Laura station on the Guantanamo & Western Railroad about 1½ mi. (2.4 km.) west of Sabanilla and is operated by the Campania Oriental de Minas, of Havana and Santiago de Cuba. The manganese-ore deposits lie near the base of the south slope of a gently rising hill at a barometric altitude of between 425 and 475 ft. (129 and 144 m.) Masses of jasper are exposed in some of the upper cuts, but the local rock is mainly a fine-grained, greenish,

thin bedded sandstone dipping about 25° toward the southeast, as shown in a prospect tunnel that extends 200 ft. (60 m.) N. 60° W. into the base of the hill. No manganese ore was encountered in this tunnel nor in the upraise that connected it with the surface 40 ft. (12 m.) above. A little higher, however, open pits disclose masses and boulders of jasper partly replaced by rich manganese ore and mingled with lumps of manganese ore and clay. These jasper masses are overlain by stratified

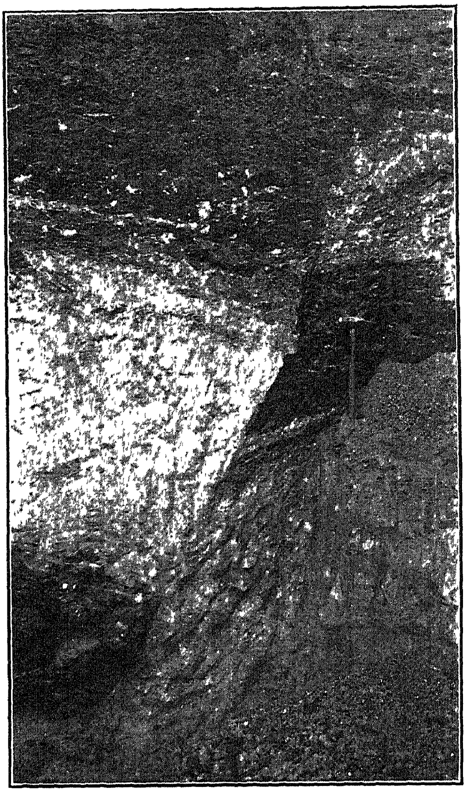

FIG 4.—SABANILLA, CUBA SMALL FAULT IN SANDY MANGANIFEROUS BEDS AT DOLORES MINE.

sandy tuffaceous beds carrying more or less manganese oxide as a cementing material. These beds, although not very firm or brittle, have been sharply folded and faulted, as is indicated in Figs. 3 and 4. At a distance of about ½ mi. to the north another opening made by this company adjoins that of the Laura mine; there are also several other claims in the vicinity on which manganese ore has been found. Taken altogether, there is still a fair tonnage of ore in reserve and more may be discovered.

The mine workings of the Dolores consist of a dozen or more open cuts and several tunnels, shafts, and test pits and are well situated with

respect to transportation. A washer with one 20-ft. (6-m.) log had been installed at the time of visit, with which it was planned to wash a considerable stock of screened ore already on hand. The output of ore with a crew of 96 miners was reported to be about 7 tons of high-grade ore (40

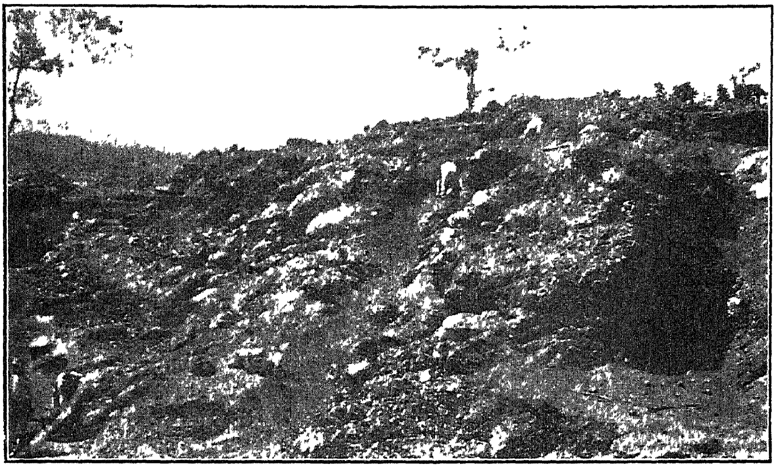

FIG. 5.—SABANILLA, CUBA, LAURA MINE. JASPER MASSES AND BOULDERS PARTLY SURROUNDED BY MANGANESE OXIDES

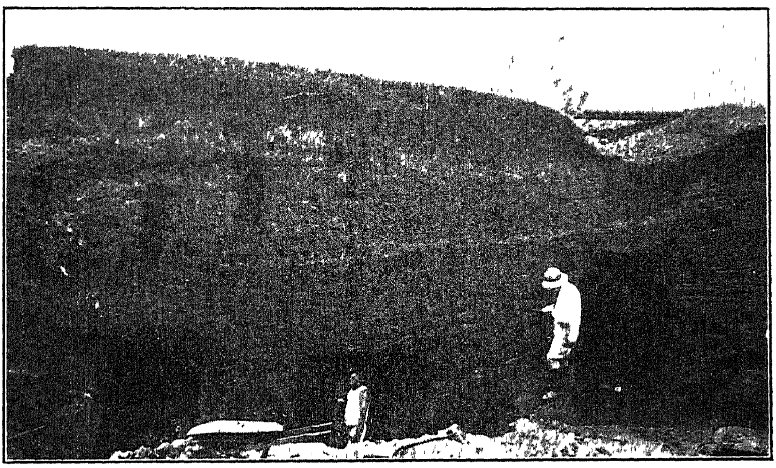

FIG. 6.—SABANILLA, CUBA. BEDDED MASS OF TUFF AND MANGANESE ORE AT LAURA MINE.

to 45 per cent. manganese), and about 39 tons of lower grade ore per day. Commercial analyses show the milling ore to carry from 26.5 to 31 per cent. manganese, 6 to 10 per cent. iron, and 27 to 32 per cent. insoluble. The ordinary shipping grade contains 38.5 to 39.75 per cent. manganese, 5 to 6.7 per cent. iron, and 14 to 16.25 per cent. insoluble. The highest

grade ore contains about 47.5 per cent. manganese, 2.7 per cent. iron, and 7 per cent insoluble.

*Laura Mine.*—The Laura mine of the Guantanamo Exploration Co. is situated about ¾ mi. (1.2 km.) north of Laura station on the Guantanamo & Western Railroad and about 1½ mi. northwest of Sabanilla station. The workings are situated on the brow of a hill at an altitude of about 600 ft. (182 m.) above sea level and about 250 ft. (76 m.) above the railroad level. The ore is mined from a large open cut which exposes, on the north side, masses and boulders of jasper partly replaced by manganese oxides and intermingled with clay and residual lumps and fragments of manganese ore; and on the south side, a bedded mass of tuffaceous material cemented by manganese oxide, constituting an ore of varying quality; see Figs. 5, 6 and 7.

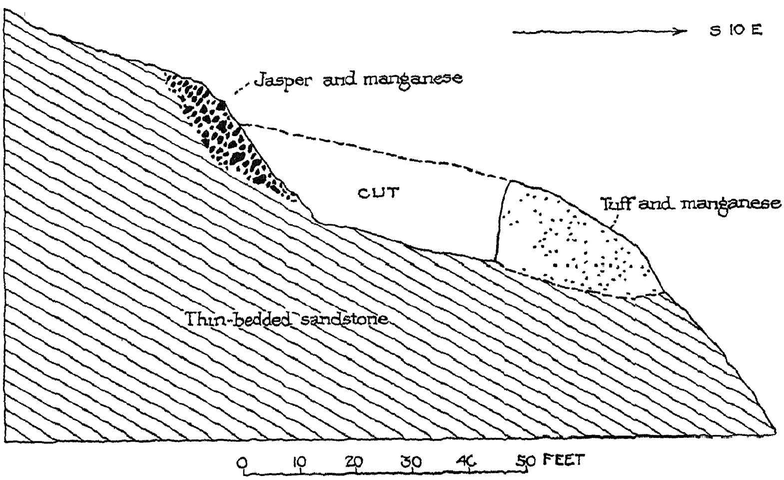


FIG. 7.—SECTION SHOWING RELATIONS OF MANGANESE-BEARING JASPER, TUFF, AND SANDSTONE AT LAURA MANGANESE MINE VIEWS IN FIGS 5 AND 6 SHOW THE TWO ENDS OF THIS CUT.

Streaks of light-colored sand and of green and pink clay are prominent in the bedded material and, as stated on page 60, the clay seems to consist largely of volcanic ash.

Debris of fine-grained white sandstone lies on the slope southwest of the open cut. The stratified mass of manganiferous ore, as blocked out, was about 30 ft. by 80 ft. by 15 ft. (9 by 24 by 4.5 m.) and was reported to average, in a natural state, about 19 per cent. of manganese in the upper part and 30 per cent. of manganese in the lower. Three drifts had been driven down the dip of this mass of ore about 15 ft. in search of richer material. A stock pile containing about 1800 tons of ore, derived from cobbing the material associated with the jasper, averaging about 38 per cent. manganese, 3.5 per cent. iron, 0.1 per cent. phos-

phorus, and 15 per cent. silica, was ready for shipment in March, 1918. The presence of barium has also been noted in this ore. The greater part of the richer and cheaply available surface ore is considered to have been removed at this mine.

*Ponupo Group.*—The mines of the Ponupo group are among the oldest and largest producers of manganese ore in Cuba, having made the first shipment in 1895. The present owners are Aguilera & Co., of Santiago de Cuba and New York City. These mines are situated about 18 mi. (28 km.) in an air line northeast of Santiago and about 2 mi. (3 km.) east of La Maya, the terminus of a branch line of the Cuba Railroad. A mine railroad connects Ponupo with the Cuba Railroad at La Maya. The maximum local relief appears to be about 200 ft. (60 m.) and de-

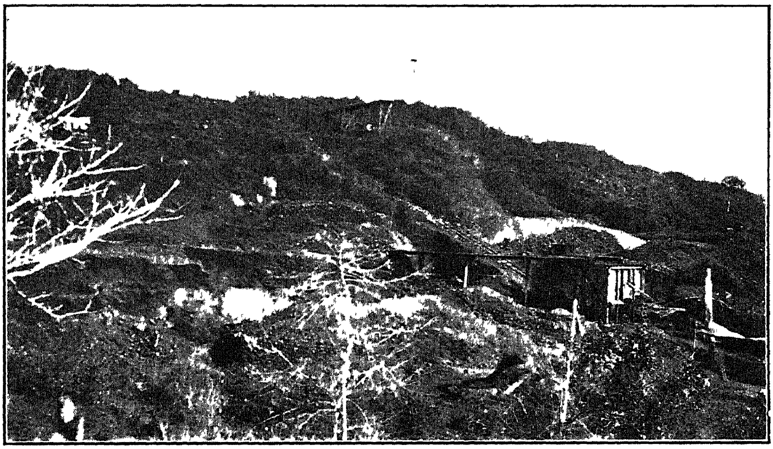

FIG. 8.—PONUPO, CUBA. GENERAL VIEW OF SOUTHWEST HILLSIDE WORKINGS AT VENCEDORA AND GENERALA MINES.

posits of ore have been opened at nearly all altitudes from stream level to the top of the hill. The general altitude above sea level, according to barometric readings, is 525 to 725 ft. (160 to 220 m.).

The Ponupo group consists of several openings distributed along an east and west belt about ¾ mi. wide and 2 mi. long. The principal openings are the Generala and the Vencedora, near the middle, the Balkanes and the Juanita at the west, and the Sultana at the east ends of the group.

Thick bedded, slightly metamorphosed limestone is here overlain by greenish to yellowish thin-bedded calcareous sandstone, interbedded with conglomerate and andesitic tuff. The conglomerate contains boulders of limestone and of volcanic rocks including tuff, porphyritic basalt, basalt porphyry, and andesite porphyry. The andesitic tuff contains abundant green glauconite or chlorite. The eroded surface of

the limestone is overlain in places by loosely consolidated tuffaceous, more or less stratified sandy material, and the sandstone and conglomerate beds also are overlain in places with similar tuffaceous beds. These tuffaceous beds generally contain more or less manganese oxide as a cement, probably as a replacement, and in places enough of it to constitute a milling ore. Masses of siliceous rock, or jasper, usually manganiferous, are associated with both the limestone and the sandstone.

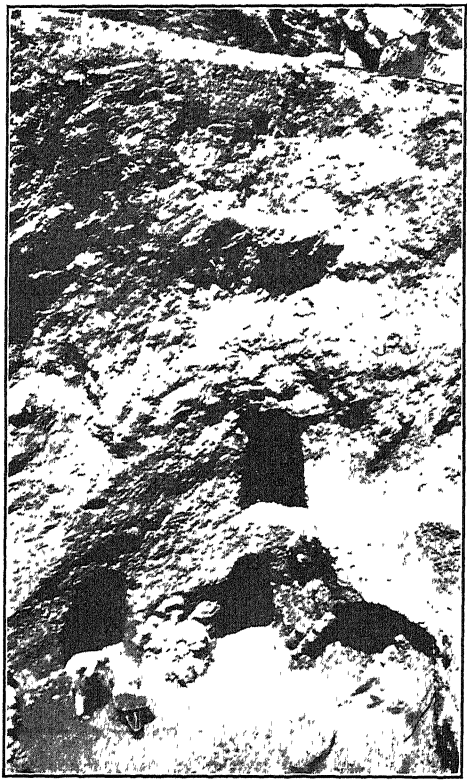

FIG. 9.—PONUPO, CUBA. GENERALA MINE, DEEP PIT IN MANGANESE-ORE POCKET BETWEEN MASSES OF JASPER AND LIMESTONE.

On account of its superior resistance to weathering, the jasper generally outcrops prominently and is often found capping hills and terraces. Unsymmetrical anticlinal structure is shown here in a broad way, but locally the rock dips are very irregular.

This mixed ore generally presents an irregular speckled or granular appearance, the granules ranging in size from mere specks up to ¼ in. (6 mm.) in diameter. In color, the granules vary from buff to pink and brown The pink areas naturally suggest a manganese mineral, and similar material at another mine was suspected of containing rhodonite,

but the material is too soft and does not carry enough manganese. Under a field lens, clear glassy minerals, probably zeolites, can be seen in the pink areas.

The three types of manganese-ore deposits and their several subordinate varieties are well displayed at the Ponupo group of mines. At the Generala openings, manganiferous tuffaceous material fills solution cavities and crevices in limestone, in some places, to depths of 30 to 40 ft. (9 to 12 m.), (see Fig. 10). At the Sultana opening, manganese oxides have replaced beds of agglomerate(?); see Fig. 12. This deposit is apparently a lens having a maximum thickness of about 20 ft. (6 m.), the upper half of which is reported to average about 34 per cent. in manganese and the lower half about 42 per cent. manganese. At the Juanita

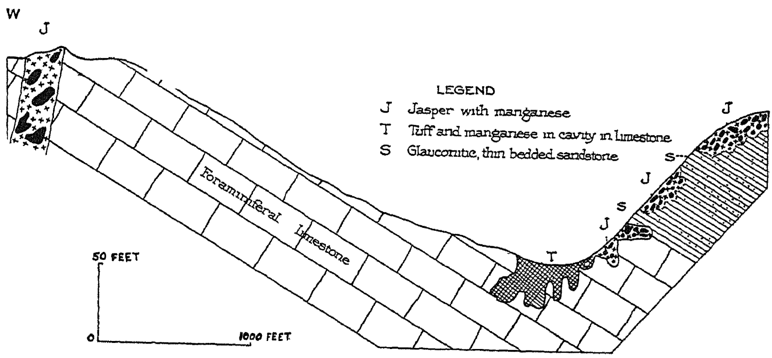


FIG. 10.—SECTION AT PONUPO MANGANESE MINES. FROM BALKANES OPENING AT THE WEST TO VENCEDORA OPENING AT THE EAST, SHOWING RELATIONS OF MANGANESE-BEARING JASPER MASSES TO THE LIMESTONE AND GLAUCONITIC SANDSTONE, ALSO CAVITY IN LIMESTONE FILLED WITH TUFF CEMENTED BY MANGANESE OXIDE.

opening, manganese has in spots replaced limestone. At the Balkanes opening, the ore is found associated with masses of jasper which are partly enclosed in massive, white to pink, medium-grained, algal limestone. The jasper may have replaced a portion of the limestone, but now stands out prominently due to the more rapid solution and weathering of the calcareous rock. The ore occurs in lumps and in earthy form in pockets and crevices in the jasper. The ore here is not generally of high grade and some of it is ferruginous. The limestone contains manganese oxides in spots and in veinlets along fracture planes in abundance near the manganiferous jasper, but in smaller quantities farther from it. A thin section of this limestone was examined by J. A. Cushman, who reports as follows: "Limestone with few foraminifera, two species of *Orthophragmina*, *Nummulites*, *Marginulina*, or Cristellaria. [Other sections I have seen from the Ponupo mine have shown conditions more like those of station 5 (Pilar mine) here reported]." At the Vencedora openings, manganese oxide partly replaces masses of jasper and also occurs in

lumps in residual clay in pockets between and adjacent to boulders of jasper.

Ore is mined chiefly from open cuts at Ponupo. At the Vencedora workings, on top of the hill, some ground is being worked that was passed over in the early days when only the richest ore was taken. Old dumps also were being worked here in 1918. At the Generala openings, which are on the southwest slope of the same hill, are many openings in the

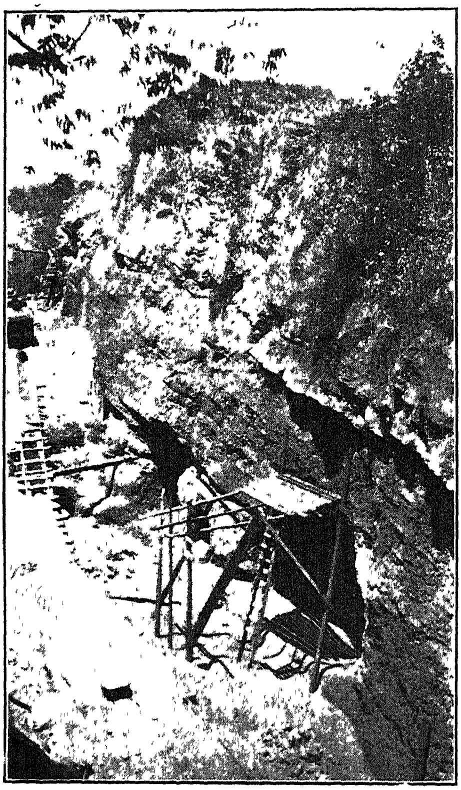

FIG. 11.—PONUPO, CUBA. CUT IN BED OF MANGANESE ORE, SULTANA MINE, LOOKING N. 65° W. ALONG FACE OF CUT.

outcrops of jasper (shown in Fig. 8), most of which are now abandoned, but most of the recent activity has been toward the base of the hill where several deep pits have been sunk in bedded and residual deposits, rich in manganese, which fill solution cavities in limestone to depths of 30 to 40 ft. (9 to 12 m.); see Fig. 9. There are also some short drifts that follow rich pockets of ore. The deposit at the Sultana has been opened along its strike by an open cut 150 ft. long and about 30 ft. deep, and by drifts about 30 ft. long down the low dip of the beds; see Fig. 11. All the important workings are connected by mule tramways and cable

inclines with a large double log washer where the low-grade ore is raised to 36–38-per cent. manganese and dumped into railroad cars.

The Ponupo group is the largest shipper of manganese ore in Cuba, from 3000 to 6000 tons a month having been shipped in the spring of 1918, the output being mainly taken by the Bethlehem Steel Co. Notwithstanding the past large production, there are evidently still available large reserves of low-grade ore that can be concentrated.

*Botsford Mine.*—The Botsford mine is situated about 3 mi. (4 km.) east-northeast of Cristo, and about ¾ mi. (1 km.) east of the Ysabelita mine The workings are at an altitude of about 600 ft (182 m.) above

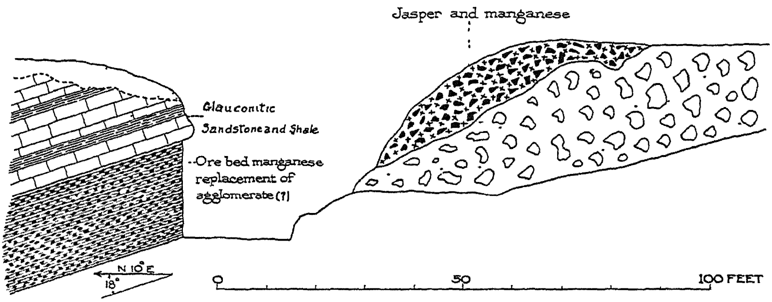


FIG. 12—SECTION AT SULTANA OPENING. PONUPO MANGANESE MINES, SHOWING RELATIONS OF ORE BED TO ENCLOSING ROCKS.

sea level and about 30 ft. (9 m.) above the nearest creek. In the spring of 1918, ore was hauled from here to the Cuba Railroad at Cristo in ox-carts and on mule back, over a road that was only in fair shape in the dry season. The Compania Mineral Bonanza, of Santiago de Cuba, operated the property at the time of visit

Locally the rocks consist of limestone interbedded with fine conglomerate containing volcanic debris. The dips are variable toward the southwest. The ore at the Botsford mine is, in part, detrital and, in part, of the bedded type. The detrital deposit appears to have been derived from the weathered parts of the bedded deposit. A small fault is shown in one of the open cuts. A fine-grained conglomerate containing volcanic debris is associated with the manganiferous bed, as exposed in open cuts on the flanks of the limestone capped hill. The detrital ore-bearing material exposed in a series of open cuts for a distance of about 200 ft. (60 m.) shows an average thickness of about 6 ft. (1.8 m.). It is coarse-grained and contains much granular pink, clay-like material, and other debris from andesite tuff and is mostly too low grade for shipment without concentration. The bedded, or tabular mass of ore, also contains grains of tuffaceous material and evidently represents a bed of tuff now in large part replaced by manganese oxide. It is 1 ft. to 2½ ft. thick, and follows the bedding of the limestone, which dips

about 35° S. 18° W. An incline driven down the dip for a distance of about 40 ft. was still in ore, and drifts turned off at the right and left indicate that the deposit extends at least 20 ft. on the strike of the beds. The ore in this mass is much richer than that mined from the open cut and is reported to carry about 42 per cent. manganese. Water is encountered at a depth of about 20 ft., but the manganese oxides continue in depth so far as explored.

The low-grade ore at the Botsford was being washed by raking it over a horizontal screen in a stream of water. About 8 tons a day was being made at this mine and there is still a small reserve of ore in sight.

*Ysabelita Mine.*—The Ysabelita mine, operated by Aguilera & Co., is situated about 2 mi. (3.2 km.) N. 80° E. of Cristo station. The mine

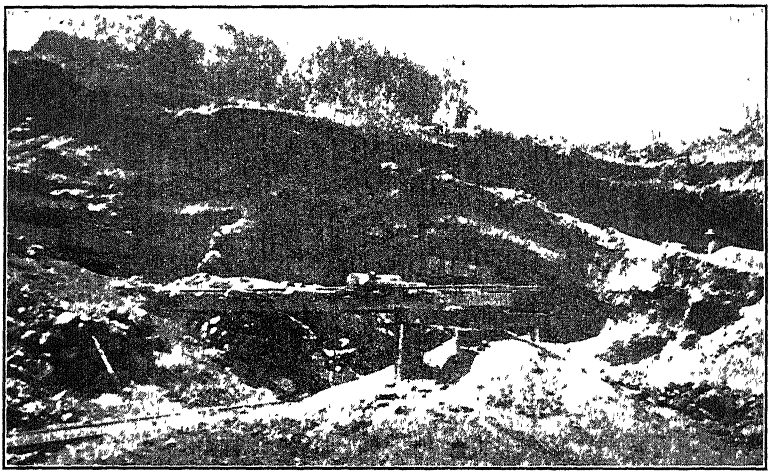

Fig. 13.—Three miles east of Cristo, Cuba. Ysabelita manganese mine, cut near top of hill showing folded (?) beds of glauconitic and tuffaceous sand impregnated with manganese minerals. Toward top boulders of jasper are included. Certain red bands of oxidized glauconitic sand are barren of manganese oxide.

openings are on the top and slopes of a hill that rises approximately 200 ft. (60 m.) above the local drainage level, or from about 550 to 750 ft. (167 to 228 m.) altitude. Outcropping ledges of jasper constitute the principal exposures of rock on this hill, the sedimentary beds being so decomposed as to afford almost no outcrops. The ore consists of manganese oxides that have replaced the jasper, of residual lumps and earthy material in clay between boulders of jasper, and of loosely consolidated sandy, tuffaceous, or brecciated material that is found in places on the uneven surface of the jasper at various points on the hill slope from base to summit; see Fig. 13. The deposits at this mine are very similar to those of the Vencedora and those of the Generala on the slope below the Vencedora at Ponupo, except that possibly there is more of the buff to

pink granular bedded type of ore present. Most of this ore is rather low in grade, at one pit carrying only about 30 per cent. of manganese, but it is concentrated in log washers so as to carry 36 to 38 per cent. of manganese. The dips of the jasper masses exposed on this hill and those of loosely consolidated granular manganiferous beds are so variable that but little indication is afforded as to whether or not the structure is anticlinal.

Many openings have been made on this property, which is one of the oldest mines in the district, and much ore has been removed, but there is still considerable low-grade ore available. The mine openings are connected by tramways with a cable incline which moves the ore down to the washer. Prior to July 1, 1918, ore was being hauled by wagon or packed on mules to the railroad at Cristo, but in the summer of 1918 a narrow-gage track was completed to these mines, which has greatly facilitated the shipment of ore and increased the production.

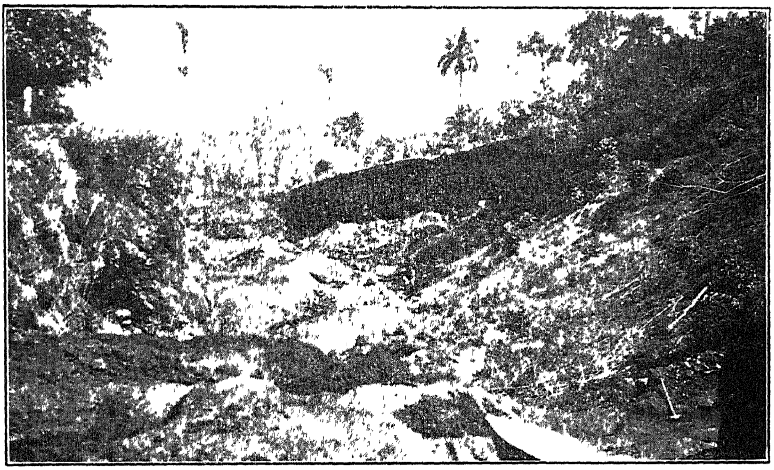

FIG. 14.—THREE MILES SOUTHEAST OF CRISTO, CUBA. BOSTON MINE, LONG OPEN CUT; GLAUCONITIC MANGANIFEROUS BEDS TO RIGHT AND JASPER TO LEFT.

*Boston Mine.*—The Boston mine, operated by Santiago parties, is situated about 2.5 mi. (4 km.) S. 75° E. of Cristo and about 1 mi. S. 15° E. from the Ysabelita mine, the two mines being separated by Guaninicum River. This mine is also situated on a hill, the rocks of which are apparently anticlinal in structure. The manganese-ore deposits at the Boston are between 600 and 750 ft. (182 and 228 m.) above sea level and from 100 to 250 ft. (30 to 76 m.) above the level of the river.

Outcrops of crystalline limestone were noted on the hill slope below the main manganese workings, but higher than some deep prospect pits in which manganese had been found. The principal working is an open cut about 300 ft. (91 m.) long, which discloses manganese ore in masses

and pockets associated with glauconitic (?) sand and tuffaceous debris deposited between boulders and ledges of jasper; see Fig 14. The ore here, apparently, is principally detrital although the presence of some stratification was noticed The deposit gives promise of considerable production, if worked with care.

At the time of visit, a hoisting engine and an incline were being installed and a log washer was nearly completed. Mining was active here many years ago but work had only recently been resumed, since which time 500 tons of hand-cobbed and dry-screened product were reported to have been shipped. Many years ago a railroad was built from the Boston mine to Cristo and several thousand tons of ore were shipped. Later the railroad was torn up; and in the spring of 1918 ore had to be hauled in carts or carried on mules to Cristo. There seemed a possibility, however, that arrangements might be made for connections with the narrow-gage railroad running from Cristo to the Ysabelita mine, in which case a larger output would be possible.

*Pilar Mine*.—The Pilar mine, operated by Aguilera & Co., adjoins the Boston property on the east. The openings are near the top of the hill at an altitude of nearly 800 ft. (243 m.) Boulders of jasper and irregular masses of crystalline, pink to white foraminiferal limestone, in places showing green bands, are exposed in the mine workings. There is also considerable powdery white clay present in streaks and masses, probably residual from the decomposition of crystals of feldspar in the limestone.

Thin sections of this limestone have been examined by Miss E. F. Bliss and E. S. Larsen. Miss Bliss reports as follows: "A medium-grained pink limestone showing small white crystals of which some are calcite and some feldspar. The rock is filled with grains of a green chloritic mineral. In thin section, it is seen to be composed of microscopic organisms, chiefly a small foraminifer. * * * The organisms are frequently surrounded by a narrow rim of calcite or a green chloritic mineral. The spaces between the individuals are filled with small angular fragments of quartz-feldspar, or fragments of a volcanic rock containing feldspar laths in a dark-colored glassy ground-mass. It appears that particles of volcanic material have been blown into an aggregate of foraminiferal organisms and later cemented into place by the action of calcite and a green mineral probably chlorite. Some iron oxide and a small amount of dense, black mineral, probably manganese oxide, have been deposited along with the filling material of the rock." A thin section of this limestone was examined by J. A. Cushman, who reports as follows. "Limestone almost entirely composed of species of *Orthophragmina*, mostly of one species. Also has sections of *Nummulites* and *Linderina* (?) but these comparatively rare. This is an Eocene assemblage and the great abundance of the *Orthophragmina* indicates that the deposit was formed in water of but a few fathoms depth."

E. S. Larsen makes the following comment on the section of this rock. "Mostly calcareous remains of some organism, but between these remains are a few grains of plagioclase, quartz, andesitic rock, and zeolite. It is probable that a small amount of volcanic sand was dropped from the air, by explosive volcanic activity, into the water in which the organisms lived."

The manganese ore occurs in lumps and boulders, in layers of clayey ground 6 to 15 ft. (1.8 to 4 5 m.) deep, and in large pockets among the boulders of jasper and the masses of limestone. The workings consist of three pits 40 to 80 ft. (12 to 24 m.) long on the eastern slope near the top of the hill. At the time of visit, about 10 per cent. of the total material removed was ore carrying 40 per cent. or more of manganese; about 30 per cent. is wash ore. Where prospected, the property is reported to have shown encouraging deposits of ore and a log washer had been ordered. It was expected that this mine would be connected with the narrow-gage railroad between Cristo and the Ysabelita mine, and the prospects for ore tonnage seemed to justify this additional outlay provided the market for ore remained satisfactory.

*San Andreas Mine.*—The San Andreas mine is situated about 2 mi. (3 km.) northeast of the station of Jutinicum on the Guantanamo & Western Railroad. This mine was operated, at the time of visit, by the Compania Miner Jutinicum of Havana but is reported to have changed hands since then It consisted of a group of seven or eight claims, comprising in all about 250 ha., which had not been fully explored. The workings consisted of five or six openings on the north side of a hill at an altitude of 600 to 650 ft. (182 to 198 m.). On the south slope of the hill, glauconitic and calcareous sandstone beds are exposed in long dip slopes; and on the north and northeast slopes, there are outcrops of jasper and artificial exposures of bedded manganiferous tuffaceous sand. At the top of the hill, 100 ft. higher than the bedded deposits, are residual masses of jasper containing seams and pockets of rich manganese oxide replacing the jasper. A great deal of float manganese ore is exposed on the north slope of this hill and many prospect pits show ore associated with boulders and ledges of jasper and disseminated in specks, nodules, and solid seams in sandy strata; there is also some rich ore in the residual soil and clay.

Surface work had not proceeded very far at the time of visit but crude methods of dry screening were yielding a product carrying about 42 per cent. of manganese. About 500 tons of ore had been carted from here to the station at Jutinicum, but large-scale operations will require a log washer or other concentrator, for which purpose water will have to be pumped 1 to 2 mi. (1.6 to 3.2 km.), depending on the quantity required. About 4 tons of ore a day were being produced here at the time of visit with a force of 34 men. The evidence of ore at the surface and in prospects indicates that an important tonnage may be found in this locality.

*Abundancia Mine.*—The Abundancia (Cauto) mine is situated at Manganeso station on the Cuba Railroad between San Luis and Palmarito, and is operated by the Cauto Mining Co. of Santiago de Cuba, and New York City. It is situated on a terrace about 60 ft. (18 m.) above Guaninicum River a few miles above its junction with Cauto River. The rocks exposed are yellowish, calcareous glauconitic sandstones, enclosing a large lens of manganese ore, see Fig. 15, which attains a thickness of 35 to 40 ft. (10 to 12 m.) as exposed in two open cuts. The beds dip 30° to 40° southwest. The ore-bearing material consists chiefly of manganite and psilomelane intimately mixed with grains and streaks of a pink clay-like material, soft enough to be scratched by the finger nail, and fragments of tuff. A little secondary calcite is present. The pink mineral was considered by the operators of the mine to be rhodonite, or manganese metasilicate ($MnOSiO_2$), and was being largely saved in the concentrates. A microscopic examination of the pink material by E. S. Larsen indicates that it is chiefly kaolinite or a related clay containing clear grains of zeolitic minerals. A chemical analysis of the pink clay-like material from which most of the black oxide of manganese had been picked showed only 5.79 per cent. of manganese, while a fairly representative piece of the mixed black and pink ore contained 30.23 per cent. of manganese. This pink mineral is also common at many other mines, such as the Ponupo, Laura, and Charco Redondo The ore-bearing lens shows stratification, which suggests that it was originally deposited as a tuff in water. The manganese oxides have probably replaced certain of the original minerals in the bed and have formed a cement in the interstices. The limit of the deposit down the dip is not indicated in the open cuts, but on account of its bedded character it may extend as far as it does along the strike, which would indicate large tonnage. In its natural state, the ore is reported to average about 34 per cent. of manganese, and to contain a few per cent. of barium.

The ore-bearing beds outcrop at the surface and have been opened for a distance of 750 ft. (228 m.) on the strike by two pits connected by a tunnel and a drift is driven some distance beyond the southeast pit; see Figs. 15 and 16. The pits are about 50 ft. (15 m.) wide and 30 ft. (9 m.) deep and are equipped with tramways and a cable incline to convey the ore to the mill nearby; see Fig. 17. The ore is concentrated by crushing, wet screening, and jigging, which was reported to bring the shipping product up to between 40 and 47 per cent. of manganese with the tailings carrying 18 per cent. of manganese. The concentration ratio was about 4 to 1. Most of the loss in the tailings was in material passing 16-mesh screens, so it was planned to further concentrate this material on tables. If improvements in concentration can be effected so as to save more of the tailings, it ought to be possible to make this mine one of the steady producers of manganese ore even after the return of normal conditions.

The main mine and mill of the Cauto Mining Co. are situated directly on the railroad and there is abundant water available for milling purposes.

About 1½ mi. (2.4 km.) east of Manganeso, along the crest of a ridge, residual deposits of manganese ore in clay and boulders of jasper have

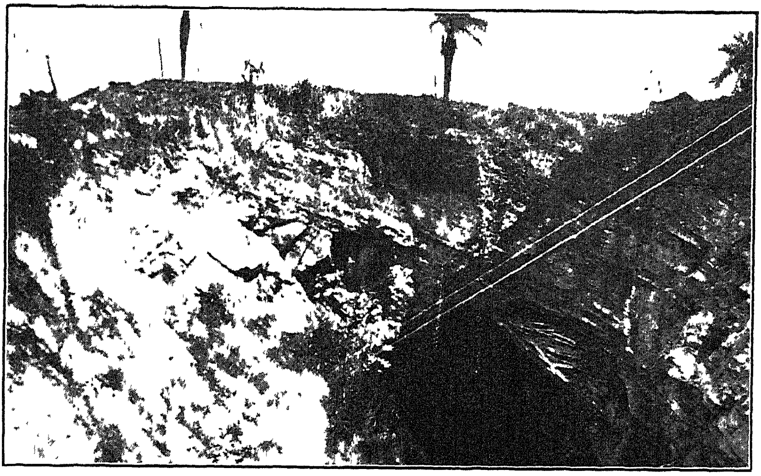

FIG 15.—MANGANESO, CUBA. CAUTO MINING CO., DRIFT SOUTHEAST FROM OPEN CUT, SHOWS WEDGING OUT OF ORE-BEARING LENS AT LEFT ABOVE TUNNEL.

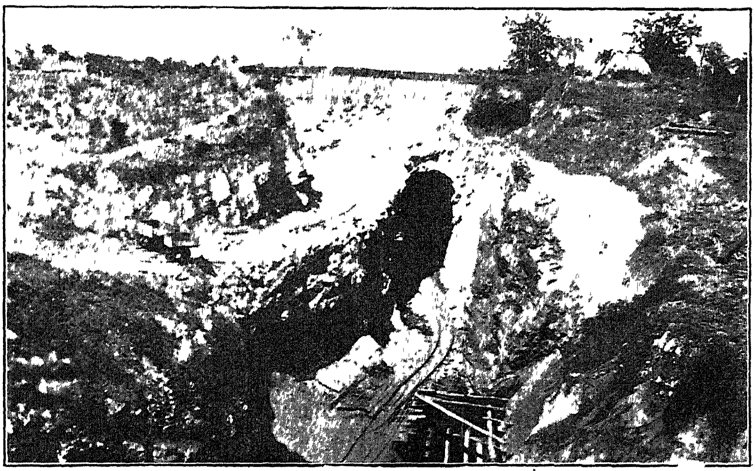

FIG. 16.—MANGANESO, CUBA. CAUTO MINING CO., TUNNEL EXTENDING NORTHWEST FROM OPEN CUT, ON STRIKE OF MANGANESE-ORE LENS.

been opened by the Cauto Mining Co. Some of this ore is said to average more than 45 per cent. of manganese.

*Llave Mine.*—The Llave mine of the Campania Mineral Bonanza is situated about 1½ mi. (2.4 km.) northeast of Manganeso station and

about $\frac{3}{4}$ mi. from the Cuba Railroad. The deposit lies on the slope of a hill that is part of the ridge on which the Cauto Mining Co. has opened its manganese-ore deposit in jasper, or bayate, rock east of the railroad. The ore is found from 25 to 150 ft. (7 to 45 m.) above the local drainage. The local rocks are limestone, shaly to thin bedded yellowish sandstone, and irregular masses of jasper associated with both. The manganese ore is principally associated with the jasper and limestone and is found in lumps and fragments in pockets in residual clay between masses and boulders of jasper, and in cavities in the surface of the limestone. The latter type is sandy bedded material, probably a water-laid deposit derived from a tuffaceous bed formerly inclosed in the limestone.

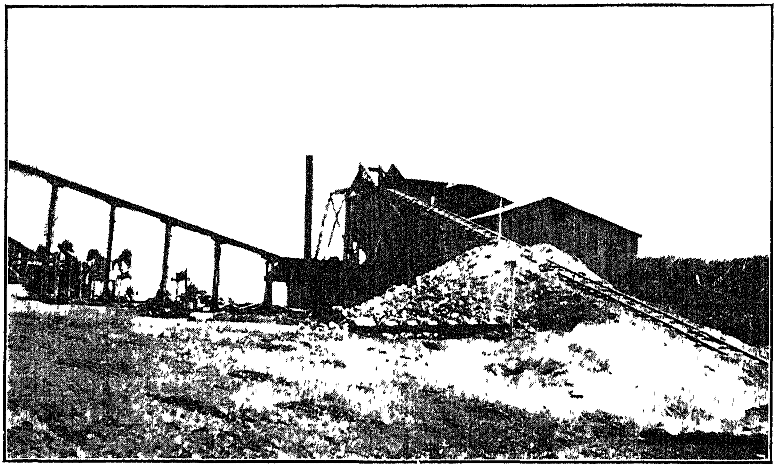

FIG. 17.—MANGANESO, CUBA. CAUTO MINING CO., MANGANESE-ORE CONCENTRATING MILL.

Ore was being mined here from several shallow open pits and trenches in the hillside, which had only recently been partly cleared of timber. A single log washer was employed and it was reported that the washed ore averaged about 40 per cent. manganese. A small tonnage of ore had been indicated by test pits over an area of 10 acres (4 ha.), and shipments of about 1300 tons were reported to have been made to Mar. 1, 1918. This mine has the advantage of a road with a rock bottom, practically down grade to the railroad, so that shipments should be possible even in the rainy season.

*Gloria Mine.*—The Gloria mine, at the time of the visit, was operated by Capt. W. F. Smith of Havana. It is situated about $7\frac{1}{2}$ mi. (12 km.) northeast of the Cuba Railroad station of Palmarito; see Fig. 18. This property occupies moderately sloping ground in the foothills at the south end of the Sierra de Nipe, at an altitude of 500 to 600 ft. (152 to 182 m.). The rock underlying the hill slope is medium-grained, heavy-bedded,

glauconitic limestone, much pitted by solution; on the top of the hill covered by the property is a thin bedded chalky limestone. There is a little chert in places, residual from the lower limestone. The rocks dip gently about S. 20° W.

FIG. 18.—EIGHT MILES NORTHEAST OF PALMARITO, CUBA. OPEN CUTS AND DUMPS AT LA GLORIA MINE, FROM SOUTH. SHOWS RECENTLY CLEARED SURFACE OF ORE-BEARING GROUND.

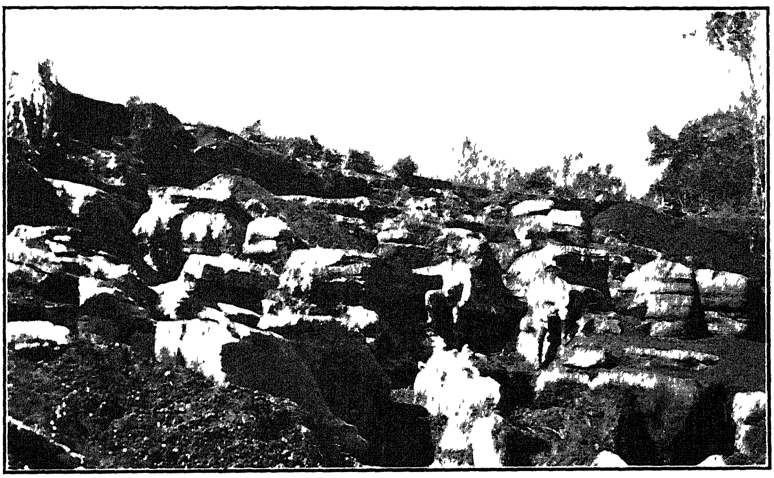

FIG. 19.—EIGHT MILES NORTHEAST OF PALMARITO, CUBA, AT GLORIA MINE GLAUCONITIC LIMESTONE BEDS SHOWING SOLUTION CAVITIES IN WHICH MANGANESE ORE OCCURS.

The manganese ore occurs here as detrital material with clay in the solution cavities and channels between masses of glauconitic limestone, and as stratified sandy tuffaceous deposits on the uneven surface of the

limestone. One of these bedded deposits is near the top of the hill. It is 4 to 8 ft (1.2 to 2.4 m.) thick, lies unconformably on the old limestone surface, and is overlain by several feet of thin bedded limestone that weathers chalky The cavities in the lower limestone range from a few feet to more than 10 ft. in width and depth and a considerable tonnage of high-grade manganese ore has been obtained from them; see Fig. 19 The manganese oxides they contain may have been derived from the higher tuffaceous sandy beds through settling of material into caves in the limestone as they were formed below the surface, and the manganese content has probably been increased by deposition from surface waters. The masses of limestone adjacent to the pockets of rich ore-bearing clay have been noticeably impregnated, in places, with manganese oxides, but not richly enough to constitute an ore.

A thin section of this limestone was examined by J. A. Cushman, who reports as follows: "Limestone with a varied foraminiferal fauna, all apparently specifically different from station 5 (Pilar mine). Numerous specimens of two or more species of *Orthophragmina*, at least two of *Nummulites*, *Carpenteria*, *Globigerina*, and *Marginulina* (?).

"A slightly deeper water condition is indicated than by the preceding station (Pilar mine) but no considerable difference. There is on the other hand enough difference indicated either in time or ecologic conditions to give an entirely different facies "

The upper stratified bed of sandy manganiferous material has been opened along the strike for a distance of 250 ft. (76 m.) and it has been prospected by a drift on the dip for 12 ft. (3.6 m.). This material is reported to carry about 30 per cent. of manganese in the natural state and would therefore require washing and jigging. If this were practicable, it might be stripped and worked as an open cut. Water, however, is not abundant here. Much of the detrital ore from the pockets in the limestone is of high grade and is reported to carry about 55 per cent. manganese; a considerable amount of such ore has been screened, hand-sorted, and sold as chemical ore.

The workings occupy an area of about 600 by 1000 ft. (182 by 304 m.) which apparently has been about three-fourths worked over. About 20 tons a day of high-grade ore were being produced in the spring of 1918. Since the property was visited, it is reported that a new and better wagon road has been built to a sugar plantation railroad about 4½ mi. distant.

*Production and Ore Reserves.*—The production of manganese ore about the middle of March, 1918, in the Santiago district was between 280 and 300 tons a day. The output was curtailed in the rainy season, which begins about the first of June, especially that from the smaller mines, which are dependent on ox-cart haulage, but the curtailment was more than offset by the increase in shipments after the railroad from Cristo to the Ysabelita mine was opened. The approximate average composition of

a large proportion of the ore recently shipped for metallurgical use is as follows: Manganese, 38.885 per cent.; silica, 12.135 per cent.; phosphorus, 0.084 per cent.; moisture, 11.201 per cent.

As each deposit of manganese ore was examined, an estimate was made of its probable reserve ore tonnage and of its probable production of ore, by grades, during the years 1918 and 1919, provided the war-time demand for ore should continue and the unusual handicaps to production and transportation of ore could be overcome. With the estimate of probable production there is now no concern, but it should be stated that the manganese-ore producers of Cuba were prepared to fulfill even more than was expected of them and were, up to the signing of the armistice, in a fair way toward realization of the goal set for 1918 even though unduly handicapped

In regard to the reserve-ore tonnage there is a certain interest in connection with the question of world distribution of this important steel-alloy metal, although until conditions adjust themselves once more there may be little demand for manganese ore from Cuba. In estimating ore reserves in a period of strong demand, some ore that would be worth recovering under such conditions, but not under normal conditions, is apt to be included if the estimates are made on a liberal basis. The present estimates of reserves are, however, believed to have been made on the basis of normal mining and milling practice. The nature of the manganese-ore deposits in Cuba is not such as to render the deposits susceptible to accurate quantitative estimation. In few instances are there sufficient geologic or mining data available to enable one to block out a definite and certain tonnage of ore, so that the estimates do not deal with certainties. On the other hand, they do not go so far as to include as much ore as is speculatively possible under the indicated conditions. In making the estimates by individual deposits, therefore, the probable ore, rather than the ore in sight or the possible ore, has been taken as the basis of the estimate and practically all of it has been figured on the basis of metallurgical grade, that is, ore containing 36 per cent. or more of manganese. This necessarily involves the recovery and concentration of considerable ore of lower grade.

In making estimates such as these the nature of the deposit and the history of its mining development and production furnished much information for guidance. Spencer,[7] who examined these deposits in 1901, giving consideration to the mode of occurrence of the deposits and the records of the industry to that date, believed that the total available high-grade ore in any one deposit could not be expected to greatly exceed 100,000 tons. So far as individual mines are concerned, it is doubt-

[7] Report on a Geological Reconnaissance of Cuba. U. S. Geol. Survey *Bull.* 213 255.

ful if any one of them has yet produced more than that quantity of high-grade ore, although it would appear that the output of at least one of them must have approached that quantity, when it is considered that prior to the European war the total output from Cuba, which came mainly from a few mines, had amounted to nearly 300,000 tons of high-grade ore. With these accomplishments in view, however, it still seems probable that much more than 100,000 tons of ore (including concentrates) are available in certain of the well-known deposits.

Details by mining properties of estimated tonnages cannot, of course, be given. The total probable reserves of merchantable manganese ore of metallurgical grade in the district near Santiago are estimated as between 600,000 and 700,000 long tons.

*Representative Mines and Prospects in District South of Bayamo*

The manganese deposits that were examined in the district south of Bayamo consist of the Manuel, the Costa group (Costa, Carbayon, Daniel, Oviedo, Vicente, and other claims), 18 to 23 mi. (28 to 37 km.) by wagon road southwest of Bayamo; the Francisco and Cadiz, 15 to 20 mi. (24 to 32 km.) southeast of Bayamo; the Guisa, Llego, and Charco Redondo, 7 or 8 mi. (11 or 12 km.) southeast of Santa Rita; and the Adriana

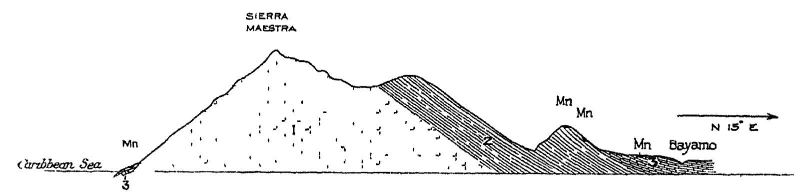


FIG. 20.—SECTION IN ORIENTE PROVINCE FROM CARIBBEAN SEA TO BAYAMO. LENGTH 35 MILES; VERTICAL SCALE EXAGGERATED. 1, INTRUSIVES; 2, VOLCANIC FLOWS AND TUFFS, 3, LIMESTONE BEDS WITH INTERCALATED AGGLOMERATE AND TUFF BEDS.

and San Antonio mines, 9 to 10 mi. (14 to 16 km ) south of Baire. Other deposits, farther southeast, are in what is known as the Los Negros district. All the deposits named are in the northern foothills of the Sierra Maestra; see Fig. 20.

The ores at the west end of the district, in the Manuel and Costa groups, occur chiefly in irregular masses of "jasper" or "bayate" associated with latite-porphyry, and those of the other properties farther east are associated chiefly with limestone and comprise bedded and residual deposits.

*Costa Group.*—The Costa group of manganese-ore claims consisting of the Costa, Carbayon, Barrabeitj, Vicente, Oviedo, and Daniel are situated about 15 mi. (24 km.) in a direct line S. 20° W. of Bayamo, and

6 to 8 mi. (9 to 12 km.) a little east of south of Bueycito. With the exception of the Daniel, all the claims are on the northeast or right-hand side of Buey River. This group of claims was being prospected in the spring of 1918 by the Sun Development Co., of Philadelphia, and Trumbo Brothers, of Havana. The country rock in this locality is of igneous origin and consists of tuff, agglomerate, and latite, with latite-phonolite and latite-porphyry phases. Limestone is reported also to occur here. Masses of jasper, locally known as bayate, are associated with these igneous rocks much as they are with limestone and sandstone northeast of Santiago. There are also remnants of stratified tuffaceous, sandy deposits, more or less cemented by manganese oxides, overlying the igneous rocks.

FIG. 21.—FIFTEEN MILES SOUTH-SOUTHWEST OF BAYAMO, CUBA. MANGANESE-ORE BOULDERS ON BUEY RIVER SLOPE, COSTA CLAIM.

The Costa claim, which covers 209 ha., lies along an east-west ridge north of Buey River, the manganese-ore deposits having been found chiefly along the summit and down the south slope, at altitudes from 600 to 800 ft. (182 to 243 m.). Along the summit of the ridge there are many small knobs capped by jasper; six of these have been found to carry manganese ore replacing the jasper. Float boulders of jasper more or less replaced by manganese ore are thickly distributed at one place on the south slope of the ridge. Some of these boulders, the exterior of which was almost wholly of manganese oxide, are 4 to 6 ft. (1.2 to 1.8 m.) in diameter; see Fig. 21. Associated with the jasper replacement deposits are masses of lean granular ore that may have resulted from cementation by manganese oxide of grains of feldspar from disintegrated porphyritic beds.

Much prospecting has been done on the Costa claim with encouraging results, but no ore had been shipped up to April, 1918. Most of the prospecting had been done by sinking test pits and cutting trenches, but a number of drill holes had been sunk along the summit and on the north slope of the ridge through surface soil and rock debris to depths of 8 to 13 ft. (2.4 to 3.9 m.); it was reported that many of these holes had reached manganese ore at the bottom. Most of the ore disclosed is of high grade but a small proportion will need concentration. The Buey River, which carries a never-failing supply of water, is reported to afford water-power possibilities.

Some actual mining of manganese ore was in progress on the Carbayon claim of 20 ha., about ½ mi. (0.8 km.) east of the Costa. Here a mass of jasper on a hill sloping eastward to Arroyo Macanocu has been cleared over an area about 100 ft. by 150 ft. (30 by 45 m.) and of this

Fig. 22.—Fifteen miles south-southwest of Bayamo, Cuba. Open-cut manganese mine, Carbayon claim. Shows masses of "bayate" or jasper surrounded by manganese ore. Miners are hand-cobbing ore.

an area about 50 ft. square had been partly worked over by breaking up boulders of jasper and hand cobbing lump ore; see Fig. 22. Shipments of high-grade ore for chemical purposes were being made by pack mules in the spring of 1918.

Promising prospects were noted also on the Vicente and Oviedo claims, in which the ore replaced jasper, or bayate. This area is practically virgin ground and the quality of the available ore has not yet been lowered by mining. The natural tendency in mining will be to take the cream of the deposits first, which will of course make it more difficult to recover ore of lesser value.

Transportation of the ore to the railroad offers the most serious problem affecting developments. In July, 1918, ore was being shipped in bags by pack mule 10 mi. (16 km ) to a highway, over which it was carried by motor truck 13 mi. to the railroad at Bayamo In the rainy season this highway became practically impassable for anything but ox-carts so that the movement of ore was further handicapped

TABLE 1.—*Analyses of Manganese Ore from Costa Group of Claims*

| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 |
|---|---|---|---|---|---|---|---|---|
| Mn . . | 58 40 | 37.40 | 53 40 | | | | | |
| Fe . | 1 03 | 3 98 | 0 40 | 0 29 | 0 24 | | | |
| $SiO_2$ | 1 43 | 18 10 | 11 20 | | | 14 16 | 7 20 | 10 15 |
| P. ... | 0 081 | 0.040 | 0 031 | | | 0 031 | 0 04 | 0 057 |
| Cu. . . | | | 0 008 | 0 021 | 0 032 | | | |
| $MnO_2$ | | | 81 03 | 86 54 | 89 7 | 63 68* | 61 28* | 64 28* |

* These percentages were shown in the original report as manganese, which is obviously impossible.

The analyses given in Table 1 were furnished by Howard Trumbo of Havana. They were made as follows.

1, Carbayon claim, analysis by Cambria Steel Co., Johnstown, Pa.; 2, Costa claim, milling ore, analysis by Cambria Steel Co., Johnstown, Pa.; 3, Carbayon claim, analysis by Ricketts & Co.. New York City; 4, Carbayon claim, analysis by Manhattan Electrical Supply Co.; 5, Carbayon claim, analysis by Manhattan Electrical Supply Co ; 6, Costa claim, average from seven pits, authority W. M. Courtis; 7, Oviedo claim, average from two pits, authority W. M. Courtis; 8, Daniel claim, authority W. M. Courtis

*Manuel Mine.*—The Manuel mine is situated about 15 mi. (24 km.) in a direct line S. 20° W. of Bayamo and 5 mi. south of Bueycito. At the time of visit this mine was operated by A. L. Colby, of Havana. The claim, which comprises 82 ha., lies on the top and slopes of a hill that forms the western terminus of the ridge northeast of Buey River and it adjoins the Costa claim on the west The rocks exposed on this claim are similar to those noted at the Costa group, latite-porphyry being characteristic of much of the material. The manganese deposits are found partly replacing jasper and as lumps and boulders in clay at the foot and on slopes of the hill, and also in a deposit of granular stratified material, in which fibrous crystalline manganese oxide is the principal cementing substance. Minerals characteristic of the latite-porphyry may be recognized in this deposit, and some hand specimens from it suggest partial replacement of the porphyry by manganese oxide; but

taken as a whole the appearance of the material suggests a water-laid deposit. Such ore as this should constitute good milling ore, and a deposit of it in a depression on the top of the ridge has been opened by a cut 40 ft. (12 m.) long, 10 ft. (3 m.) wide, and 15 ft. (4.5 m.) deep. This deposit is not large enough, however, to warrant installation of a washing plant. Several hundred tons of ore, derived chiefly by cobbing from lumps and boulders in clay on the west slope of the hill, had been shipped in the spring of 1918, but the output is certain to be limited until better transportation facilities are afforded. Conditions in this respect are similar to those of the Costa group already described

*Francisco Mine.*—The Francisco mine, operated by Messrs. Thomas & Hanover, of Bayamo, is situated about 13 mi. (20 km.) S. 25° E. of Bayamo, on the left bank of Boca de Guama River about ¼ mi. above its confluence with Bayamo River. The mine is about halfway up a steep bluff at a height of about 60 ft. (18 m.) above the river and at an altitude of about 600 ft. (182 m.) The river here flows in a gorge cut through thick bedded crystalline limestone. The rocks exposed in this locality are sandstone, shale, and limestone that have been much disturbed by folding and faulting. The manganese ore occurs as a replacement of massive and brecciated limestone beds The beds are nearly vertical and strike a little west of north. The replacement is irregular and incomplete, but in places has proceeded entirely across certain beds. The manganiferous zone, as exposed in the open cut, is about 40 ft. (12 m.) in its vertical dimension and has been opened about 75 ft. (22 m.) from northwest to southeast and about 20 ft. (6 m.) across the beds in the middle where the quarry is the widest. The distance beyond the quarry face to which the manganese deposition has reached has not been determined. The ore is compact and much of it has to be shot down in lumps, as in quarrying, but its component minerals are not very hard and appear to consist largely of pyrolusite and manganite, but even where replacement has apparently been thorough the ore is calcareous. The analyses given in Table 2 were furnished by Howard Trumbo. Analysis 1 was made by the National Carbon Co., Brooklyn, N. Y., and analysis 2 by A. J. Trumbo, Bueycito, Cuba.

TABLE 2.—*Analyses of Manganese Ore from Francisco Mine*

| | 1 | 2 |
|---|---|---|
| $MnO_2$ (equivalent in Mn, 53.36 per cent.) | 84 4 | |
| Fe | 0 77 | 2 05 |
| Cu | 0 0 | 0 11 |
| N (as nitrates) | 0 0 | |
| $CaCO_3$ | | 2 84 |

The Francisco deposit is worked as an open cut and the product is lowered by bucket and cable to a small wharf on the river bank, from which it is loaded on a motor truck and trailer, which stand in about 18 in. (45 cm.) of water. The distance to Bayamo is about 16 mi. (25 km.) and it is necessary to ford Bayamo river several times on the way. The transportation of ore is therefore not possible in seasons of high water. Other deposits of manganese ore of similar nature are reported to occur in this vicinity.

*Cadiz Mine.*—The Cadiz mine, controlled by J. E. Barlow, of Havana, is situated about 15 mi. (24 km.) S. 30° E. of Bayamo and 2½ mi. east of Guama. The deposits are situated at an altitude of about 1250 ft. (381 m.) on a high ridge about 100 yd. (91 m.) from a precipice overlooking Rio Giesa. The country rock here is massive gray limestone of the upper Eocene age. A thin section of this limestone was examined by J. A. Cushman, who reports as follows: "Limestone with very few foraminifera: *Globigerina, Nodosaris* (?) and *Textularia* (?). A somewhat deeper water condition is indicated (than at Pilar, Ponupo, and Gloria localities)."

The manganese ore has been formed by replacement of the limestone and occurs as large masses and as crystalline crusts and veinlets. In places, it has a faint brownish hue as though slightly ferruginous. The ore is reported to carry high percentages of manganese dioxide but also about 0.25 per cent. copper, which renders it unsuitable for chemical purposes.

The workings consist of three open pits and five or six small prospects extending about ⅛ mi. (0.2 km.) northwest-southeast. A road built by the lessees at a reported cost of about $8000 extends from Guama to the mine, a considerable part of the distance being cut in solid limestone, and was in fair condition, but the public road to Bayamo was in bad condition for 3 mi. beyond Guama. The mine was idle in April, 1918, because of the high cost of transportation to Bayamo. Some few hundred tons of ore had been shipped from here in 1917 and about 300 tons of selected ore were piled at the weighing shed in sacks, which were rotting to pieces.

*Llego and Guisa Prospects.*—Deposits of manganese ore occur in places on the west side of a prominent limestone ridge about 5 mi. (8 km.) east of Guisa and 2½ mi. east of Coralillo. These deposits are at barometric altitudes of 1000 to 1200 ft. (304 to 364 m.) and have been opened at three or four places within a distance of 1 mi. The feature of principal interest in connection with these deposits is that the manganese ore simulates thin beds, conformable with the thick strata of limestone above and below them.

At the Llego the strata dip gently to the southeast, and there has been exposed the following section:

| | Feet |
|---|---|
| Limestone, gray, massive, crystalline, containing scales and small lumps of manganese oxide in lower part | 50+ |
| Manganese ore, 1 to 3½ ft. thick, average | 1½ |
| Limestone, with small, sparsely scattered lumps of manganese oxide | 6½ |
| Manganese ore | 2+ |
| Limestone | 4− |
| Manganese ore | 1± |
| Limestone, exposed | 75+ |

The manganese-ore beds are connected in places by films of manganese oxide deposited along vertical joint planes. Calcite is present in cavities in the ore and along the contact with the limestone. The ore is reported to carry about 45 per cent manganese and to be high in phosphorus in places. The ore is probably a replacement of calcareous, tuffaceous lenses in the limestone. The opening showing the ore extends about 150 ft. (45 m.) along the face of the bluff, but there is no evidence as to its extent down the dip.

At the Guisa opening conditions are similar, except that the upper bed contains more calcium carbonate and less manganese oxide than the corresponding one at the Llego. There are better opportunities for observing these deposits at this opening because they are exposed in the direction of the dip in lateral ravines, the total distance of exposure being about 600 ft. (182 m.). These deposits are about 7½ mi. (12 km.) by trail to the Cuba Railroad station of Santa Rita, and a shipment of about 60 tons of ore from Llego and Guisa was reported to have been made by mule back in the spring of 1918.

*Charco Redondo Mine.*—The Charco Redondo mine, operated by N. O. Pierce, of Minneapolis, Minn., and Wm. Carleton, of Sabanaso, Cuba, is situated about 8 mi. (12.8 km.) southeast of the Santa Rita station of the Cuba Railroad. There is a relief of more than 500 ft. (152 m.) between the lowest and the highest manganese deposits observed here and the rocks exposed consist of a series of gently dipping beds of argillaceous sandstone and limestone and massive crystalline limestone. The deposits are of three types, each occurring at a particular horizon. The lowest is a bedded or tabular mass in the limestone bluff just above the level of Cautillo River, at an altitude of about 700 ft. (213 m.), see Fig. 23; the next is a breccia deposit, resembling a talus, near the foot of a higher limestone escarpment at an altitude of about 950 ft. (289 m.); and the highest is a bedded deposit on the upland just back of that escarpment, at an altitude of about 1250 ft. (381 m.); see Fig. 24.

The tabular mass of ore appears to be interbedded with massive gray crystalline limestone about 100 ft. (30 m.) below the top of the bluff. It is reported to have been traced for about 900 ft. (274 m.) along the river, which flows parallel to the strike of the beds. A series of falls have

developed on the limestone ledges at this place, so that the manganese bed is about 25 ft. higher than the river level below the falls and only a few feet higher above the falls. The manganese bed is 1 ft. 4 in. to 2 ft. 6 in. (0.4 to 0.7 m.) thick, and may average 1 ft. 6 in. (0.45 m.). A prospect drift 6 ft. (1.8 m.) high has been driven in on the dip, which is about 8° toward the north, following a joint plane in the limestone for a distance of 27 ft. (8 m.) without cutting through the deposit. The limestone roof over the bed is comparatively flat, but the floor is less even. The following section of the bed is representative of the portion cut by the drift:

| | | Feet | Inches |
|---|---|---|---|
| Massive, gray limestone.... | | 50+ | |
| Manganese ore bed.. | Manganese oxides mixed with small fragments of tuff, grains of pink clay, and volcanic ash; contains shark's teeth..... | | 2 |
| | Manganese oxides, in places crystalline, contains tuff sand and calcite in places ..... | 2 | 6 |
| | Ocherous, sandy material ..... | | 2 to 4 |
| Massive, gray limestone .... | | 25+ | |

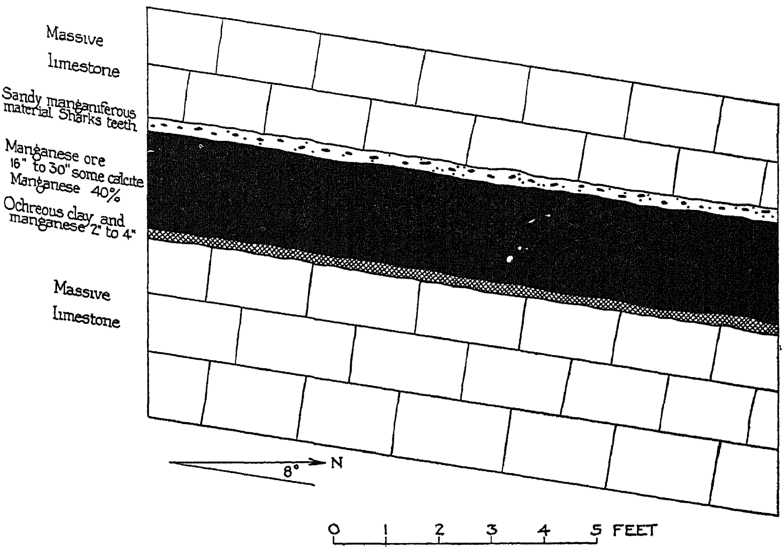


Fig. 23.—Section of manganese-ore bed in limestone bluff of Cautillo River, at Charco Redondo mine.

The material is reported to average 40 per cent. or more in manganese. To all outward appearances, this deposit is of sedimentary origin and it is conceivable that the original may have been tuff laid down as volcanic sediments during the formation of the limestone series. Later most of the tuff bed was replaced by manganese minerals. The shark's teeth are reported to be of Eocene age, as are the limestones generally in this region. An alternative explanation of the formation of this bed is that

an extensive horizontal crevice that had been formed by solution of the limestone was subsequently cut open by the river and filled with sediments derived from the breaking down of tuff deposits at a higher level Later replacement by manganese minerals of portions of the deposit was effected by circulating waters.

The breccia deposit that occurs at two or three places at the base of the next higher escarpment contains manganese oxides mixed with fragments of tuff and coarse grains of buff to pink clay-like material, the mass being soft and poorly consolidated. The deposit is 6 to 10 ft. (1.8 to 3 m.) thick and is reported to carry 22 to 30 per cent. of manganese. The base of the deposit was not exposed, but the overburden consists of clay, sand, and boulders of oolitic limestone. At the top of the mixed ore, there is usually a crust slightly richer in manganese oxide. In this deposit the writer found an abundance of shark's teeth. A few specimens were submitted to J. W. Gidley of the U. S. National Museum, who kindly examined them and made the following report:

"The fossil shark's teeth from Charco Redondo mine represent three species determinable as follows: *Lamna cf. L. elegans*, these are not typical *L. elegans* but seem nearer this species than any other described. *Isurus hastalis*, or very near this species. *Carcharodon auricularis* (?), the large serrate tooth is like this species only more slender than is usual. These teeth seem to be of Cretaceous or Tertiary, probably Eocene age."

This material may have been formed by the breaking down of a higher bed of tuff and have been recemented by manganese ore, but it may also represent a remnant of a bed deposited on an eroded surface of sedimentary rocks.

Prospecting had been done at one place by open cuts and at another a face about 118 ft. (35 m.) long had been cut and two drifts 54 ft. (16 m.) apart and 5½ ft. (1.67 m.) high had been driven perpendicular to the face, to distances of 52 ft. (15 m.) and 76 ft. (23 m.) respectively, and from the longer of these nearly 100 ft. (30 m.) of additional drifts had been driven—all in mixed ore. This work disclosed a good tonnage of lean material, all of which will require concentration in order to render it of commercial value. The manganese minerals are soft and it will probably be difficult to effect a large saving in cleaning the ore, and whatever is saved will probably be so fine grained as to have to be briquetted for metallurgical use.

The deposit at the highest altitude is also of an unusual type. Here, on the upland back of the escarpment, a bedded deposit of manganese oxide, dipping at a low angle toward the north, has been uncovered by stripping over an area about 1200 ft. (365 m.) along the strike by 40 to 50 ft. (12 to 15 m.) down the dip; see Figs. 24 and 25. The ore bed ranges from 3 in. (7 cm.) to 1 ft. 8 in. (0.5 m.) thick, and averages about 1 ft. to 3 in. (0.38 m.) thick. The ore bed is hard and compact and much

of the manganese oxide is in nodular incrustations of crystalline minerals, probably mixtures of pyrolusite and manganite. The content of manganese is reported to be from 42 to 44 per cent. The bed is overlain by 2 to 8 ft. (0.6 to 2.4 m.) of thin-bedded chalky argillaceous limestone, containing chert in places, and the underlying limestone is of a harder variety. The bottom of the deposit is less regular than the top and shows many small round pits into which the manganese oxide conforms. The top of the ore bed shows mammillary structure, in places incrusted by a greenish-brown deposit, that appears to be an iron silicate containing potassium and is probably glauconite.

The resemblance in attitude and structure that this deposit bears to certain bog-iron-ore deposits observed by the writer in the southern United States is rather close and suggests that the manganese minerals were concentrated in a bog—if that were possible—together with calcareous sediments, and later the whole deposit became covered by the chalky limestone strata. The overlying limestone is relatively thin and differs considerably in texture from the underlying beds and may be a comparatively recent deposit.

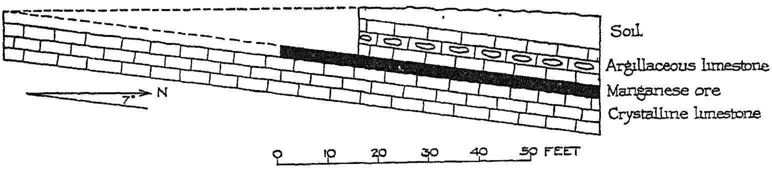


FIG 24.—SECTION OF FOSSIL BOG DEPOSIT OF MANGANESE ORE AT CHARCO REDONDO MINE.

This bed has been mined by stripping 2 to 8 ft. (0.6 to 2.4 m.) of overlying rock and then taking up the ore in lumps. The bed extends beyond the present stripping, which has reached a practicable limit. It was reported that about 1200 tons of ore were shipped in 1917 and about 530 tons of lump ore were awaiting shipment in April, 1918. Under normal conditions but little more ore can be regarded as probably available. The breccia deposit of milling ore is of doubtful quality. The bedded deposit in the river bluff may contain a few thousand tons of ore, but until prospected further there can be no certainty of this. Profitable mining of such a thin bed would be difficult on account of the large amount of dead work necessary to be done in hard limestone in order to provide room for mining.

*Adriana Mine.*—The Adriana mine, operated by the Cuban Industrial Ore Co., of Philadelphia, Pa., and Baire, Cuba, is situated about 9 mi. (14 km.) south of Baire, in the outlying hills north of the foothills of the Sierra Maestra, and about 4 mi. S. 65° E. of the Charco Redondo mine. The mine is near the base of a prominent ridge of limestone 200 ft. (60 m.) or more in height. The altitude of the base of the ridge is about

1000 ft. The dip of the limestone is 20° to 50° about N. 75° W. and averages about 40°.

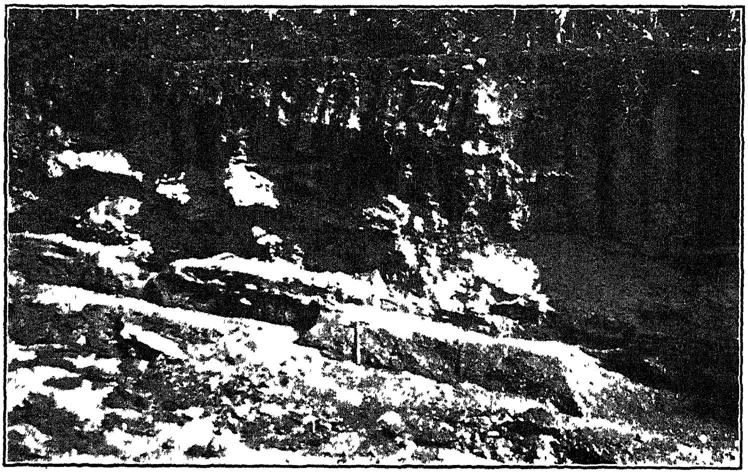

FIG. 25.—EIGHT MILES SOUTHEAST OF SANTA RITA, CUBA, CHARCO REDONDO CLAIM. OPEN CUT ON UPPER BED OF MANGANESE ORE. ORE BED SHOWN BY HAMMER.

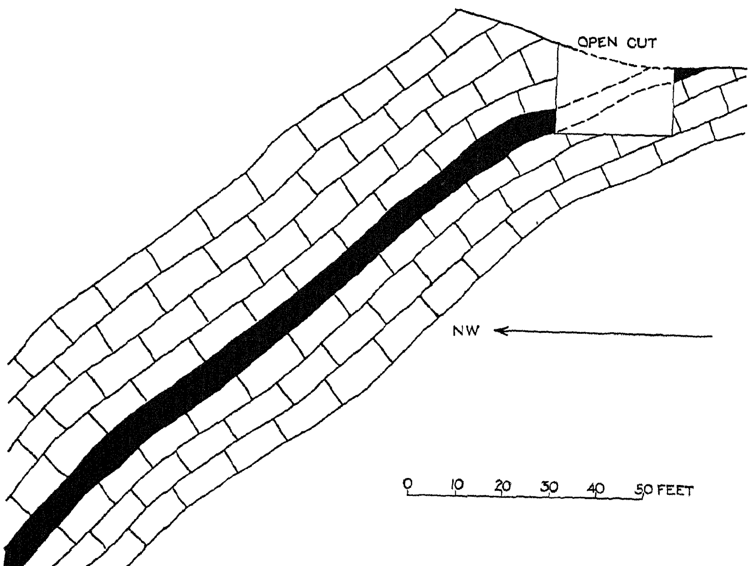


FIG. 26.—SECTION SHOWING RELATIONS OF TUFF, MOSTLY REPLACED BY MANGANESE OXIDE, TO ENCLOSING THICK-BEDDED LIMESTONE IN ADRIANA MINE.

The principal manganese-ore deposit is in the form of a tabular mass, or bed, between strata of limestone, see Fig. 26, and outcrops near the base

of the ridge. Its thickness ranges from 2 ft. 6 in. to 6 ft. (0.75 to 1.8 m.), and averages about 4 ft. (1.2 m.). There is an outcrop of a similar deposit about 20 ft. higher stratigraphically, but as it has not been traced far it may be the same bed dislocated by faulting. The ore in the main deposit is of good quality and was reported to be averaging about 45 per cent. of manganese where mined. Some pockets of rich powdery pyrolusite were encountered, but most of the material was mined as lump ore. The ore bed was perhaps originally a bed of tuff interbedded with the limestone Replacement of the tuff by manganese oxide seems to have been unusually complete in this instance.

The workings consisted of an open pit about 30 ft. (9 m.) deep from which a series of inclines extended down the dip of the ore deposit to a total depth of about 90 ft. Drifts turned off on the strike of the ore showed the deposit to pinch out at the right and left, with a total length of the workable ore shoot of about 50 ft. There was still good ore in the bottom and a new straight incline having a slope of about 45° was being constructed Ore was being hauled in ox-carts 8 or 9 mi. (12 to 14 km.) to Baire, over a road that is said to be passable even in the wet season.

This is a type of deposit on which no reasonable estimates of ore tonnage can be made in the absence of prospecting. It is bedded material of fairly high grade and might be expected to continue to considerable depth, but it must be considered that the deposits of tuff interbedded with limestone are lenticular in shape and are probably limited to a few hundred feet in length and to a smaller width. Moreover, it is uncertain as to whether manganese oxide may have replaced the whole lens or not. Prospecting the bed by drilling down the dip in the mine workings would seem the most practicable way of ascertaining the ore ahead of the workings since the bed cannot be drilled from above ground.

*San Antonio Mine.*—The San Antonio mine, operated by Prudencio Bravo, of Santiago de Cuba, is situated about 8 mi. (12 km.) S. 15° W. of Baire and about 2½ mi in a direct line a little south of east from the Adriana mine. The altitude of the deposits is about 1000 ft. and the country is hilly and broken, the main topographic feature being an eastward continuation of the limestone ridge noted at the Adriana. Manganese ore occurs as bedded material between strata of massive oolitic limestone, and as lumps in residual clay overlying a deeply weathered surface of the same limestone. The fragments of manganese in the clay may have been derived by the breaking down of a lens of ore formerly enclosed in the limestone. The grade of the ore appears to be good and it was reported to carry 40 to 44 per cent. of manganese. The bedded material is about 2 ft. (0.6 m.) thick in the face of a large open cut about 200 ft. (60 m.) long, and is 6 in. to 1 ft. (0.15 to 0.3 m.) thick in other places.

The largest active workings at the time of visit were in the residual

clay. The 200-ft. cut had been worked back a distance of about 15 ft (4.5 m.) to where the stripping of overlying limestone became 10 ft. thick and could not profitably be stripped farther. At another small pit lower down and ½ mi. (0.8 km.) toward the southwest, a bed of ore 8 in. to 1 ft. (0.2 to 0.3 m.) thick has been mined by stripping off a few feet of limestone. A small tonnage of ore was regarded as still available here, mostly in the residual clay deposits. A few tons per day of ore was being moved to Baire by ox-carts and pack mules. The road from the Adriana is followed part way, but the road near the mine is boggy in spots and is over soil and clay for 2 or 3 mi. (3 to 4 km.) and probably would not be passable in the rainy season.

*Los Negros Area.*—The San Antonio mine is on the border of what is known as Los Negros manganese area, which extends for about 15 mi. (24 km.) farther southeast. This area was not visited by the Government party as no production was being made from there in the spring of 1918 and the deposits are so remote from the railroad and so poorly provided with roads that the prospect of any production of ore during the period of the war seemed remote. According to several capable and reliable mining engineers, the occurrence of manganese ore is similar in type to that at the Llego, Guisa, Charco Redondo, Adriana, and San Antonio properties; viz., small high-grade bodies associated with limestone. Mr. John B. Stewart, of Santiago de Cuba, a member of the Institute, who has recently visited the Los Negros area, believes[8] that the district was drained for a long period through an extensive cave system developed in the limestone and that the manganese deposits later filled portions of these caves and solution channels. He finds the Unica mine, which is about 26 mi. (41 km.) south of Baire to be a cave more than 80 ft. (24 m.) in depth and more than 20 ft. in width. He noted several similar caves in the area and states that the hardest and purest manganese ore seemed to be closely related to a pink dolomite that underlies a purer limestone characterized by tooth-shaped weathered surfaces and popularly termed the "diento de perro" or "dog tooth" limestone. The crevices and caves may have been filled first with debris of tuff which was replaced by manganese oxide. The nature of the cavities and of such contained material as has been studied by the writer does not indicate that the limestone formerly occupying the cavities had been replaced by the manganese minerals.

The Unica and other mines are reported to have made small shipments of high-grade chemical ore by mule back in the winter of 1917. There may be an important tonnage of ore in the Los Negros area distributed among many small deposits, but its development is very doubtful unless good roads are built by the Cuban Government.

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[8] Personal communication to the writer.

*Production and Ore Reserves.*—The production of manganese ore from the district south of Bayamo has amounted to a few thousand tons altogether, the mining operations being on a small scale. In the spring of 1918, only 50 to 60 tons a day were being produced from all the mines, but much of this was high-grade ore for chemical purposes The development of the deposits is handicapped by their remoteness from the railroad and by the lack of good wagon roads. Government aid for the completion of a road from the ore-bearing area south of Bueycito to Julia, a station on the Cuba Railroad between Bayamo and Manzanillo, is very much needed, and if several bridges had been completed in the spring of 1918 a much larger output of ore could have been produced. Strenuous efforts were being made during the war period to get ore to market, notwithstanding the handicaps. Ore had to be carried 10 to 25 mi. (16 to 40 km.) in sacks on the backs of mules and in ox-carts, and in the dry season a motor truck carried ore 16 mi. to Bayamo, fording Bayamo River many times and standing hub deep in the water of Guama River to load the ore from a wharf.

The manganese-ore deposits in this district are of great variety and comprise certain unusual types of ore. The nature of the deposits, their general richness, and the possibility of the occurrence of undiscovered deposits renders this district of considerable interest to the geologist and to the producer and consumer of manganese. Little mining has been done here and since most of the deposits, like the Cuban manganese-ore deposits generally, are richer near the surface than deeper, it is still possible to produce high-grade ore here by selective mining. The "cream" of the ore is now being skimmed off, but deposits of milling ore are also available. There is plenty of water at most of these deposits but it is doubtful whether it will be profitable to attempt their mechanical concentration until good roads are built and the price of ore again reaches a high figure.

The reserve of manganese ore in the district south of Bayamo is conservatively estimated at between 50,000 tons and 75,000 tons, most of which is in the western part; this estimate does not include the Los Negros district. It would not be surprising, however, if many times the estimated tonnage should eventually be developed.

### *Prospects in the Camaroncito District*

By the Camaroncito District is meant the area along the south coast of Oriente Province between Portillo and Torquino, in which deposits of manganese ore have been found. There are several places where such deposits occur, but only two representative ones need be noted here, the Camaroncito properties and the Rio Magdalena prospect.

*Camaroncito Group.*—The Camaroncito group of manganese-ore prospects, owned by the Compania Oriental de Minas of Havana and Santi-

ago, is situated on the Caribbean coast about 6 mi. (9 km.) east of Portillo and 80 mi. west of the harbor of Santiago de Cuba. The manganese deposits that have been opened are all in the narrow strip of foothill country between the Sierra Maestra and the coast and are at low altitudes, generally between 30 ft. and 250 ft. (9 m. and 76 m.) above sea level. The rocks along this belt consist of partly metamorphosed shale and siliceous limestone. The shale is sparingly glauconitic in places. Andesite has been reported as having been encountered in one of the tunnels. The manganese deposits generally are replacements of siliceous limestone and glauconitic shale. (See description of thin section, page 59.)

Considerable development work has been done on the Camaroncito properties, particularly at the Pittsburgh, the Concha, and the San Juan openings. The Pittsburgh, or Don Tomas, incline is about ½ mi. (0.8 km.) from the wharf at an altitude of about 35 ft. (10 m.). A bed of manganiferous siliceous limestone outcrops here at the foot of a low ridge and has been followed for 45 ft. (13 m.) down a 30° dip by means of a timbered incline in a direction N. 60° W. Right and left drifts have been turned off about 15 ft. (4.5 m.) in length about 20 ft. below the entrance. At the entrance the manganiferous bed is about 1 ft. (0.3 m.) thick but it increases to 5 or 6 ft. in the workings and then thins out. The deposit is evidently lenticular in shape. The incline and both drifts passed completely through the ore and work was discontinued.

The Concha prospects are situated about ½ mi. (0.8 km.) north of the Don Tomas incline, on a hillside along the face of which a manganese-bearing zone 6 to 12 ft. (1.8 to 3.6 m.) thick ascends toward the northeast. Three open cuts have been made here at altitudes of 160 ft. (48 m.), 200 ft. (60 m.), and 240 ft. (73 m.). An incline sloping about 23° to the northwest was driven about 45 ft. (13 m.) from the bottom of the lowest cut. The rock is reddish, slightly glauconitic material weathering shaly. Rich manganese oxide, probably pyrolusite, occurs in streaks and pockets, but most of the deposit is lean and appears ferruginous and siliceous. The open cuts and incline pass entirely through the deposit.

The claim known as the San Juan is about ¼ mi. (0.4 km.) southwest of the Pittsburgh. Much manganiferous float is encountered in the form of boulders on the north slope of a hillside from 50 to 240 ft. (15 to 73 m.) in altitude, and four or five prospect cuts and a tunnel have been made on an ore-bearing zone 2 to 8 ft. in width associated with reddish glauconitic shale overlain by light-colored quartzite. Some small rich pockets and thin seams of needle-like or fibrous crystalline ore were noted. All the cuts passed through the ore.

A series of commercial analyses of samples collected by an American mining engineer and checked by the company chemist show that the ores are of very much lower grade than might be expected from visual examination. In each instance a sample of several hundred pounds of

ore was taken from a pile that had presumably been partly sorted and hand-picked and the sample was further hand-sorted into ore and waste rock. The percentages of ore and waste rock were recorded and duplicate samples of each were then taken. The manganese carried by the ore ranged from 17.4 to 32 per cent., the iron, from 6.6 to 12.6 per cent.; and the silica from 19.7 to 31.8 per cent. The waste rock sorted out by hand carried 11.4 to 22 6 per cent. manganese, 8 to 10.25 per cent iron, and 29 to 41 per cent silica

Good wagon roads and a commodious camp, supplied with water piped from a mountain stream, had been built here, but all work on these prospects was suspended in the spring of 1918, due no doubt to the fact that high-grade ore sufficient for commercial exploitation could not be found at any place. In many places near the prospects, as well as at the wharf, are dumps containing ore mixed with siliceous material, which by cobbing could probably be made to yield a small proportion of ore of shipping grade. There is at Camaroncito a small indentation in the coast, with a channel deep enough for boats of moderate draft to land at a pier in fair weather

*Rio Magdalena Prospect.*—Rio Magdalena flows into the Caribbean Sea about 16 mi. (25 km ) east of Portillo. About ½ mi. (0.8 km.) east of this river and ¾ mi. from the shore some manganese ore has been disclosed by prospecting. The prospects are on a hillside at an altitude of about 250 ft. (76 m ), and are in siliceous limestone associated with ferruginous glauconitic shale as at Camaroncito. The principal prospect is a northwest-southeast trench 70 ft (21 m.) long and 3 to 7 ft. (0.9 to 2 m.) deep. A manganiferous layer 3 to 4 ft. (0.9 to 1.2 m.) thick is disclosed, dipping about 50° toward the southeast. Float ore around the hill indicates that the manganiferous layer extends beyond the prospect. The presence of the mineral, braunite, was determined in a specimen from this prospect. The ore is good in spots but the showings do not appear to warrant further prospecting.

## Deposits in Santa Clara Province

The only deposits of manganese ore in Santa Clara Province that have come to the attention of the writer are on the south coast west of the town of Trinidad.

*Trinidad Group.*—A group of claims owned by the Trinidad Manganese Co., of Havana, is situated on Rio Guanayara about 1½ mi. (2.4 km.) from its mouth, or about 35 mi. (56 km.) east of Cienfuegos, and 6 to 8 mi. west of Trinidad. These claims, or denouncements, comprise six tracts, known as the Mercedes, Serafina, Adela, Hipolita, Purita, and San Juan, with a total area of 347 ha., or about 860 acres. These properties were examined in September, 1917, by G. Sherburne Rogers,

of the U. S. Geological Survey; the following notes are adapted from his report to the Survey.

From near Cienfuegos to far beyond Trinidad, the Caribbean coast of Cuba is bordered by the Sierra de Trinidad, a rugged chain of mountains that rise to a height of about 3000 ft. (914 m.) above sea level within 5 mi. (8 km.) of the coast. The properties of the Trinidad Manganese Co. lie in the southern foothills of this range. The southernmost claim, the San Juan, lies on a relatively gently sloping surface; but the surface of the area covered by the other claims is rugged, the relief on the Purita and Hipolita claims amounting to 500 ft. (152 m.) or more. The whole area is densely timbered, chiefly by the thorny arroma, which forms an impenetrable thicket. Guanayara River is a small mountain stream flowing almost due south. It is said to flow throughout the year and to be subject to floods during the rainy season. In the northern part of the property, the river has a rapid fall that culminates in a series of cascades 60 ft. (18 m.) high, below which the gradient is low and the stream is said to be navigable for small boats. If the flow is sufficient during the dry season this stream should be a valuable source of power.

The rocks exposed in this locality in a section beginning at the coast and extending northward to beyond the manganese deposits are: coral limestone, probably of Pleistocene or Pliocene age, called locally "diento de perro," or dog-tooth limestone, on account of its rough surface covered with solution cavities surrounded by sharp pinnacles; an older, cavernous limestone, probably of Oligocene age; and a thick series of crystalline limestone, micaceous marble, and calcareous mica schist, probably pre-Cretaceous in age. The first outcrops in a strip about 500 ft. (152 m.) wide along the coast. The second, which underlies it, outcrops in a belt about 3000 ft. wide and is probably between 500 ft. and 1000 ft thick; it shows southward dips in places. The third unconformably underlies the second, dips 20° to 80° toward the south, and has been extensively deformed. This schist has been faulted to some extent and considerably shattered with development of quartz and calcite veins. Many crushed and brecciated zones occur in the schist and contain quartz, calcite, siderite, and manganese and iron oxides. The crystalline limestone in places contains considerable ankerite and siderite and some of the schistose beds contain enough rhodonite in places to give them a pink color which weathers black.

The manganese ore appears to occur in irregular bodies in faulted and crushed zones in the schist. At one point, an orebody reaches a width of 18 ft. (5 m.) and has replaced a limestone bed besides having impregnated the inclosing crushed schist, but the average width at this place is not more than 8 ft. (2.4 m.) and in other places is less than 5 ft. (1 5 m.). The depth to which ore continues is not shown, but it is not probable that it continues below ground-water level, which probably lies at a depth

of 60 to 100 ft. (18 to 30 m.) in this locality. The principal ore mineral is psilomelane, but there is considerable pyrolusite and some well crystallized manganite present. The associated minerals are quartz, calcite, siderite, and limonite, the latter mineral occurring to so large an extent in some deposits as to render the ore valueless as a source of manganese. The bulk of the ore is found in a detrital form in a soft, brown, earthy gangue derived by alteration of the schist. Five analyses, probably of selected specimens or of the ore stacked for shipment, furnished by the operators show a range in manganese content of 43 to 52 per cent.; iron, 1.41 to 7 35 per cent.; silica, 2.63 to 13 per cent.; phosphorus, 0.03 to 0.45 per cent. One sample shows 0.036 per cent. alumina and 13.80 per cent. of barium oxide.

Rogers mentions nine openings but from only one of these, the main opening on the Mercedes claim, had any shipments of ore been made. This opening consists of a pit, averaging 55 ft (16 m ) square and 5 to 20 ft. (1 5 to 6 m.) deep, from which a drift has been driven 55 ft. into the hill to the southeast. In the bottom of the pit a 7-ft. (2-m.) shaft, 10 ft. (3 m.) deep, and an irregular hole, about 15 ft deep, have been dug. Here the ore occurs in irregular bodies; in general, it is parallel to the strike, in a rotten schist zone altered to a brown, earthy, ocherous material. In places the ore has replaced the brown limestone wall rock and parts of other associated limestone beds. From this working about 400 tons of hand-picked ore had been taken, most of which came from the hole. In the best part of the deposit, the manganese minerals appear to constitute 25 to 30 per cent. of the material. It is estimated that about 24,000 cu. ft. (672 cu m.) of vein material has been excavated in order to obtain 400 tons of commercial ore. Rogers considers that the best deposit of ore on the property has been uncovered in this opening, but that it is not possible to make any estimate of tonnage remaining, on account of the pockety nature of the ore and the absence of neighboring workings. Exploration along the strike in both directions, but particularly in a line about N. 50° E. from the Mercedes main opening, may indicate other similar deposits. In the other openings, for the most part, the showing of ore was too meager to warrant further developments.

A wagon road had been constructed at a reported cost of $8000 from the main opening to the coast at the mouth of the river, a distance of about 1¾ mi. (2. 8 km.). About 350 tons of ore had been hauled to this point, 200 tons of which had been shipped The indentation of the coast at the mouth of Rio Guanayara is too shallow for any but the smallest vessels, and the nearest port is the Ensanada de Casilda, the port of Trinidad, 10 mi. to the southwest. It is rumored that some ore was shipped from this property to the United States in the spring of 1918.

*Dario Claim.*—The Dario claim was given a very brief inspection by the writer on Apr. 28, 1918. It is about 1½ mi. (2.4 km.) from the Caribbean coast near a small stream known as Rio Cabogones, and is about 11 mi. west of Trinidad, or 5 mi. west of the deposits in Rio Guanayara just noted. The property, which covers 200 ha., was owned by Sr. Manuel Gomez-Valle of Cienfuegos. The topographic and geologic relations of the manganese minerals at this place are practically the same as those of the Trinidad deposits, described by Rogers. The deposits noted lie at altitudes of 400 to 450 ft. (121 to 137 m.) above sea level where the relief in the foot hills begins to become moderately rugged, and the rock containing them is schistose crystalline limestone and calcareous mica schist. The strike of the beds is northeast-southwest and the dip is steep toward the southeast. The rock is somewhat shattered in places and contains considerable white quartz in veins and also segregations of chert. The lowest outcrop consisted of ledges and boulders of manganiferous limonite. Although dark in color, and nearly black in places, this material appears to carry more iron than manganese. Manganese oxides, however, have been disclosed practically at the surface in five or six shallow pits about 50 ft. (15 m.) higher on the slope and on this slope, which is partly clay covered, float, consisting of grains and lumps of manganese oxide up to 4 in. (10 cm.) in diameter was noted. This material appears to be of high grade when the lumps are broken, but on the weathered surface it shows a dark reddish-brown color, just as does much of the rich residual manganese ore found in clay in the Shenandoah valley of Virginia and the Tennessee River valley of East Tennessee. The deposits noted in the prospect pits were of two varieties; one apparently directly replaces parts of beds of the schistose limestone and the other partly fills and cements brecciated zones. The manganese minerals are pyrolusite, psilomelane, and manganite. The replacement bodies follow the beds but they are only 1 or 2 ft. thick and extend only a few feet or yards along the strike. The brecciated material is mixed with quartz and schist, so that some form of concentration would be necessary if it were mined. The residual fragments in the clay would require log washing.

The results of the examination were inconclusive, because prospecting has relatively only scratched the surface. The prospects, to be sure, are where manganese oxides actually outcrop and they do not show any large deposits but they are not extensive enough to give very definite information. The surface clay between places where manganese minerals are known to occur has not been prospected. At the time of visit, the deposits were reached from the coast only by a trail. At the mouth of Rio Cabogones, there is a small shallow indentation in the coast, but it is not suited to the landing of cargo vessels so that ore would have to be

lightered to boats anchored off shore or else barged to the Ensanada de Casilda, about 15 mi. to the southeast.

## Deposits in Pinar del Rio Province

Manganese minerals have been found in two localities in Pinar del Rio Province, one near Vinales, north of the city of Pinar del Rio, the other near Mendoza, manganiferous iron ore occurs near Arroyas de Mantua None of these deposits promise to be of importance as sources of manganese but those near Vinales and Mendoza will be noted for the sake of completing the record.

*Vinales* —The village of Vinales is on the Pinar del Rio—Esperanza highway about 15 mi. (24 km ) north of Pinar del Rio. Vinales lies in a picturesque valley bordered by rugged ridges of Jurassic (?) limestone, the Sierra Organos The Organos here consist of several parallel ridges of massive hard limestone reaching heights of 800 to 1000 ft. (243 to 304 m ), separated by flat valleys carved in softer schist, shale, and sandstone. The parallel limestone ridges appear to have been formed by repetition of the strata through faulting. In these Sierras to the west and east of Vinales deposits of manganese minerals have been found. One of the deposits is situated about 3½ mi. (5.6 km.) a little north of west of Vinales but is reached by a more circuitous route, a trail that leaves the highway about 1½ mi north of Vinales, crosses an intermediate valley or "vega," ascends one of the Sierras, then follows along its north flank for more than 1 mi. and finally descends into another flat-bottomed, picturesque "vega" in which nestles the hamlet of Ancon devoted to the activities incidental to producing tobacco. Here, on the north side of the "vega," 75 to 150 ft. (22 to 45 m.) above the valley floor, or at altitudes from 575 to 650 ft. (175 to 198 m.) in a cliff or "mogote," of massive limestone that dips 30° to 40° N. 20° W. is a band of hard, black to brownish, manganiferous material conformable with the bedding of the limestone. This band is 6 to 10 in. (15 to 25 m.) thick, but a great many measurements showed a general thickness of 9 in. The material is overlain by a few inches of pinkish limestone which, in turn, is overlain by gray limestone containing very thin streaks of chocolate-colored carbonaceous shale, with particles of limonite in places, and is underlain by gray limestone. There is a very sharp transition in color from the brown manganiferous material to the limestone above and below it, but otherwise the rocks seem to be part of the same bed. The brown rock can be traced 300 to 400 ft. (91 tó 121 m.) along the cliff. It has been stripped for a few feet on the dip in a few places, evidently for the purpose of sampling, but this is the only development work that has been accomplished.

This material is cut by clean sharp joint planes and breaks into more

or less rectangular fragments. It gives a dark brown streak and has a specific gravity of 3.1628 A partial chemical analysis by George Steiger gave the following results Insoluble in HCl, 25.09; $(AlFe)_2O_3$, 16.91, CaO, 16.83; MgO, 3.29, Mn, 19 46.

This deposit is of considerable scientific interest, but it is of little or no commercial value, because it is not thick enough to be mined and is relatively so inaccessible. Whatever production might be obtained would have to be carried on mule back 3 mi. (4 km ) or more to the highway, where auto truck haulage could be used, or a cart road 1 mi. or more long would have to be constructed northward through the Sierras to connect with an ox-cart road 10 mi. in length, which leads to a small cove in the coast west of Esperanza.

The other reported deposit of manganese ore in this vicinity was not visited by the writer. It is said to be about 2½ mi. northeast of Vinales, but nothing definite could be ascertained concerning its character.

*Mendoza.*—Deposits of manganese ore are known to occur at two localities near the town of Mendoza. One of these localities is known as the Turrialba property and lies about 1¼ mi. (2 km.) northeast of the railway station at Mendoza; the other is known as the Maria Cristina property and lies about 4 mi. south-southeast of Mendoza, near the settlement of Catalina. The Turrialba property was examined by Dr. Max Roesler, in December, 1917, and the following notes are based on his report.

The Turrialba property consists of 625 ha. The surface is mostly gently rolling, but at the northern end rises the Cerro Pasco Real, an outlying limestone hill, or "mogote," of the Cerro de Guane. The underlying rock is limestone, which incloses a horizon of sandstone and sandy shale. The beds dip very steeply and have been considerably dislocated by faults. The contact between the limestone and the sandy horizon is the locus of the manganese mineralization. The manganese ore occurs in veins or small pockets in the weathered zone between the limestone and the sandy beds. On the surface this contact is marked by float ore, which consists of small pebbles of manganese oxide with a core of calcite or limonite, but pure manganese oxide pebbles are also found, especially on the northern part of the property. The vein, or pocket, ore appears to be in the earthy form and as crystalline pyrolusite.

The property has been prospected to some extent; the most promising results have been obtained on the Buena Vista, near the foot of the Cerro Pasco de Real. Here a shaft was sunk 15 or 20 ft. (4 or 6 m.) in depth, from which was obtained the bulk of a shipment of about 35,000 lb. (15,855 kg.) of ore that was sent to Chicago. Part of the material shipped was obtained by washing in a "long tom" dirt containing manganiferous float, from the Buena Vista and from Arroyo Gordo working, also along a limestone-sandstone contact zone 300 to 400 yd. (274 to

365 m.) southeast from the shaft. A sample from the ore pile showed, on analysis, 45 15 per cent manganese, 2 30 per cent. iron, 10 65 per cent. silica, 0 017 per cent sulfur, and 0.063 per cent. phosphorus; but a copy of the analysis on the invoice showed 41 20 per cent. manganese and 12 93 per cent. silica. There was still ore in sight in this shaft, in the form of veinlets or bunches, in the fractures of the weathered sediments at the time of Dr. Roesler's visit. Analyses of samples, some of them concentrates, from other prospects showed manganese ranging from 36 to 43.5 per cent. and silica from 12 5 to 18.7 per cent. Some of the ore that has been found is manganiferous limonite, containing about 45 per cent. iron, 6 per cent manganese, 11 per cent silica, 0.035 sulfur, and a trace of phosphorus.

The Turrialba prospects are excellently situated for transportation, since the Western Railroad of Havana crosses the property. There is an abundant supply of water in Rio Cuyaguateje, which borders part of the west side of the property.

The Maria Cristina prospects were examined by the writer on Apr. 21, 1918. This property is located in rolling country having an altitude, according to an aneroid, of 125 to 175 ft. (38 to 53 m ). The locality is underlain by limestone interbedded with sandy shale, the beds striking N. 70° E. and dipping almost vertically. The rocks show considerable crystalline quartz and calcite in fractures. Manganese oxides have been deposited chiefly in the shaly beds along their contact with the limestone. Along these zones the limestone is argillaceous and the shale is sandy and calcareous. Fine manganese oxide gravel occurs as float on the surface along these zones or on slopes below them. There have been opened a shaft about 50 ft. (15 m.) deep, a trench 30 ft. long on the strike of the beds cut by the shaft and about 50 ft. northeast of the latter, a pit 25 ft. deep on the same strike about 1500 ft. southwest of the shaft and a pit 20 ft deep about 500 ft. north of the shaft on a parallel zone of shale. The shaft was being retimbered at the time of visit and contained water at the bottom so that no ore could be seen, but it was reported that the vein of ore has been found to be from 1 to 3 ft. thick. The trench showed only streaks and seams of manganese oxide in the beds, which comprised a gradation from hard argillaceous limestone to calcareous shale. The pit to the southwest showed an 8-in. (20-cm.) vertical bed of carbonaceous limestone with manganese oxide in seams and joint cracks. The oxide is rich powdery pyrolusite, but the quantity visible is very small. The pit to the north discloses iron and manganese-stained shale, but there appears to be no depth to the replacement.

About 15 tons of ore, mostly pyrolusite but containing a little calcite, was stored in bags. This ore was reported to contain 52 per cent. manganese and to have been obtained principally from the shaft. The road

to Mendoza is over a rather rough rocky surface part of the way, but cart haulage would be feasible if sufficient ore is found.

## Conditions Affecting Manganese Industry in Cuba

Although the owners and operators of manganese properties in Cuba desired to speed up production during the period of the war while the need for the ore was great and the prices were good there were certain hindrances, aside from climatic conditions, that tended to retard their output. Some of these hindrances will continue to affect the manganese industry in Cuba, and unless certain of them are removed and others remedied it will probably be difficult to maintain the industry in the face of reduced prices for ore. It was difficult to obtain and hold a sufficient number of miners at certain mines, because an adequate supply of staple foodstuffs could not be furnished them, they therefore left the mines and went to work in sugar mills, where they more easily obtained food to their liking Mining was also handicapped by shortage of explosives.

There seems but little chance for improvement in the transportation of ore from mines to railroads without assistance from the Cuban Government in building and improving cart roads. Haulage by caterpillar tractors may eventually supplant some of the haulage by animals. The high cost of animal haulage prevents the production of ore from many deposits at a great distance from railroads. The limitation of this traffic to 5 or 6 mo of the year also handicaps production, for though mining might be carried on during practically the whole year, ore would have to be stacked up for many months awaiting the drying of the roads; such storage would lock up considerable capital, and this few of the smaller operators can afford.

Shortage of railroad cars and the inability of the Cuba Railroad to handle adequately all the manganese ore during the dry season, when traffic is heaviest because this is also the cane-grinding season, is also a serious handicap to the output of ore. During 1918, a shortage of ships permitted ore to accumulate at the docks in Santiago faster than it could be removed, but post-war conditions should be better in this respect. The marketing of ore by small producers is attended by more or less friction between buyers and sellers over sampling and analyses. If the Cuban Government could detail two men, one a chemist and the other a man experienced in sampling ore, to act as umpires at Santiago in the sampling and analysis of manganese ore, small producers would be encouraged to steadier efforts; the service might be made self-supporting by charging the cost to the interested parties.

The production of manganese ore seems to have been handicapped by the attitude of some owners of lands and leaseholds, who have raised the

price of royalties so high as to discourage operations. In the course of its trip, the Government party heard complaints of many forms of sharp practice, which are not conducive to a hearty coöperation between property owners, miners, and buyers of manganese ore.

Despite the handicaps outlined there was a strong interest taken everywhere, in Cuba, in developing manganese prospects and the operators of manganese mines made every effort to increase their output. Large investments were made at the larger properties in mine equipment, railroad spurs, and general developments so that these mines were placed in a position to produce a large output of ore. The industrial situation at the close of 1918 is so uncertain, however, and consumers in the United States appear to have such large stocks of ore on hand that it is very doubtful whether there will be an important production of manganese ore from Cuba in 1919.

## Total Production and Reserves of Manganese Ore in Cuba

The complete production of manganese ore in Cuba is not shown in any available statistics, but there are in the volumes on Mineral Resources published by the U. S Geological Survey, records of exports from Cuba from 1888 to 1905, and of imports of manganese ore from Cuba into the United States from 1897 to 1917, except 1911 to 1914, when the mines were idle. From these records and estimates of certain other data, a fair approximation of the quantity of ore produced from the Island and its value is obtained in which the figures are likely to be below, rather than above, the actual production. There is thus indicated a total production of 430,448 long tons, valued at $6,438,481, an average value of $14.95 a ton. The figures for 1918 are lower than was considered probable at an earlier date. The shipments of ore to the United States were curtailed through the withdrawal of vessel tonnage some time after midsummer, and later the armistice still further retarded production. The imports into the United States for 1918, amounting to 82,974 tons, valued at $2,751,193, represent about 19 per cent. of the total recorded output and more than 42 per cent. of its value.

The results of the investigations by Mr. Burch and the writer indicate a total of between 700,000 and 800,000 long tons of manganese ore containing 36 per cent. or more manganese, including material that can be concentrated, in reserve in Cuba, 600,000 to 700,000 tons of which are credited to the Santiago district, 50,000 to 75,000 tons to the Bayamo-Baire district, and the remainder to all other districts including Los Negros, Camaroncito, Trinidad, and Mendoza.

Table 3 indicates the quantity and value of manganese ore exported from Cuba 1888 to 1896, and imports from Cuba into the United States 1897 to 1918, inclusive.

TABLE 3.—*Manganese Ore Exported from Cuba* 1888 *to* 1896, *and Imports from Cuba into the United States* 1897 *to* 1918

| Years | Long Tons | Value | Years | Long Tons | Value |
|---|---|---|---|---|---|
| 1888 to 1896[a] | 77,228 | $695,393[c] | 1906 | 11,701 | $117,050 |
| 1897[b] | 6,992 | 81,126 | 1907 | 30,006 | 262,847 |
| 1898 | 1,600 | 8,026 | 1908 | 1,469 | 13,489 |
| 1899 | 16,359 | 221,785 | 1909 | 2,950 | 11,800 |
| 1900 | 20,582 | 259,348 | 1910 | 2 | 26 |
| 1901 | 21,627 | 307,084 | 1915 | 5,141 | 69,453 |
| 1902 | 36,294 | 285,571 | 1916 | 30,563 | 514,184 |
| 1903 | 17,721 | 111,670 | 1917 | 44,511 | 612,413 |
| 1904 | 16,239 | 80,974 | 1918 | 82,974[d] | 2,751,193[d] |
| 1905 | 6,489 | 35,049 | | | |
| | | | Total | 430,448 | $6,438,481 |

[a] Exports from Cuba. [b] Imports to United States.
[c] Estimated. [d] From Bureau of Foreign and Domestic Commerce

# Recent Studies of Domestic Chromite Deposits*

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(Chicago Meeting, September, 1919)

IN 1827, chromite was discovered near Baltimore by Isaac Tyson, Jr., who initiated the mining of chrome ore and later (1845) the manufacture of chromium compounds in this country. From 1828 to about 1860 the United States, supplanting Russia, supplied chromite for most of the world. Chrome deposits were found also in California, and the Tyson Company shipped ore to Philadelphia by way of Cape Horn.

In 1848, J. Lawrence Smith, an American in the employ of the Porte, discovered large bodies of chromite in Asia Minor. They were developed in 1865 and became the chief source of the world's supply for a number of years. The discovery of the large high-grade bodies in New Caledonia, and later in Rhodesia, with other foreign sources, gives at present an abundant supply for all demands.

For a number of years before the great war, the United States produced only a few hundred tons of chromite annually, for local use, but during the war the demand for domestic ore resulted in a production of more than 82,000 long tons of crude ore in 1918, and it was demonstrated that the United States has reserve deposits equal to a war demand of several years' duration. Now that the war is over, the country is conserving its domestic supplies by employing higher-grade but cheaper ore from foreign countries.

## CHROMIUM-BEARING MINERALS

| | |
|---|---|
| *Oxides:* | |
| Chromite—chromic iron ore | $FeO.Cr_2O_3$ |
| Picotite—chromic iron ore | $(MgFe)O.(Al,Cr)_2O_3$ |
| *Chromates:* | |
| Crocoite—lead chromate | $PbCrO_4$ |
| Phoenicochroite—basic lead chromite | $3PbO.2CrO_3$ |
| Vauquelinite—phospho-chromate of lead | $2(PbCu)CrO_4.(PbCu)_3P_2O_8$ |
| *Sulfates:* | |
| Redingtonite—hydrous chromium sulfate | |
| Knoxvillite[1]—hydrous basic sulfate of chromium | |
| Daubreelite—iron-chromium sulfide (known in meteorites only) | $FeS.Cr_2S_3$ |

* Published by permission of the Director, U. S Geol. Survey.

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[1] G. F. Becker: U. S. Geol. Survey *Monograph* 13 (1888), 279 and 389.

*Silicates:*

| | |
|---|---|
| Chrome diopside | $CaMg(SiO_3)_2\ NCr_2O_3$ |
| Emerald—chrome beryl | $3BeOAl_2O_36SiO_2\ NCr_2O_3$ |
| Uvarovite—chrome garnet | $3CaO(AlCr)_2O_3.3SiO_2$ |
| Fuchsite—chrome mica | $2H_2O.K_2O.3(Al\ Fe.Cr)_2O_3\ 6SiO_2$ |
| Kammererite and kotschubeite—chrome chlorite | $4H_2O,\ 5MgO(AlCr)_2O_3,\ 3SiO_2$ |
| Wolchonskoite—a chrome-bearing clay | |

Of the chromium-bearing minerals listed above, the oxide, chromite, is the only one of economic importance as a source of chromium.

The silicates are of the next most common occurrence. They are usually found in connection with deposits of chromite, as the latest chrome-bearing mineral to form. Chrome garnet occurs as minute bright-green dodecahedral crystals along joint planes cutting chromite, and they are often slickensided by later earth movements.

Crystals of chrome chlorite frequently line cavities in the outer portions of chrome deposits. They were noted especially at the Castle Crags mine, Shasta County, Calif., and at the Deer Creek mine in Wyoming, where an amorphous green clayey deposit like wolchonskoite rests on chrome chlorite in some of the cavities. It is interesting to note that wolchonskoite has recently been identified in this country for the first time by W. T. Schaller from a vein mine near Ely, Nev. Chrome garnet, chrome chlorite, and wolchonskoite largely if not wholly appear to originate in the final hot gaseous emanations from the cooling, though more or less solid, deposit of chromite.

The sulfates have a somewhat similar origin. Knoxvillite occurs in the Redington mercury mine, and as pointed out by Becker it is due to the action of hot solfataric gases on chromite. The same may be true of redingtonite that is associated with knoxvillite.

## Peridotite and Serpentine

Wherever found in considerable quantities, chromite occurs in peridotite or serpentine.

Peridotite is a basic igneous rock; by alteration, it is the source of practically all the serpentine in the Klamath Mountains, Calif. It consists chiefly of olivine, with which may be associated more or less pyroxene, generally enstatite, but in many places diallage, rarely chrome diopside, as well as subordinate grains of magnetite and chromite. The relative proportion of these minerals varies greatly in different portions of a rock mass. A rock composed almost wholly of olivine is dunite; of pyroxene, pyroxenite. Both of these extreme types of peridotite are abundant in the Klamath Mountains, together with intermediate grades, and appear to be products of differentiation during the solidification of the same original magma. Special names, such as saxonite, lherzolite, and wehrlite, have been given to certain of the intermediate forms of

peridotite. Perhaps the most common type in California and Oregon is saxonite, a rock rich in olivine with a considerable quantity of rhombic pyroxene, enstatite. On a weathered surface enstatite gives rise to bright fibrous spots of bastite-serpentine which are generally characteristic of saxonite. This form of peridotite (saxonite) is dominant in Nickel Mountain,[2] 3 mi. northwest of Riddle, Douglas County, Ore., where it contains bodies of high-grade chromite that are being mined.

Olivine and pyroxene both alter to serpentine, but the pyroxenes, excepting enstatite, generally alter less readily than olivine. Thus it happens that weathered saxonite shows patches of bastite while those peridotites rich in diallage have a very rough surface, owing to the projecting crystal grains of unaltered pyroxene left exposed by the wasted olivine that was changed to hydrous silicates and washed away. On the other hand, the surface of weathered dunite is comparatively smooth. Fresh rocks of all these types are well exposed in the higher portions of most of the large serpentine areas of the Klamath Mountains, where they have been laid bare by erosion.

The result of surface weathering upon fresh peridotite is to produce a hydrous alteration of the olivine and pyroxene, liberating oxide of iron which causes the rock surface and soil to be strongly colored yellowish red, but beneath this red coating the fresh rock has the dark grayish green color and vitreous luster of granular olivine and pyroxene. The red surface characteristic of weathered peridotite has caused many portions of the serpentine areas to be called "Red Mountains."

Serpentine is always a secondary mineral and is formed by alteration of olivine and pyroxene in peridotitic rocks. In some parts of the Klamath Mountains the peridotites are fresh or but little altered; at other places they are largely or completely altered to serpentine. As all intermediate degrees of alteration occur, it is not possible to map the altered and unaltered parts separately; however, it is certain that by far the greater portions of the masses outlined on the map (Fig. 23) are serpentine. In general, where the rock is massive the peridotite is unaltered, but where crushed so as to facilitate water circulation it has been altered to serpentine. The outcrops of serpentine are usually greatly fissured, rough, jointed, and sheared, and the fragments slickensided by the movements within the mass, caused not only by mountain-making forces but also by the increase of volume as the rock changes to serpentine. Serpentine generally decomposes, on a gentle slope, to a reddish sterile soil, supporting scanty vegetation, but on a slope where erosion is rapid the fresh serpentine has a yellowish green color and a dull to waxy luster.

A thin section of this rock shows that the chief mineral is serpentine,

[2] Nickel Mountain is named from the occurrence there of an interesting nickel silicate which has been described with the igneous rocks by Dr Geo. F. Kay in the as yet unpublished text of the Riddle Folio. See U. S. Geol. Survey *Bull.* 340, 134–152.

some of which is derived from olivine and other portions from pyroxene; remnants of both minerals may occur in the serpentine to prove its source. Many dustlike particles and grains of magnetite are present in the serpentine and frequently show an irregular meshlike or gratelike arrangement determined by the lines of fracture in the altering olivine and pyroxene, from which the magnetite in serpentine is chiefly derived. Grains or crystals of chromite are usually more or less abundant. In a thin section, these are generally opaque, but when sufficiently thin, by transmitted light they have a coffee-brown color.

The igneous origin of peridotites, including pyroxenites and dunites, is a matter so widely accepted by geologists and petrographers that their origin in the Klamath Mountains and elsewhere on the Pacific Coast needs no special consideration, especially since they are no exception to the general rule.

The peridotites of the Klamath Mountains penetrate not only the Paleozoic rocks, which form the bulk of the central Klamath Mountains, but also the Mesozoic rocks which, although they extend into portions of the Klamath Mountains, constitute, under the general name of Franciscan series, a larger part of the Coast Range. While it is possible that the peridotites were not all erupted during the same epoch, it seems most likely that they were principally erupted about the close of the Jurassic. Those associated with the gneiss in the middle portion of the Klamath Mountains may be older.

## Properties of Chromite

Chromite is iron chromate, represented by the theoretical formula $Cr_2O_3.FeO$, with 68 per cent. chromic oxide ($Cr_2O_3$), and 32 per cent. ferrous oxide ($FeO$). The iron may be replaced by magnesium, and the chromium by aluminum and ferric iron, affording all gradations in composition from spinel ($MgO.Al_2O_3$) and picotite $(Mg.Fe)O.(AlCr)_2O_3$, which may contain 10 per cent. of chromic oxide, up to chromite proper. This gradation in composition was emphasized years ago by Wadsworth,[3] and it is very important to bear it in mind when considering the concentration of chrome ore, for much of the mineral is chemically low-grade and cannot be concentrated into a rich chrome ore by any mechanical process.

The color of chromite is between iron-black and brownish-black. In very thin sections it is coffee-brown or yellowish-red. Its streak is brown, luster submetallic to metallic, fracture uneven, hardness 5.5, specific gravity 4.32–4.57. It is therefore not quite so heavy as magnetite, with which it may be associated, and it may be distinguished by its brown streak and by being generally nonmagnetic.

[3] M. E. Wadsworth: Lithological Studies Harvard College Mus. Comp. Zool., *Memoirs*, **11**.

Chromite crystallizes in the isometric system, usually in octahedrons which are sometimes modified by the dodecahedron. Complete crystals are not common, and large crystals are rare. The largest I have seen was found on the F. M. Stockwell property, 1½ mi. west of Canyon City, Ore. It was nearly half of a regular octahedron, and its octahedral edge was 2 cm. in length. Crystals of chromite are generally microscopic and occur most frequently included in the olivine and pyroxene of peridotites where, among other more or less regular forms, the diamond-shaped, square, and triangular cross-sections of euhedral crystals (Fig. 1*a*) appear. Occasionally one finds a grain of chromite showing free crystallization upon one side but irregular on the other side, where it is anhedral (Fig. 1*b*). The form in which separate grains of chromite occur most frequently in peridotite, as seen in a thin section of the rock, is anhedral, completely irregular (Fig. 1*c*), angular without a definite trace of crystallographic boundary.

In contrast to the angular outline just noted, which is most common in low-grade spotted or disseminated ore, there is another anhedral form

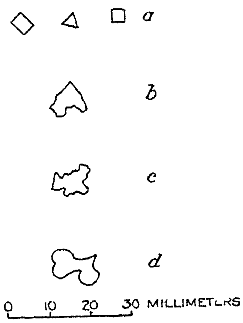


FIG. 1

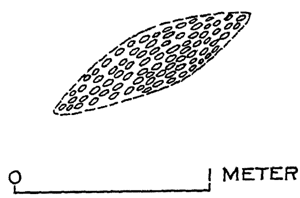


FIG 2.

FIG. 1.—FORM OF CHROMITE GRAINS: *a*, EUHEDRAL; *b*, SUBHEDRAL; *c*, ANHEDRAL; *d*, ANHEDRAL, ROUNDED

FIG. 2—LENTICULAR BODY OF CHROME ORE HAVING NODULAR STRUCTURE, COGGIN MINE, DUNSMUIR, CALIF

characterized by curves (Fig. 1*d*). In some places this is the most abundant form and occurs in orebodies that have a more or less distinct nodular-concretionary structure. It suggests spherulitic forms of early crystallization, from which it is readily distinguished, however, by being wholly crystalline.

## CHARACTERISTICS OF CHROMITE DEPOSITS

### *Form*

Deposits of chromite are generally lenticular in form. That illustrated in Fig. 2 is a sharply defined body of chrome ore inclosed in serpentine; most of the chromite bodies are much longer and wider in proportion to thickness than this one. They are, in fact, bands, layers

or sheets, sometimes spoken of as tabular rather than lenses. The distinctness of the outline varies from abrupt change to perfect gradation. The orebodies are most sharply defined along their flat sides (Figs. 3 and 5*e*), although they frequently send off projections into the peridotite.

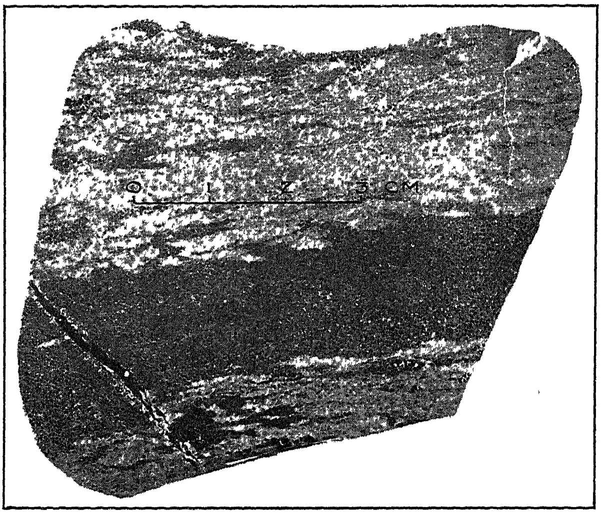


FIG 3

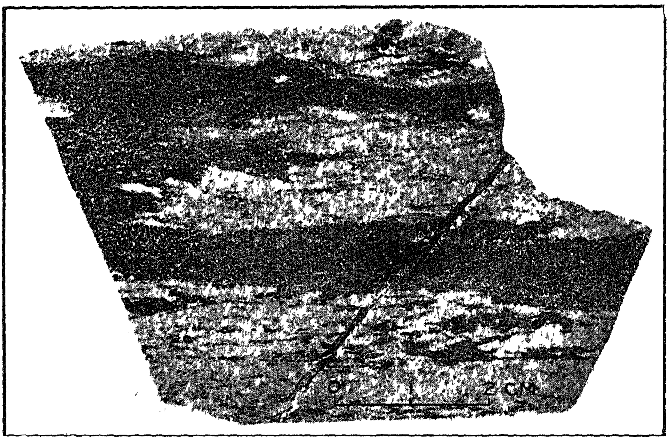


FIG. 4

FIGS. 3 AND 4.—OPPOSITE SIDES OF SPECIMEN FROM SEXTON PEAK, OREGON, SHOWING SHARPLY MARKED BAND OF CHROMITE (DARK) IN OLIVINE (LIGHT) NATURAL SIZE.

Their edges are generally less sharply marked, and the gradation is accompanied by much irregular interfingering of ore and country rock, as shown in Fig. 4. Many of the orebodies are comparatively smooth, especially in sheared serpentine, but others are extremely irregular, as in Fig. 5*d*.

*Size*

The size of chromite bodies ranges from a small nodule to a mass containing many thousands of tons [4] The largest body of chrome ore mined recently in the United States was 150 ft. (46 m ) long, 40 ft. (12 m.) wide, and 54 ft.(16 m.) high. It occurred in the Castle Crags mine on Little Castle Creek, in Shasta County, Calif , and yielded about 12,000

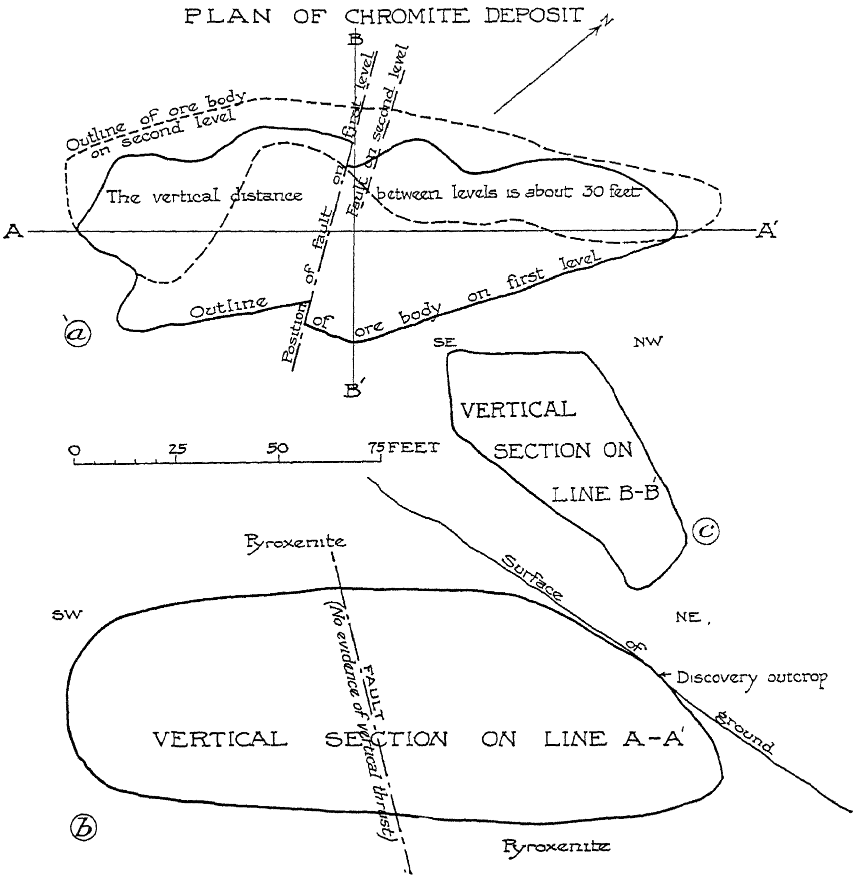


FIG. 5*a*, *b*, *c* —PLAN AND SECTIONS OF CASTLE CRAGS OREBODY. (*J. R Van Fleet* )

tons of merchantable ore; assuming a shrinkage of one-third in grading and washing, the orebody originally contained about 18,000 tons of ore. The form of this large orebody is shown in plan and cross-sections, in Fig. 5, *a*, *b*, *c* by J. R. Van Fleet, the engineer in charge of the mine.

[4] It is reported by Dr. P. Frazer in Vol. CCC (Lancaster County) of the Second Geol Survey of Pa., p. 192, that 95,000 tons of chrome ore was removed from the Wood mine in Lancaster County, Pa The mine was opened in 1828.

Sketches of chromite bodies in eastern Oregon by L. G. Westgate (Fig. 5, *d*, *e*), further illustrate the variability in form, size and border of chromite orebodies. Other large bodies have been found in Fresno, San Luis Obispo, and Eldorado Counties in California, as well as in Josephine and Grant Counties in Oregon, although in some of these cases the ore is contained in a series of overlapping lenses so near together as to be easily

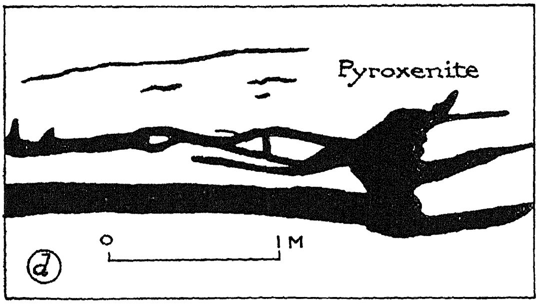


FIG 5*d*.—SECTION OF CHROMITE OREBODY, BRANNAN CLAIM. (*L G. Westgate*)

operated as one mine. The majority of chrome orebodies mined out in the United States within the last few years were relatively small. The number of chrome mines in the United States which contributed to the total output of ore in 1918 was about 240; the average yield, calculated at 50-per cent. grade, was therefore about 270 tons. Very few mines worked upon the same body of ore throughout the year, and many mines operated on five or more orebodies. As compared with the extensive

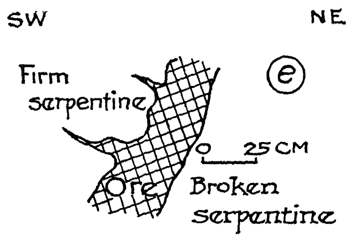


FIG 5*e*.—DIKE-LIKE BODY OF CHROMITE WITH CONTRASTED SIDES. CAMPBELL'S PROSPECT. (*L G Westgate.*)

orebodies of New Caledonia and Rhodesia, those of our Pacific Coast appear small, but in other respects they are much alike.

*Structure*

*Even-granular structure* is the most common, and is best illustrated by disseminated or spotted ore in which the black grains of chromite are evenly scattered among the grains of olivine or pyroxene (Figs. 6 and 7). With an increase in the number or size of the chromite grains the dis-

seminated ore finally becomes compact chromite; even the best ores, however, are seldom found entirely free from small grains of serpentine, olivine, or pyroxene.

*Nodular structure* is characteristic of chromite deposits at many places. The nodules range from 4 to 20 mm. in diameter, and may vary from spherical to lenticular in form. The lentils are generally in parallel position. Fig. 8 illustrates a specimen of nodular ore from Sec. 14, T. 16 S., R. 9 W., on Brush Creek, a tributary of Briggs Creek, in Josephine County, Ore. The wholly crystalline nodules of

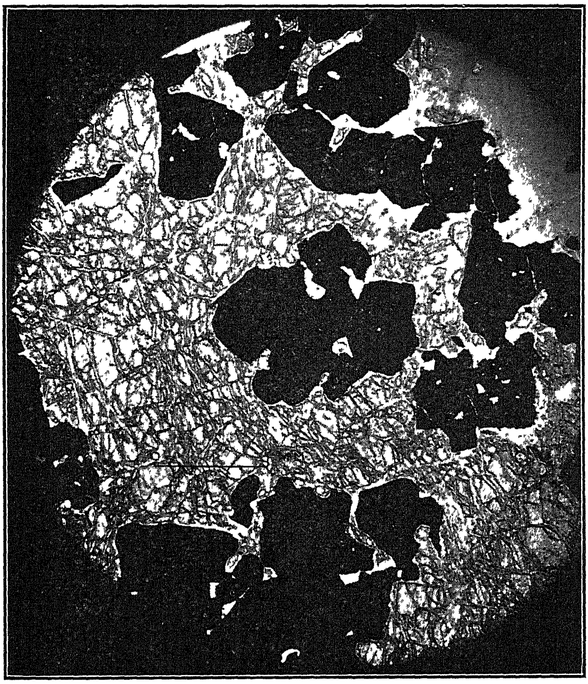

FIG. 6.—EVEN-GRANULAR STRUCTURE OF DISSEMINATED CHROME ORE, CASTLE CRAGS MINE, DUNSMUIR, CALIF. ANHEDRAL (RARELY EUHEDRAL) CHROMITE INDENTED AND INCLUDED BY OLIVINE. × 15.

chromite are in a matrix of serpentine, derived apparently mainly from olivine, of which the meshlike structure outlined by iron oxide still remains surrounded by deep-green serpentine.

The chromite is penetrated by many fractures invisible to the naked eye, most of them belonging to one or the other of two sets approximately at right angles to each other. The more abundant set extends nearly parallel to the length of the specimen and contains less serpentine than the minute transverse fissures which, in the hand specimen, generally show bright reflecting slickensided surfaces.

The surfaces of the nodules are slightly indented and traversed by

FIG. 7.—EVEN-GRANULAR STRUCTURE OF DISSEMINATED CHROME ORE, CASTLE CRAGS MINE, DUNSMUIR, CALIF. ANHEDRAL CHROMITE AND ANHEDRAL OLIVINE. MUTUALLY INDENTING AND ENCLOSING. CHROMITE RARELY EUHEDRAL; GENERALLY BOUNDED BY CURVED LINES AND SERPENTINE BORDERS × 25.

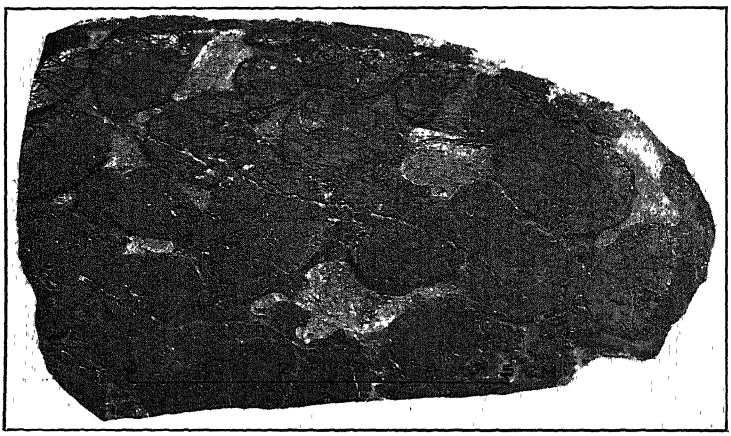

FIG. 8.—NODULAR CHROME ORE, BRIGGS CREEK, OREGON. INTERSTICES BETWEEN NODULES OF CHROMITE ARE FILLED WITH BRIGHT GREEN AND YELLOWISH SERPENTINE, FROM PYROXENE AND OLIVINE. A VEIN OF SERPENTINE CUTS THE CHROMITE NODULES AND ALSO THE INTERSTITIAL SERPENTINE. NATURAL SIZE.

veins of serpentine. The original silicates, chiefly olivine, from which the serpentine is derived, appear to have enveloped the chromite nodules as if the chromite were formed before the silicates.

Fig. 9 represents nodular ore in which the nodules of chromite (4 to 10 mm. in diameter) are wholly crystalline and apparently uniform throughout each nodule, although there is considerable variation in the luster of the grains, which appear on the polished surfaces of the nodules. This may be due to difference in chemical composition or more likely to the different angle of the crystal section. The nodules are much fractured, and the fissures, as well as the internodular spaces, are filled with

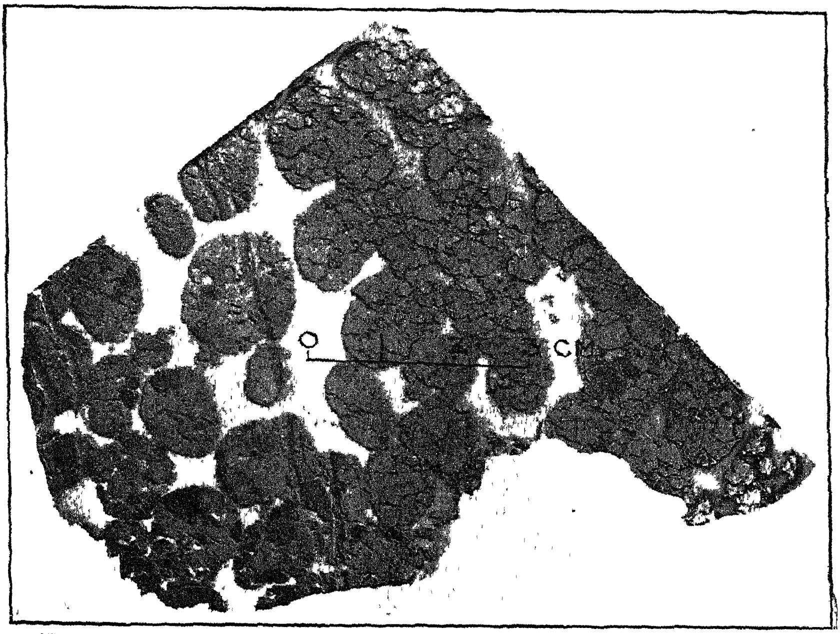


FIG. 9.—NODULAR CHROME ORE, PLACER MINE, ELDORADO CO., CALIF. CHROMITE (BLACK) ENCLOSED BY TREMOLITE (WHITE). THE CHROMITE GRAINS ARE CUT BY VEINS OF SERPENTINE AND TREMOLITE, WHICH REPRESENT EARLIER SILICATES. NATURAL SIZE

a greenish-white mineral, the acicular crystals of which locally have a radial fibrous arrangement; W. T. Schaller identifies this as tremolite, which readily alters to serpentine, with which it is intermingled. In a thin section, the clear, acicular crystals have the distinct cleavage of hornblende, and their occurrence in veins cutting the nodules of chromite shows that the tremolite is secondary. In this respect it is like the serpentine with which it is associated; both form veins in the chromite. The original silicate from which the tremolite is derived was probably pyroxene. Judging from the structure, the primary silicates appear to be younger than the chromite nodules, which they enclose. In one

part of the mine of the Placer Chrome Co. the ore is nodular, grading into compact. The groundmass of the internodular spaces is serpentine, from altered tremolite. These nodular forms strongly suggest the chondrules of meteorites, described by Merrill[5] and others. It is a matter of surprise, however, that in meteorites the chondrules are commonly enstatite and olivine; chromite, although present in many meteorites, is not reported as forming distinct chondrules.

The nodules of chromite, being granular, show no evident concentric or radial structure such as characterizes many concretions. The weathered surface of nodular ore is generally very rough, as in Fig. 2,

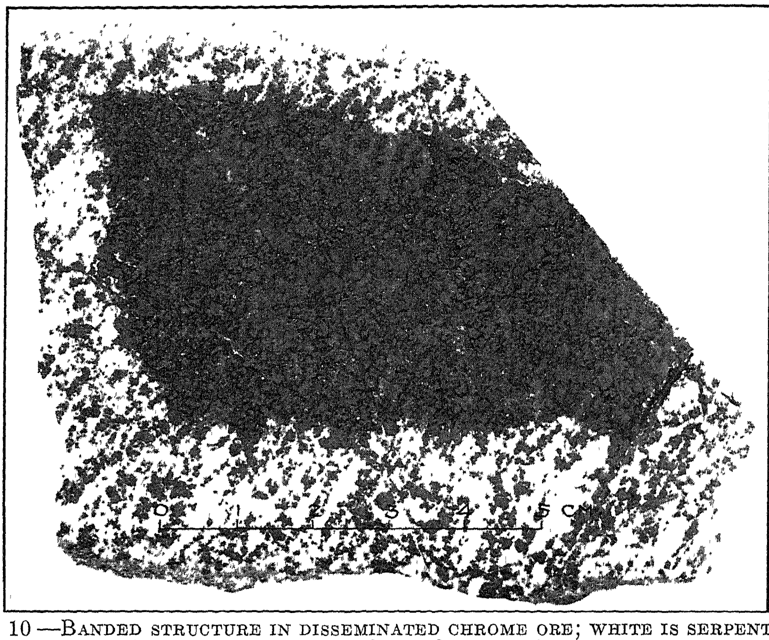


FIG. 10—BANDED STRUCTURE IN DISSEMINATED CHROME ORE; WHITE IS SERPENTINE. L. & M MINE, NEVADA CITY, CALIF NATURAL SIZE.

which represents a lens of nodular ore 4 ft. (1.2 m.) long and 1.5 ft. (0.45 m.) thick, projecting from an exposure or a steep slope of serpentine. The nodules are abundant in some portions of the Coggin mine, and range in size from a pea to a hazel nut, although generally lenticular in form. They appear to contain about 45 per cent. chromic oxide but may be of higher grade, for C. B. Kinney,[6] chemist of the Sawyer Tanning Co., of Napa County, Calif., found nodules of chrome ore from that region to contain 51.20 per cent. chromic oxide.

*Banded Structure.*—In many bodies of ore the chromite is arranged in parallel sheets or layers, alternating with those of olivine, pyroxene,

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[5] U. S. National Museum *Bull.* 94 (1916) 18–20.

[6] Communication to the U. S. Geol. Survey, Oct. 18, 1917

or serpentine, so as to produce more or less distinct banding. In disseminated ore the bands are thin, irregular, and of small lateral extent, but in more compact ore the bands range in thickness from ½ in. or less to several feet. Fig. 10 illustrates disseminated chrome ore with incipient banding. The rock is reported to contain about 15 per cent. of ore and occurs in a belt 30 ft. (9.1 m.) wide a few miles west of Nevada City, Calif., where it is crushed and concentrated, I am told, to about 36-per cent. ore for market.

Figs. 3 and 4 illustrate opposite sides of a hand specimen of banded chrome ore from Sexton Peak, 12 mi. (19 km.) north of Grants Pass, Oregon. The largest and richest layer of ore, Fig. 3, contains many small particles of olivine, and in Fig. 4 the interfingering of the chromite and olivine illustrates the lateral passage of a layer of chrome ore into the country rock. In Fig. 11 the small bands are much more distinct, and Fig. 12 illustrates the structure of a larger layer of compact ore from the same mine penetrated parallel to the banding by small cross-fiber veins of serpentine, which are also banded parallel to the banding of the ore.

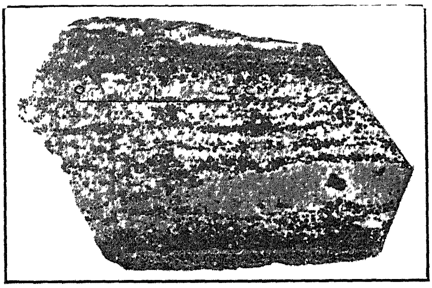

FIG. 11.

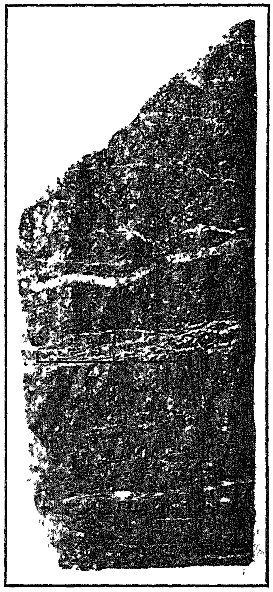

FIG. 12.

FIG. 11.—BANDED DISSEMINATED CHROME ORE, SEXTON PEAK, OREGON. BLACK IS CHROMITE; WHITE, OLIVINE. NATURAL SIZE.

FIG. 12.—BANDED STRUCTURE IN COMPACT CHROME ORE, SEXTON PEAK, OREGON. WHITE STREAKS ARE CROSS-FIBER VEINS OF SERPENTINE. NATURAL SIZE.

The most remarkable bodies of banded chrome ore occur in Siskiyou County in a mass of peridotite extending from Scott River, above Scott Bar, northwest across Klamath River and Sciad Creek, into Oregon. This area is about 25 mi. (40 km.) long and in places 3 to 6 mi. (4.8 to 9.7 km.) in breadth. It embraces many claims, of which six or more groups have developed active mines. The characteristic ores are illustrated by Fig. 13, representing a sample from the Red Butte mine

operated by Reichman and Milne, 3 mi. southwest of Scott Bar. At Red Mountain, the belt of intermingled layers of chrome ore and olivine or serpentine is, in places, about 15 ft. (4.5 m.) thick and has been traced north and south, with a number of interruptions, for several miles.

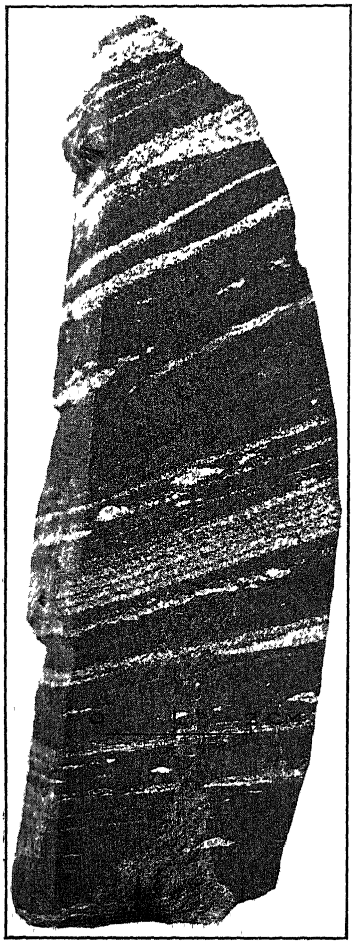

FIG. 13.—BANDED STRUCTURE IN COMPACT CHROME ORE, RED BUTTE, SISKIYOU CO., CALIF. WHITE STREAKS ARE BANDS OF OLIVINE AND SERPENTINE. NATURAL SIZE.

From the Octopus mine, looking north along the slope of Red Butte (Fig. 14), the jointed (1) and banded (2) structure of the peridotite is well illustrated. The jointed structure (1) dipping easterly subdivides the rock so that it appears stratified. The fine banded belt (2) is much less distinctly marked and contains relatively small bands of chrome ore. There are several parallel ore belts in the Red Butte region, all of which have afforded promising prospects, and in August, 1918, these were undergoing rapid development. Locally, in the same region, the structure of the ore is nodular, but this is exceptional.

### *Relation of Magnesite*

The simplest form of association between chromite and magnesite is illustrated in Fig. 15, which represents some banded ore of the Jumbo prospect in the Red Butte region. The four solid layers of chromite are cross-jointed and the joints are filled by a film of magnesite. Each layer has its own cross-jointing, which recalls that of dikelike and veinlike masses seen in the Red Mountain mine of Tehama County, and elsewhere.

Veins of magnesite occur occasionally in deposits of chromite; the best I have seen are in the Red Mountian region of Fresno County, Calif. A most interesting association of magnesite and chromite was noted in a banded specimen (Fig. 16) from a chrome mine 6 mi. south of Newcastle, Eldorado Co., Calif. Chromite with a very small amount of magnetite forms the dark layers, while the white layers are magnesite. In a thin section the chromite is seen to be full of fractures, which are filled with magnesite. The chromite is the oldest

and the only primary mineral of the specimen, and the magnesite appears to replace the peridotite with which the chromite was originally interbanded. The occurrence is of no economic importance, but tends

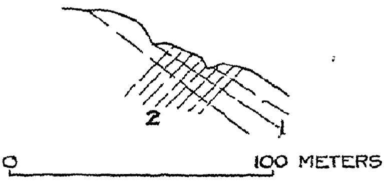


Fig. 14.

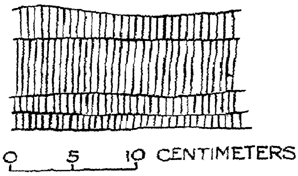


Fig. 15

Fig. 14.—Jointed (1) and banded (2) structures in the peridotite of Red Butte, looking north from the Octopus mine.

Fig. 15.—Cross-jointed layers of compact chromite; joints filled with fine white films of magnesite

to show the primary character of the bands of chrome ore. Vogt[7] has clearly illustrated and described various forms of banded structure in


Fig. 16.—Banded chrome ore, Placer mine, New Castle, Calif. White is magnesite. Natural size.

Norwegian chrome ores, and observes that one set or type of chrome-ore bands may transect another set formed at an earlier stage of development.

[7] *Zeit. prak. geol.* (Oct., 1894) 2, 390–391.

*Deformation*

There is abundant evidence that deformation, accompanied by more or less displacement, has occurred in the deposits of chromite since they were formed. In the solid massive peridotite, as shown by Van Fleet's sections of the Castle Crags orebody,[8] the displacement is relatively small, producing distinct slickensides but without deformation; but where the rocks are altered into serpentine, which is particularly fissile, slippery, and readily susceptible to readjustment under earth stresses, the displacement may be large and cause the fragmentation and separation of deposits into distinct smaller bodies. Bands of solid ore raised to a nearly vertical position may have cross-jointing well developed so as to make the mass closely resemble an intrusive dike or vein.

Banded ores are in some places folded without fracture in a way to indicate plasticity of the rock matter at the time of deformation. Joints and other fissures of the compact ores in northern Shasta County, Calif., and elsewhere, are locally coated with green chrome garnet or purple to pink chrome chlorite, and subsequent rock movements have polished some of these brightly colored surfaces. It appears that the orebodies have experienced several epochs of folding or faulting, and that the one just referred to, which is so conspicuously marked on some of the Rhodesian ore, is perhaps the result of the smallest and latest earth movement.

## Origin of Chromite

The works of Rosenbusch (1873) contain about the earliest reference to this subject; the translation of his work by Iddings[9] (page 126), states that "chromite is common in crystalline rocks rich in magnesia. In these it belongs to the oldest secretions, like magnetite, and is therefore usually inclosed in the next oldest constituents, especially in olivine. Chromite is also widely disseminated in the magnesian rocks of the Archaean formation, particularly serpentine."

As crystalline grains, chromite is widely disseminated in peridotites and serpentine; in addition to this practically uniform distribution, there is a local irregular distribution into more or less well defined orebodies, varying greatly in size, structure, and chemical composition. The widespread chromite grains are considered by most observers as a product of earliest crystallization, preceding that of the olivine and pyroxene in which it is, in many cases, clearly included. Perhaps this early crystallization of chromite may be a fractional crystallization at an early stage of the differentiation of the peridotitic magma. On the other hand, the

[8] Fig. 5*a*, *b*, *c*

[9] H. Rosenbusch. Microscopical Physiography of Rock-making Minerals, 1886. Translated by J. P. Iddings, 1893.

bodies of chrome ore may be regarded as a product of a later and more complete differentiation. In the Klamath Mountains the bodies of chromite, although sometimes associated with saxonite, an early phase in the scale of magmatic differentiation, are commonly associated with dunite or pyroxenite of a later phase of magmatic differentiation.

Vogt,[10] who was one of the first to give special consideration to the origin of chromite deposits, has shown that in the Norwegian occurrences chromite in large masses generally represents purely magmatic separations in peridotite magmas, and that in all cases the chromite appears to be the earlier consolidated constituent.

Lindgren,[11] while apparently accepting Vogt's opinion as to many chrome orebodies, remarks that part of the ore may be secondary, being developed, together with magnetite, during the process of serpentinization from primary chromite, picotite, and chrome diopside. In the Klamath Mountains, the abundance and large size of the chrome orebodies associated with fresh dunite, pyroxenite, and saxonite, shows clearly that these orebodies are primary, and antedate the serpentinization. The same origin is ascribed to the bodies of chromite inclosed in serpentine, although it is clear that much of this ore has been fractured, sheared, and displaced by stresses developed in the formation of serpentine. How much, if any, of the primary chromite may have been dissolved in the process of serpentinization and redeposited as chromite is not easily determined. However, at most it appears to be of small quantity even as compared with the secondary silicates developed. Clarke remarks[12] "When a peridotite alters to serpentine, the refractory chromite remains unchanged."

Lindgren observes, "It is noteworthy that during the weathering of chromite few chromium silicates are formed, while nickel silicates often develop." This might well be expected from the great chemical stability of chromite. Chromium, unlike iron, appears to have no hydrous chromic oxide corresponding to limonite, which is so abundant in nature, especially among the products of weathering. Unlike copper, it has no carbonate readily formed under the influence of surface weathering; nor is it like nickel as noted by Lindgren. In the case of chrome silicates, however, their origin, to be noted presently, appears to be deeper seated than ordinary weathering and aids in understanding the primary character of chromite.

Among the chrome silicates occasionally found with chromite are chrome diopside, chromiferous muscovite (fuchsite), chrome garnet (uvarovite), and chrome chlorites (kotschubeite and kämmererite). Chrome diopside is reported at only one locality in the Klamath Moun-

[10] *Zeit. prak. geol.* (1894) **2**, 384–394.

[11] W. Lindgren: "Mineral Deposits," 747. N. Y, 1913. McGraw-Hill.

[12] F. W. Clarke U. S Geol Survey *Bull.* 616, 3d edition, 344

tains,[13] where, judging from the thin section, it appears to be associated with pink chrome chlorite, of secondary origin, derived from the alteration of the chrome diopside, which appears to be a primary part of the rock structure. The rare occurrence of chrome diopside with chromite is remarkable.

Uvarovite occurs in well defined bright green crystals at the Red Ledge mine, 2 mi. southwest of Washington, in Nevada County, Calif. It occurs in many other localities as a thin chrome-green coating, and is clearly of late secondary origin.

Chrome chlorite was abundant in the surface portions of the large body of chromite won at the Castle Crags mine in Shasta County, Calif., and also in the Deer Creek mine, Wyo, where some good crystals

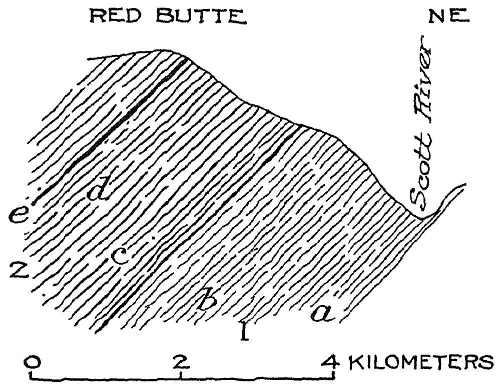


FIG 17.—1a, HORNBLENDE GNEISS, STREAKS OF BLACK HORNBLENDE MINGLED WITH A SMALLER AMOUNT OF PLAGIOCLASE AND A LITTLE QUARTZ; IT CONTAINS DARKER BANDS OF NO. 2 b, HORNBLENDE-PYROXENE ROCK, FORMING BANDS IN THE HORNBLENDE GNEISS; THE PYROXENE CONTAINS MANY INCLUSIONS OF HORNBLENDE. 2, GNEISSOID PERIDOTITE, IN THE BASAL PORTION OF WHICH ARE THIN LAYERS OF HORNBLENDE-PYROXENE ROCK c, d, MASSIVE OR SCHISTOSE-OLIVINE ROCK, CONTAINING THE BELTS OF BANDED CHROME ORE (e) OF RED BUTTE.

occur lining cavities in the ore, evidently being secondary; so also is the associated wolchonskoite.

The banded ores of the Hamburg region of Siskiyou County, Calif., throw an interesting light upon the origin of chromite. A general section, Fig. 17, approximately northeast and southwest across Red Butte to Scott River, disclosed the relation of the banded chrome ore to the underlying gneiss,[14] which is well exposed along the Scott River road for 6

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[13] The specimen was handed me by J. F. Dwyer, of Yreka, who said that it came from the Ewing property, southwest of Yreka, Sec. 24, T. 44 N., R 8 W., Siskiyou County, Calif. (See Fig. 21.)

[14] The term gneiss is here used as defined by Van Hise, U. S. Geol. Survey *Monograph* 47, 782–783, "The term gneiss is defined to apply to a banded rock the bands of which are petrographically unlike one another and consist of interlocking mineral particles"

mi. above Scott Bar. The general parallelism and structural similarity of the banded chrome ore and peridotite to that of the underlying gneiss are evident, and strongly suggest that the banding of the ore is a gneissoid structure, determined by the same conditions that caused the gneissic structure in the underlying rocks. No definite contact was found between the hornblende gneiss and the gneissoid peridotite, but the presence of dark bands of hornblende-pyroxene in both gneissic rocks is evidence of their similar and practically contemporaneous origin. It should be noted, however, that the gneissoid structure does not appear so prominent in the upper portion of Red Butte as it is in the underlying hornblende gneiss. It is least conspicuous in dunite, but this may be due to the predominance of olivine, for in small bands of pyroxene traversing dunite the elongation and alignment of the interlocking grains of pyroxene is decidedly more conspicuous than that of the adjoining olivine.

Although the Hamburg serpentine area contains the most extensive and conspicuous banded chrome ore yet reported, there are other serpentine masses in the Klamath Mountains and elsewhere, as already noted, that contain banded ore. At the chrome mine on Sexton Peak, north of Grants Pass, Ore., it is well developed, also about Red Mountain, near the head of Swift Creek in Siskiyou County, California.

The grains of chromite in the ore bands alternating with bands of olivine or pyroxene are scarcely longer in a direction parallel to the banding than in other directions. In the case of olivine, however, the grains are longest parallel to the banding, and the granular structure is in general the same as that of the gneissoid banded ore of the Hamburg region.

The relation of the banded chrome ore to the associated gneiss indicates that many of the chrome-ore deposits originated under the same conditions that induced the development of the associated gneiss. Van Hise, in considering gneiss, recognizes it as a structure which may be developed in rocks of igneous origin as well as of sedimentary origin, and ascribes the structure to crystallization under differential stress of sufficient intensity to control the direction of crystal growth. No distinct evidence has as yet been observed in the banded chrome ore of the Klamath Mountains to indicate that it is not primary. Furthermore, slight change of stress during the progress of development may fracture ore bands and associated rocks formed during an early stage and permit them to be cut by bands intruded during a later stage of development.

The views considered thus far are those that grew mainly from the work of the German petrographers, but within the last decade the views of French investigators, especially those of De Launay[15] as advanced by Singewald[16] have attracted attention. Singewald refers to the work

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[15] L. De Launay: "Gîtes Minéraux et Métallifères," 1913.

[16] Magmatic Segregation and Ore Genesis. *Min. and Sci. Pr.* (1917) **114**, 733–736.

of Tolman and Rogers,[17] and emphasizes the effect of mineralizers in the later stages of magmatic or subsequent hydrothermal segregation, especially of ore deposits. While many authors consider that there is a sharp distinction between magmatic and hydrothermal deposits, the French school recognizes a complete gradation or transition from purely igneous deposits to hydrothermal veins, and ascribes a large influence to mineralizers.

The segregation of oxidic ores, especially of chromite, while most common near the mass borders of their enveloping igneous rock, occurs throughout its body but is rarely found on its contact Daly[18] notes a body of chrome ore on the lower contact of a laccolith as if differentiated by gravitation, and Bowen,[19] after a special study of differentiation, says "The decision is reached that this differentiation is controlled entirely by crystallization. The sinking of crystals and the squeezing out of residual liquid are considered the all-important instruments of differentiation and experimental evidence to show that under the action of these processes typical igneous rocks series would be formed from basaltic magma if it crystallized (cooled) slowly enough."

As remarked by Singewald,[20] "There are so many admirable illustrations of gradation from rock with feeble concentrations of metalliferous minerals to important orebodies (and this is true of chromite as well as other ores) that no particular significance has been attached to the observation that ore-minerals are often later than the silicates." This sequence, in many cases, appears certain, especially with reference to sulfide ores. Singewald suggests studying the genesis of chrome-ore deposits with the possible influence of mineralizers in mind, and apparently anticipates that they will finally be found in harmony with the deposits of other ores. However, he recognizes that deposits of chromite "now seem to be an exception and to represent a direct segregation as the first product of crystallization from a molten magma."

In considering the genesis of chromite from the viewpoint of the sulfide ores, as advanced by De Launay, it should be borne in mind that terrestrial sulfides of chromium are unknown,[21] although as daubreelite it occurs in meteorites. Clarke has estimated that chromium constitutes about 0.03 per cent of the lithosphere, and it exists almost wholly in chromite,

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[17] C F. Tolman, Jr, and Austin F. Rogers. A Study of the Magmatic Sulfide Ores. Stanford Univ. *Pub.* 26, 1916 *Min. and Sci. Pr.* (1917) **114,** 550

[18] R. A. Daly· "IgneousRocks and Their Origin," 455 N Y, McGraw-Hill, 1914.

[19] N. L. Bowen. Later Stages of the Evolution of the Igneous Rocks. *Jnl. Geol.* (1915) **23,** supplement, 90.

[20] Joseph T. Singewald, Jr : Magmatic Segregation and Ore Genesis *Min. and Sci. Pr.* (1917) **114,** 733–736.

[21] F.W. Clarke: U. S. Geol Survey *Bull.* 616, 696 and 697.

which, as he remarks,[22] occurs almost exclusively in sub-silica rocks and is distinctively a magmatic mineral. Chromite is one of the most refractory and insoluble of minerals and the process of its artificial production, as noted by Clarke, appears to throw but little, if any, light upon its genesis in nature.

It has lately been stated by J. B. Platts[23] that there are many commercial bodies of chromite in Del Norte and Siskiyou counties, Calif., that "are undoubtedly fissure fillings in the ordinary sense." He is of the opinion that some of the orebodies are of magmatic origin but thinks those referred to are exceptional This view I have not been able to confirm, possibly because I may not have seen the deposits referred to, although I saw nearly all of the deposits operated in 1918 within the counties mentioned, especially the large deposits.

It is evidently a matter of prime importance to determine the relative order of crystallization of chromite and the ferromagnesian silicates, olivine or pyroxene, in the orebodies. For this purpose a microscopical study has been made of a number of thin sections from various parts of California and Oregon

The body of chromite mined at the Castle Crags mine, on Little Castle Creek, 3 mi. south of Dunsmuir, Calif., is the largest single body of chrome ore yet discovered on the Pacific Coast. It occurs chiefly in pyroxenite and dunite, the two extreme differentiation products of peridotite, which occupies a large area in that region. Much of the peridotite has been changed to serpentine, but a large portion, chiefly pyroxenite and dunite, but with considerable saxonite, is still fresh in contact with the body of chromite. Fig. 5, *a*, *b*, *c*, represents sections of the orebody;[24] intrusions of the country rock into the orebody are noteworthy. Figs. 6, 7, 18, and 19 are thin sections of ore from that body, and illustrate the relation of chromite to olivine and pyroxene.

In attempting to apply to the problem in hand, we appear to be met at the outset by the fact that the rock masses inclosing important bodies of chrome ore are all of plutonic origin. A corresponding type of effusive rock is lacking, unless it be represented by a portion of those basalts which carry a small amount of chromite either as single regular crystals or groups of crystals included in the silicates, chiefly olivine.

Some information concerning the order of crystallization may be gained by a study of the lines of contact between the grains, especially with reference to the crystallographic lines, euhedrism of one mineral

[22] Personal communication, May 23, 1919.

[23] *Min and Sci. Pr.* (1917) **114,** 872. A similar view was published in U. S. Geol. Survey *Folio* 101 (1904) by H. W. Fairbanks, concerning the chromite deposits of San Luis Obispo County, Calif

[24] J. S. Diller: Chromite in 1916. U. S. Geol Survey *Mineral Resources*, 1916 pt. I, 29–30.

against another, the indentation of one mineral by another, and the complete inclusion of one mineral by another, but the only safe conclusion, as pointed out by Bowen,[25] is as to the order in which minerals ceased to crystallize.

In Fig. 18, the net of serpentine (white), which includes the chromite (black), is derived from pyroxene, which was the last mineral to crystallize. The chromite includes many minute particles of serpentine, which locally have a more or less well defined zonal arrangement with reference to the outline of the including grain, which, as it grew, enveloped the

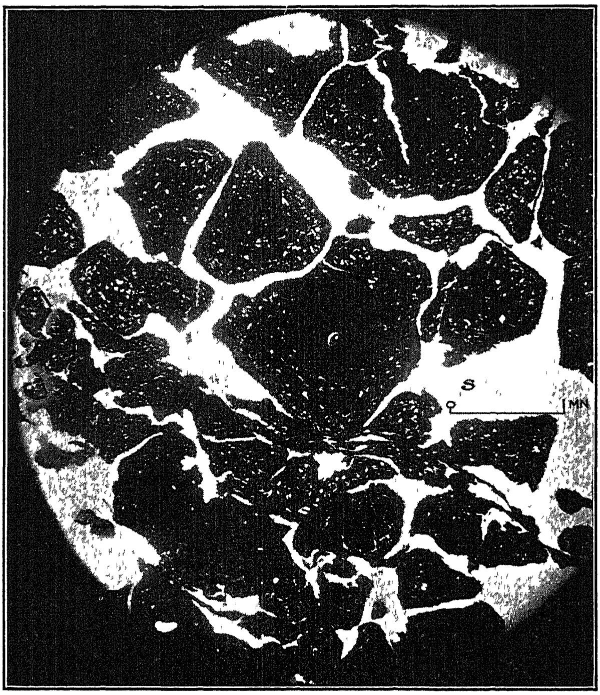


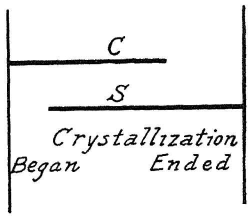

FIG 18.—THIN SECTION OF CHROME ORE FROM CASTLE CRAGS MINE, DUNSMUIR, CALIF. ANHEDRAL CHROMITE (*c*) WITH NUMEROUS ZONAL INCLUSIONS OF SMALL WHITE PARTICLES OF SERPENTINE, REPRESENTING SILICATES OF EARLY CRYSTALLIZATION, AND ENVELOPED BY A NET OF SERPENTINE (*s*) DERIVED FROM A SILICATE OF LATE CRYSTALLIZATION. × 15.

inclusion. That the included particles were crystallizing at the same time as the chromite is suggested by the fact that some of the grains of chromite have no inclusions toward their centers but many inclusions nearer their borders. The included particles are generally anhedral; however, many of the larger ones, about 0.006 mm. long and 0.002 mm. broad, have

[25] N. L. Bowen: Order of Crystallization of Igneous Rocks. *Jnl Geol.* (1912) **20**, 457–468.

straight sides, as if euhedral, and are arranged with their longer axes parallel to the zonal structure. These inclusions are now serpentine, apparently derived from original pyroxene of early crystallization. These early silicates, included in the larger grains of chromite, themselves enclose many smaller particles of chromite and dustlike particles, which appear to be chromite, perhaps the first products of crystallization.

The crystallization of the chrome ore of slide No. 8 may be represented as follows:[26]




*C* representing chromite, and *S* silicates. While a part of the chromite crystallized before the silicates and some of the silicates crystallized after the chromite had finished, most of the chromite and a smaller proportion of the silicates were practically contemporaneous in crystallization.

A sample of disseminated ore (Sec 12) from the Castle Crags mine is represented in Fig. 6. The grain of chromite near the middle of the figure is decidedly roundish and deeply indented by projections of the olivine groundmass, of which one portion, containing a grain of olivine in serpentine, looks as if wholly surrounded by chromite; but the particles are anhedral and the lines from the indenting olivine indicate that it is connected with a projection of olivine from outside the plane of the section. There are some small included particles in the chromite similar to those of Fig. 18, but much less abundant. The fractures in the olivine (Fig. 6) are locally curved about the chromite grains, suggesting flow structure; the general relations here appear to be essentially the same as those of Fig. 18

At another point the disseminated ore in the Castle Crags mine contains a roundish grain of chromite (Fig. 19). It is indented by projections from the surrounding groundmass of pyroxene and olivine, which nearby contains euhedral crystals of chromite. The crystals of chromite afford strong evidence that they are of early crystallization, perhaps antedating most of the particles of silicates by which they are inclosed. The small particles of silicates, now serpentine, inclosed in the larger grains of chromite show that the crystallization of silicates began early and apparently continued after the chromite was completed. A large part of both chromite and silicates may have crystallized contemporaneously.

[26] Graphic method suggested by N. L. Bowen. *Loc. cit.*

The grains of disseminated chrome ore in Fig. 20 are largely euhedral[27] and enclosed in olivine which is partly altered to serpentine; the greater portion of the chromite crystallized before the olivine. The small irregular particles of original silicates (now serpentine) included in chromite may have crystallized slowly, while the chromite proceeded more rapidly to completion, which was followed by the crystallization of most of the silicates.

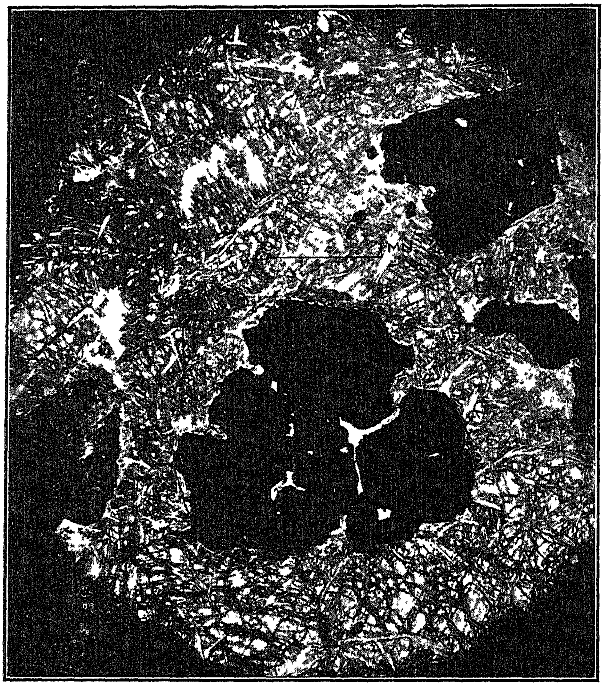


FIG. 19.—THIN SECTION OF CHROME ORE FROM CASTLE CRAGS MINE, DUNSMUIR, CALIF. ANHEDRAL CHROMITE INDENTED AND INCLUDED BY PYROXENE WHICH INCLUDES ALSO SMALL CRYSTALS OF EUHEDRAL CHROMITE. × 15.

About 6 mi southwest of Yreka, toward Fort Jones, there is a remarkable chrome orebody.[28] The thickly set grains of lustrous black chromite form about 60 per cent. of the rock, and the interspaces are filled with brilliant green chrome diopside tinged here and there by a pink

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[27] This general relation of the silicates and euhedral chromite, shown in Fig. 20, is well illustrated by others. See Vogt, *Zeit. prak. geol.* (1894) **2**, 389; R. Beck: "The Nature of Ore Deposits," translated by W. H. Weed, 29, 1905; N. Wyssotzky: "Die Platinseifen gebirge, von Iss und Nischny, Tagil im Ural," 1913, plate VIII, No. 3 and 4, plate IX, No. 3, plate X, No. 1. In Fig. IX he shows euhedral chromite including anhedral platinum and itself included in groundmass of anhedral platinum.

[28] Sample was given me by J. F. Dwyer, of Yreka, and was said to come from Mr. Ewing's property, sec. 24, T. 44 N., R. 8 W.

chrome chlorite derived from the alteration of the diopside. In the thin section, Fig. 21, the chromite is seen to be limited at a number of places by straight lines—crystal boundaries— and in so far is euhedral; but the interferent crystallization of the chromite itself prevented the development of perfect crystals. The crystallographic planes of the chromite appear to indent and limit the pyroxene which was, at least in part, the last mineral to crystallize. The diopside is fresh and appears to be of primary origin. If so, one wonders why chrome diopside is not a more common pyroxenic associate of chromite. The veins of chrome chlorite, tremolite,

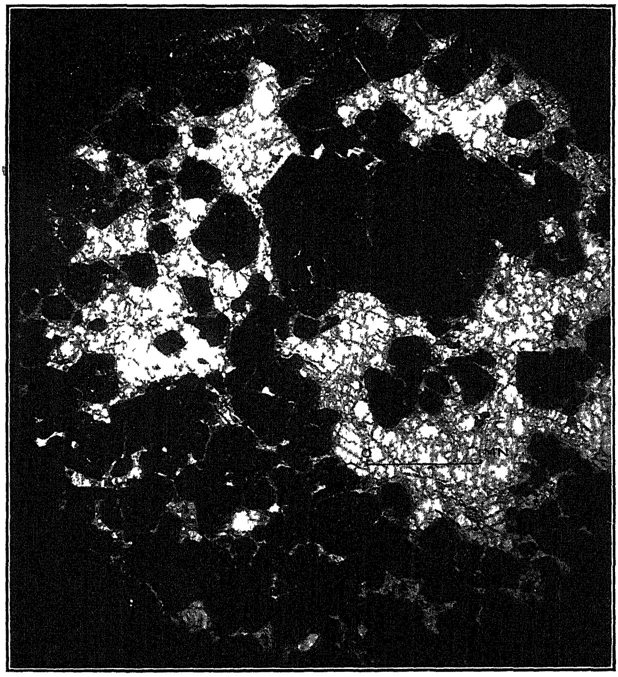

FIG. 20.—EUHEDRAL CHROMITE INCLUDED AND INDENTED BY ANHEDRAL OLIVINE. SCIAD (DOLBEAR) MINE, KLAMATH RIVER, CALIF. × 15.

and chrome garnet, which appear to be of hydrothermal origin, cutting fresh unaltered masses of chromite, relegate the chromite they penetrate to an earlier stage of development, that is to the magmatic stage.

## SUMMARY REGARDING CHARACTERISTICS AND ORIGIN OF CHROMITE DEPOSITS

1. In the Klamath Mountains, areas of peridotite and serpentine are numerous, the serpentine being derived wholly from alteration of peridotite.

2. The various phases of peridotitic rocks are products of differentia-

tion from a common magma, and range from normal peridotite, composed of olivine and pyroxene (with accessory chromite and magnetite), to pyroxenite, in which the silicate is wholly pyroxene, or to dunite, in which the silicate is wholly olivine.

3 Both peridotite and serpentine contain not only disseminated grains of chromite, but segregated bodies commercially interesting as sources of chromite. These orebodies may have resulted from the differentiation of the magmatic mass.

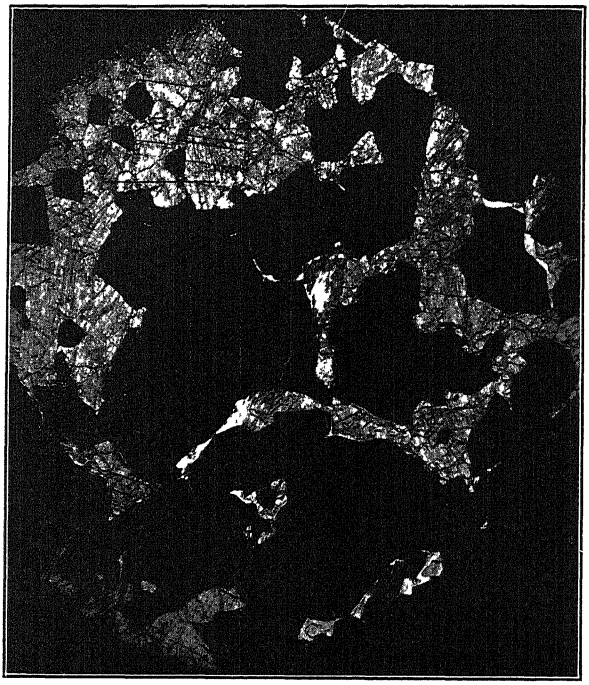

FIG. 21.—EUHEDRAL CHROMITE INCLUDED AND INDENTED BY PYROXENE (CHROME DIOPSIDE) IRVING PROPERTY, NORTHEAST OF YREKA, CALIF. × 15.

4. Grains of chromite, whether singly or in aggregates, may be euhedral (bounded wholly or partly by crystallographic faces); but most of them are anhedral, having crystallized under conditions of interference which prevented the development of crystal faces.

5. The chromite bodies, where unmodified in form, in some deposits are irregular, lens-shaped, short and thick, while in others they are relatively thin, long, and broad, resembling sheets or layers.

6. The structure of these deposits may be even granular, nodular, or banded, according to the distribution of the ore with reference to associated silicates.

7. In the even granular structure, the chromite grains are distributed throughout the associated silicates with approximate uniformity; in

nodular ore the grains are segregated in nodules resembling concretions, and in banded ore they are arranged in bands alternating with bands of silicates.

8. The magmatic origin of chromite is generally conceded, but its exact place in the continuous series of events, beginning with crystallization in the magma and ending with hydrothermal action, may be uncertain.

9 Evidence concerning the proportion of crystallographic forms indicates that the generally accepted view, that chromite is one of the earliest minerals to begin to crystallize in a magma, is probably correct.

10. The earliest chromite was soon followed by the crystallization of small particles of silicates.

11. The crystallization of chromite and silicates proceeded together, but the chromite crystallized more rapidly than the silicates, which it enveloped, and thus completed its crystallization before the silicates.

12. The final crystallization of silicates took place after that of the chromite was completed, and filled the remaining spaces.

13. While it is true that chromite is the earlier product of crystallization and the silicates, olivine and pyroxene, the later, it must be remembered that they are largely contemporaneous, as shown by their mutually interlocking anhedral boundaries.

14. Bowen[29] has reached the conclusion that differentiation is controlled entirely by crystallization and states that "the sinking of crystals and the squeezing out of residual liquids are considered the all-important instruments of differentiation."

15. The chromite deposits resulting from magmatic differentiation under the controlling factors noted above, which may be considered as producing even granular bodies, may be greatly modified by the operation of other concurrent or subsequent factors causing variations of original forms, as well as later deformation or displacement.

16. The nodular ore appears to be simply concretionary, but the special form of serpentine and of tremolite associated with the nodules suggests unusual local conditions different from those which generally prevail. However, the order of development of chromite and silicates appears to be the same as elsewhere.

17. The banded ore, where best developed, is closely related to gneiss and the structure of both has the same origin

18. As urged by Van Hise,[30] the gneissic structure may be developed by differential stress during the slow cooling and crystallization of an intruding magma.

19. The variations in the differential stress may fracture a crystallized

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[29] N L Bowen· *Jnl. Geol* (1915) **23,** supplement. [30] *Loc. cit.*

mass of ore or peridotite and intrude into it ore which, in the process of differentiation, has not yet fully solidified.

20. No conclusive evidence has been seen by the writer anywhere indicating the secondary origin and deposition of chromite. It is sometimes residual, but has always appeared of primary magmatic origin.

21 The magmatic origin of the sulfide ores, as well as of ilmenite and magnetite, advocated by Tolman and Rogers,[31] and ably seconded by Singewald,[32] has been regarded by these authors as indicating that chromite belongs in the same category; that is, that chromite was generally formed later than its associated silicates, and that its development was rendered possible by mineralizers at a late magmatic stage.

To the writer, it appears that while much of the chromite and silicates crystallized at about the same time, some of the chromite appears to be earlier than the earliest silicates and much of the silicates is later than all the chromite, thus essentially confirming the views of Vogt, Beck, and Wyssotzky.[33]

## Distribution of Chromite in the United States

As shown in Fig. 22, the chromite regions of the United States are associated with the great mountain belts of the Atlantic and Pacific coasts and the Rocky Mountains, in California, Oregon, Washington, Montana, Wyoming, Pennsylvania, Maryland, and North Carolina.

### *California*

The occurrence of chromite is limited practically to those mountain ranges that contain masses of basic intrusive rocks belonging to the peridotite group, which were involved in the development of the Sierra Nevada and Coast Ranges of California, and the Klamath Mountains of California and Oregon.

*The Sierra Nevada.*—The Sierra Nevada is a great tilted block mountain 375 mi. (603 km.) long and 70 mi. (113 km.) broad, having a short steep slope to the east but a long gentle slope to the west. The higher portion is composed mainly of granitic rocks, but the middle and lower western slopes of the northern part are of sedimentary rocks, chiefly Carboniferous. They are cut by a belt consisting of numerous masses of peridotite, which are largely changed to serpentine and at many places contain bodies of chromite. Proceeding northward along the range from Tulare County, chromite is mined in the following counties: Fresno, Mariposa, Tuolumne, Calaveras, Amador, Eldorado, Placer, Nevada, Sierra, Butte, and Plumas;

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[31] *Loc cit*, 68 [32] *Loc cit*

[33] N. Wyssotzky· "Die platinseifen gebirge, von Iss und Nischny, Tagil im Ural," pl VIII, 3 and 4, XI, 3 and 4, X, 1.

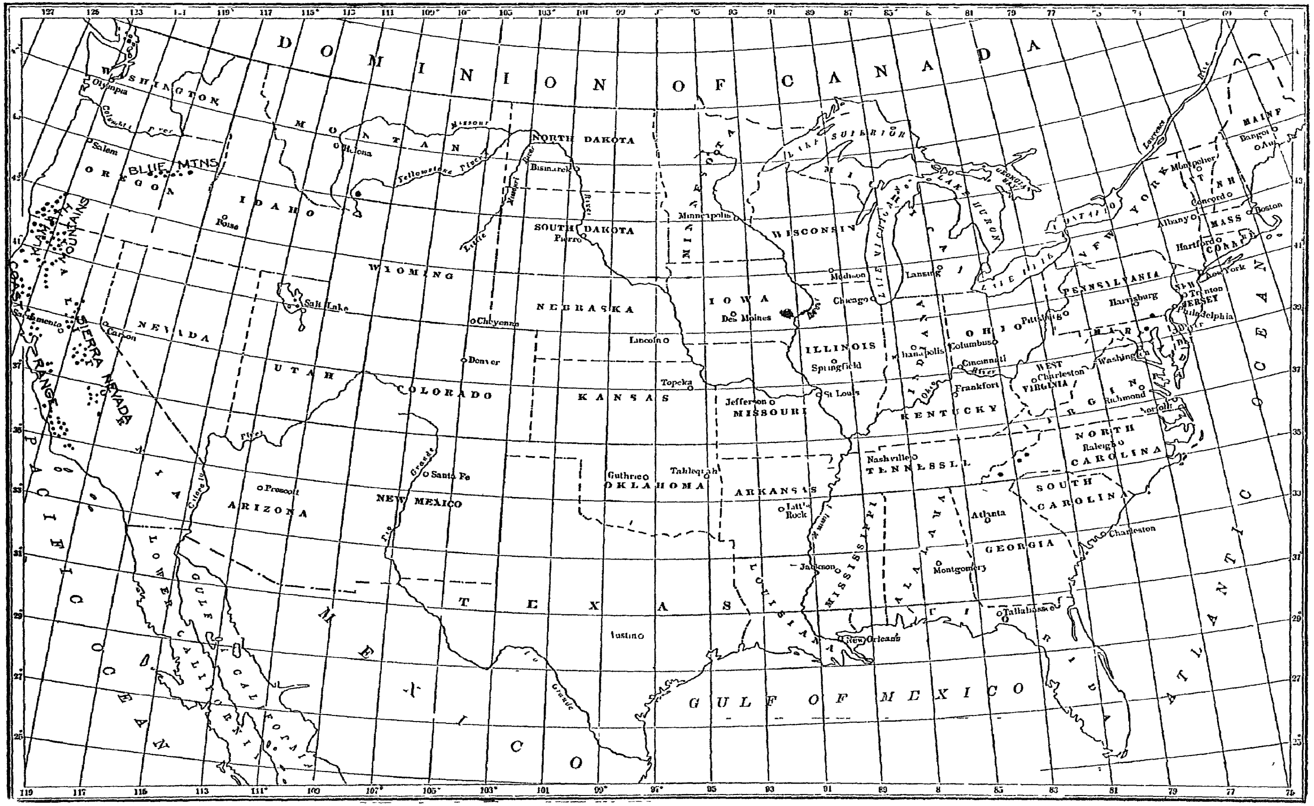


FIG 22.—MAP OF THE UNITED STATES SHOWING CHROMITE LOCALITIES IN CALIFORNIA, OREGON, WASHINGTON, MONTANA, WYOMING, PENNSYLVANIA, MARYLAND, AND NORTH CAROLINA.

the total shipped production in 1918 was of 30,810 short tons, with an unshipped stock of 14,424 tons, making the total 1918 output of the Sierra Nevada 45,234 short tons.

Eldorado is the ranking county of the state; it shipped 14,243 short tons in 1918 and reports 7112 short tons on hand. The three most important mines of the county are the Placer Chrome, Pilikin, and Latrobe. Much of the ore is concentrated to high grade.

*The Coast Range.*—This range, from Santa Barbara to the mouth of the Klamath River, has a length of about 560 mi. (901 km ) and a width of about 30 mi. (48 km.), passing through more than 12 counties. It is composed of igneous and sedimentary rocks of the Franciscan series, of Mesozoic age. The rocks have been greatly crushed; the basic intrusives, having been altered to serpentine at many points, contain notable bodies of chromite. The California state council of defense, cooperating with the U. S. Geological Survey, examined and reported on 166 chrome deposits in the Coast Range, of which 73 were producing mines with a total output of 18,805 short tons of chromite sold and shipped in 1918. The three principal producing mines of the Coast Range, the Castro, the Manzanita, and the Pick and Shovel, are all in San Luis Obispo County.

The most southern deposits are in Santa Barbara County, northwest of which is the large rich chrome field of San Luis Obispo, which has been worked at intervals for a number of years. The geology of this field has been mapped and described by H. W. Fairbanks.[34]

The chrome-bearing rocks are of the normal types of peridotite and pyroxenite, largely changed to serpentine. They occur in relatively long, narrow, irregular, dike-like masses in which the stringers and bunches of chromite are more or less irregularly distributed, and disconnected. North of San Luis Obispo, these are important producing centers in Santa Clara, Stanislaus, Alameda, Napa, Sonoma, Lake, Mendocino, and Humboldt Counties.

Owing to the sudden collapse of the chrome market at the close of the war, many producers were left with large stocks of ore on hand, amounting in all to 7859 tons, making a total production of 26,164 short tons for the Coast Range. More than three-fourths of this total came from San Luis Obispo County, which shipped 11,127 tons of ore and still has several thousand tons in stock.

### *The Klamath Mountains*

The Klamath Mountains of northwest California and southwest Oregon, outlined in Fig. 23, have a length north and south of 260 mi. (418 km.), 170 mi. (274 km.) in California and 90 mi. (144 km.) in Oregon. Their average width is about 80 mi. (129 km.).

[34] U. S. Geol. Survey *Folio* 101 (1904).

The eastern boundary is sinuous against the lavas of the Cascade Range. The western border is a long shallow curve with its middle portion along the coast between the Rogue and Klamath rivers. The northeastern limb of the curved border, overlapped by the Coast Range of Oregon, trends toward the Blue Mountains, while the southeastern limb,

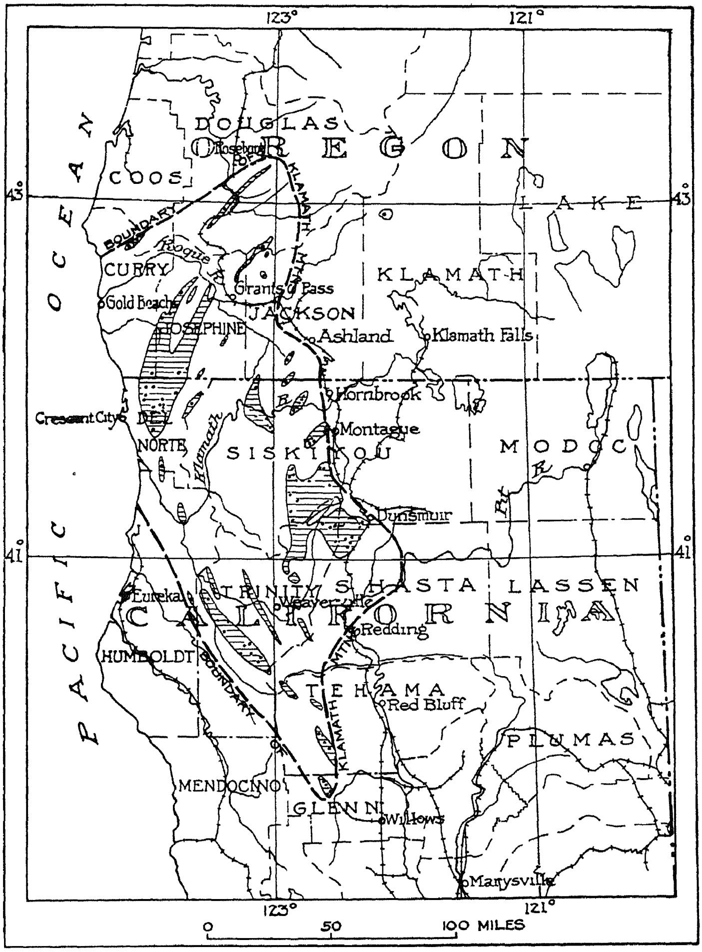


FIG. 23.—MAP OF THE KLAMATH MOUNTAINS, SHOWING LOCATION OF SERPENTINE AREAS AND CHROMITE DEPOSITS.

overlapped by the Coast Range of California, trends toward the Sierra Nevada, indicating the ranges to which the Klamath Mountains are most closely allied.

The geological survey of the Klamath Mountains is not yet complete. Only the Redding and Weaverville quadrangles of California, and the Roseburg, Port Orford, Riddle, and the general outlines in the Grants

Pass quadrangles of Oregon have been finished. Much of the middle portion of the Klamath Mountains has been traversed in considerable detail, so that the prevailing formations are known and their general distribution outlined. They are composed of rocks that are older and more complicated in structure than those of the Coast Ranges of California and Oregon, and are closely related, in materials and structure, to the Sierra Nevada and the Blue Mountains.

The map of the Klamath Mountains displays two broad belts of the same or closely related rocks, of which the most ancient, pre-Cambrian crystalline rocks, mica and hornblende schists, in the western belt form the South Fork Mountains, and in the eastern the bulk of the Salmon Mountains. In a general way these two belts define, by their eastern borders, broad stretches of Paleozoic rocks including many areas of slates, sandstones and limestones, apparently of Silurian and Carboniferous age, besides great volumes of Paleozoic volcanic rocks and later intrusives of both acidic and basic types. The acid types are represented by various forms of granitic and dioritic rocks, while the basic types are gabbros and peridotites.

Peridotite alters to serpentine, and whether fresh or altered it is generally called serpentine by miners. Peridotite and serpentine are the only rocks that contain commercial bodies of chrome ore, and they are the only ones of which the areas are outlined in the accompanying map (Fig. 23) of the Klamath Mountains.

*Distribution of Peridotite and Serpentine.*—Serpentine appears to be relatively more abundant in the Klamath Mountains than anywhere else in California.

The serpentine masses of the Klamath Mountains are very irregular (Fig. 23), and their forms vary in different portions of the mountains. Those of the western half are long and narrow, and parallel to the great curve of the Klamath Mountain boundary on the west. In Trinity County, Calif., their greatest extent is northwest and southeast, and they have the large structural features pointing toward the Sierra Nevada. In Del Norte County, Calif., and the adjacent portion of Oregon, their trend is southwest and northeast, toward the Blue Mountains of Oregon. Between these two arms of the curve, especially east of the Salmon Mountains, which appear to be the solid core of the Klamath Mountains, the masses of serpentine are not only large and very irregular, but more nearly equal in length and breadth.

*Distribution and Production of Chromite.*—In the map of the Klamath Mountains (Fig. 23) the deposits of chrome ore are located by dots. There were 147 producers of chrome ore in 1918; 120 in the 6 counties of California and 27 in the 5 counties of Oregon, with a total shipped production of 27,185 short tons of all grades; of this amount 21,021 tons came from California and 6164 tons from southwest Oregon. The total

amount of ore reported mined in the Klamath Mountains in 1918 was 48,571 short tons, of which 21,386 tons were unshipped.

### *Blue Mountains of Eastern Oregon*

The chromite deposits of eastern Oregon, of which the general location is shown in Fig. 22, occur chiefly in Grant County, about John Day Valley between Dayville and Prairie City, the main point of railroad shipment. A less important group occurs in the valley of Granite Creek, near Sumpter. There are small areas in Baker County, on Conner Creek, about 20 mi. (32 km.) north of Huntington, and in Malheur County, 15 mi. (24 km.) west of Huntington, where it is associated with red jaspery beds of chert full of radiolaria.

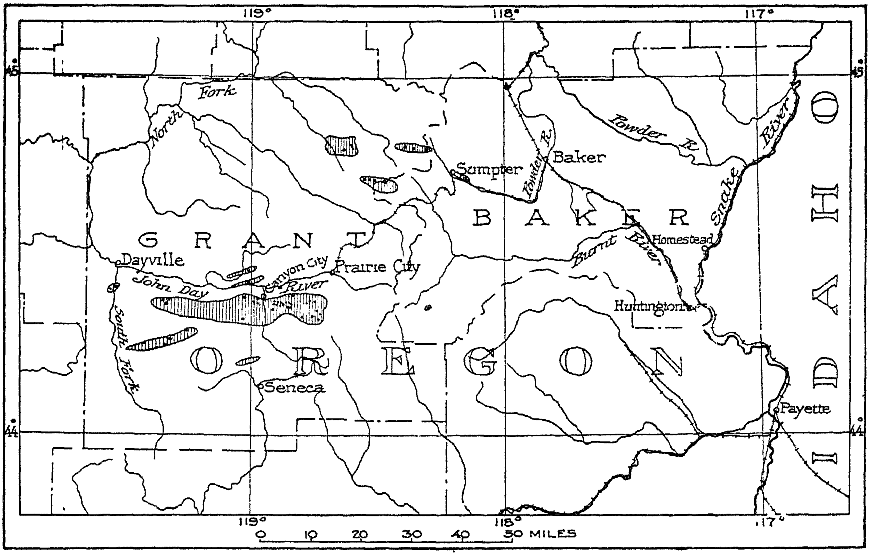


FIG. 24.—MAP SHOWING SERPENTINE AREAS AND CHROMITE DEPOSITS OF EASTERN OREGON. (*L G. Westgate.*)

Lewis G. Westgate examined the Blue Mountain region in August, 1918, he mapped the serpentine area (Fig. 24), and indicated the location of the mines. His report, which is my principal source of information, is now in course of publication by the United States Geological Survey.

The general geology of the Blue Mountains is similar to that of the Sierra Nevada and Klamath Mountains. There are two main series of rocks—an older series of Paleozoic and Mesozoic sedimentaries with some interbedded lavas, cut by igneous intrusives, and a younger series of Tertiary lavas with associated sedimentary beds. The sedimentaries of the older series are slates and shales, with a small amount of limestone.

Carboniferous, Triassic, Jurassic,[35] and early Cretaceous[36] fossils have been identified in these rocks of the Blue Mountains Most of the series has been so folded and faulted that it is difficult to work out its structure in detail.

This ancient series of sediments, like those of the Sierra Nevada and Klamath Mountains, was intruded by masses of acid and basic igneous rocks, among which peridotite is one of the most important. The serpentine of eastern Oregon is in many places so completely crushed and sheared as to obliterate the original features of the rock from which it was derived, but in other places, especially around the large masses of chromite, where the rocks are more resistant, much of the olivine and pyroxene of the original peridotite are preserved.

As pointed out by Westgate, there are two main modes of occurrence

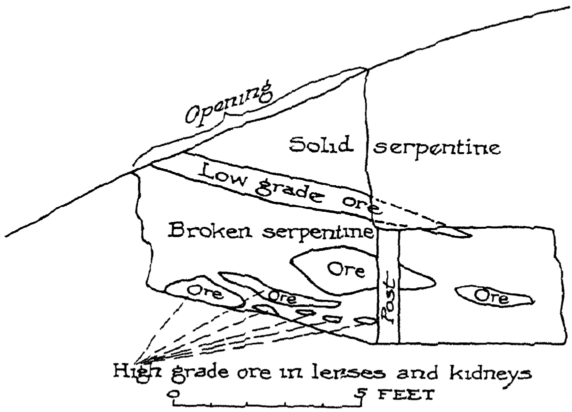


FIG. 25.—CHROMITE LENSES IN SMITH AND GEITSFIELD PROSPECT, CANYON CITY, OREGON (*L. G. Westgate*)

of chromite in this region. On most of the properties the chromite is in vein-like bodies, lenses, or kidneys, in shattered serpentine. Here the original rock has generally gone over wholly to serpentine, and its earlier character is a matter of inference. On other properties, and especially at the larger mines, the chromite occurs in vein- or dike-like masses, or in irregular bodies of considerable size, in a serpentinized rock of basic character. This country rock, though jointed, is much less shattered and serpentinized than that of the first class. Practically all of the chromite mines of eastern Oregon belong to the first class except some of those in the John Day area between Prairie and Canyon City. Fig. 25 illustrates

[35] Chester W. Washburne: Notes on the Marine Sediments of Eastern Oregon. *Jnl. Geol.* (1903) **11**, 224–229.

[36] The present writer found fossils in a limestone by an outcrop of chromite and serpentine on the ridge a short distance east of the road from Canyon City south to Bear Valley. The fossils were identified by Stanton as Knoxville.

some of the sizes and shapes of chromite bodies inclosed by sheared serpentine in the wall of the Smith and Geitsfield mine, 1 mi. west of Canyon City.

Within the mapped areas of serpentine in eastern Oregon about 60 bodies of chromite have been noted, and a third of them are described in Westgate's report. Most of the mines are shallow workings, but the Ward and the McIntyre mines have reached a depth of 70 ft. (21 m.); both are near being abandoned.

The three most important mines of eastern Oregon in 1918 were the Chambers mine of the Blue Mountain Chrome Co., the Iron King of the Zenith Chrome Co., and the Black Jack, of the Grant Chrome Co. Of these the Iron King is the largest; it has been in operation several years and is said to have yielded 6000 tons of crude ore averaging 32 per cent. chromic oxide. The chrome ore occurs in lenses in shattered serpentine and is worked in a large open cut. One lens on the back face of the cut in July, 1918, showed dimensions 10 by 4 ft. (3 by 1.2 m.). No vein or lead is followed; the whole rock is quarried, the ore being taken out as the lenses are reached. The shattered serpentine containing the chromite is separated from the firmer serpentine, especially on the north, by a curving, slickensided, fault face. Serpentine breccia lies between and is cemented to form a resistant rock.

The Chambers mine is on a large body of low-grade ore in serpentine; one-fourth is too poor to work; the rest is generally 31 to 32 per cent. $Cr_2O_3$; the best (36 per cent.) was worked first by open-cut and later by glory-hole about 40 by 100 ft. and 40 ft. deep. The large orebody appears to be more than 100 ft. (30 m.) long, 20 ft. (6 m.) wide, and over 46 ft. (14 m.) in depth. It cannot be followed to any distance east or west. A smaller body of ore and some float occur to the southeast of the main body.

Although some of the chromite shipped from eastern Oregon is reported to run as high as 50 per cent., and much of it over 40 per cent., the greater portion of the ore shipped runs down to 26 and 28 per cent. The production and shipment of chromite began in eastern Oregon in 1917, and about 3700 tons was shipped during that year. In 1918 the shipments were increased to more than 14,500 tons.

Westgate's estimate of the amount of ore in sight in the summer of 1918 gives 4000 tons of ore running 40 per cent. or more $Cr_2O_3$, and 25,000 tons from 30 to 40 per cent. He remarks that the ultimate production from this region will doubtless be much greater than this

## Production of Chromite in California and Oregon

The quantity and value of the chrome ore sold in California and Oregon during 1918, by counties, are given in Tables 1 and 2.

TABLE 1.—*Shipments and Stocks of Chromite in California* in* 1918

(Short Tons; all Grades of Ore)

| County | Shipments | Stocks | Value of Ore Sold | Average Price per Ton |
|---|---|---|---|---|
| Alameda | 187 | 315 | $7,749 00 | $41.44 |
| Amador, Mariposa, Plumas, Sacramento, Sierra | 470 | 770 | 20,591.20 | 43 81 |
| Butte | 1,554 | 1,270 | 56,900 00 | 36.61 |
| Calaveras | 2,819 | 832 | 134,746 05 | 48 15 |
| Del Norte | 7,903 | 4,345 | 380,536 34 | 48 15 |
| Eldorado | 14,243 | 7,112 | 556,586 37 | 39 09 |
| Fresno | 2,995 | 975 | 126,947 82 | 42 39 |
| Glenn | 1,466 | 520 | 72,046 17 | 49 14 |
| Humboldt, Mendocino, Monterey | 790 | 2,195 | 38,194 68 | 48 35 |
| Lake | 534 | 62 | 27,108 35 | 50 76 |
| Napa | 1,457 | 533 | 55,599 00 | 38 16 |
| Nevada | 2,362 | 1,334 | 64,531 62 | 27 32 |
| Placer | 4,081 | 1,339 | 195,704 54 | 47 95 |
| San Benito, Santa Barbara, Santa Clara | 784 | 923 | 46,452 50 | 47 95 |
| San Luis Obispo | 11,127 | 3,248 | 545,629 29 | 49.01 |
| Shasta | 2,019 | 1,045 | 89,327 84 | 44 24 |
| Siskiyou | 6,125 | 2,917 | 261,962 03 | 42.77 |
| Sonoma | 920 | 145 | 43,038 21 | 46 90 |
| Stanislaus | 3,006 | 438 | 77,026 16 | 25.62 |
| Tehama | 1,952 | 4,790 | 95,610 54 | 48.98 |
| Trinity | 1,556 | 2,729 | 57,021.14 | 36.64 |
| Tulare | 630 | 125 | 26,546 00 | 42.13 |
| Tuolumne | 1,656 | 667 | 80,730 17 | 48.73 |
| Total California | 70,636 | 38,629 | $3,060,585.02 | $43 33 |

* Subject to revision.

TABLE 2.—*Shipments and Stock of Chromite in Oregon* in* 1918

(Short Tons; all Grades of Ore)

| County | Shipments | Stocks | Value of Ore Sold | Average Price per Ton |
|---|---|---|---|---|
| Baker and Wheeler | 103 | 130 | $2,681 03 | $26.03 |
| Coos, Curry, Douglas | 600 | 1,327 | 32,575.58 | 54.29 |
| Grant | 14,401 | 2,479 | 549,882.70 | 38 18 |
| Jackson | 168 | 50 | 6,718.95 | 39.99 |
| Josephine | 5,396 | 3,663 | 263,191.62 | 48.77 |
| Total Oregon | 20,668 | 7,649 | $855,049 88 | $41.37 |

* Subject to revision.

*Washington*

Workable deposits of chromite in serpentine occur on Cypress Island, near Anacortes, and in the vicinity of Mt. Hawkins, Kittitas County. A small amount of high-grade ore has been produced at each locality, and on Cypress Island considerable ore suitable for concentration has been developed. In the vicinity of the deposits mentioned, and in the northern Cascade Mountains, there are large incompletely prospected areas of serpentine in which additional orebodies will doubtless be found. The deposits on Cypress Island were visited in September, 1917, and those near Mt. Hawkins in July, 1918, by J. T. Pardee, who prepared this statement.

Chromite was mined on Cypress Island in 1917 by the Bilrowe Alloys Co., for use in their smelter at Tacoma. Prior to this no production is reported, but a moderate amount of development work had been done on several claims that were located 15 or 20 years earlier. In 1918 the Cypress Chrome Co. reported shipments, one of which was treated at the laboratory of the Faust Concentrating Co., Seattle; samples from the same lot were tested at the Seattle station of the U. S. Bureau of Mines.

Analyses reported by the Cypress Chrome Co. show an average of 47½ and 25½ per cent. chromic oxide, respectively, in shipments that represent the selected and the milling grades of ore. From the latter, a concentrate containing 45 per cent. chromic oxide was produced at the Faust concentrating plant, and a product containing 50 per cent. or more was obtained from the samples tested at the Bureau of Mines. Samples from the deposits near Mt. Hawkins are said to assay as much as 51 per cent. chromic oxide. These high percentages indicate that in composition the chromite approaches the theoretical formula $FeO.Cr_2O_3$ and contains little or no ferric oxide or other substance capable of replacing the chromic oxide.

On Cypress Island orebodies have been found in several places in an area of 3 or 4 sq. mi. They are composed of grains and stringers of chromite scattered through the serpentine. Their boundaries are indefinite and their form irregular, though as a rule they show a trend of about N. 25° W. parallel to the main shear zones in the serpentine. One body yielded 40 tons of high-grade ore, little waste having been handled. Bodies that average from 10 to 25 per cent. chromic oxide range from a few tons to several hundred tons in size. A reserve of several thousand tons of ore of this kind is estimated.

In the Mt. Hawkins region a body of chromite about 18 in. (45 cm.) thick is exposed by an open cut north of Boulder Creek and ½ mi. east of the road up Cle Elum River. The deposit is well defined and free of waste, and a sample is said to have assayed 51 per cent. chromic

oxide. Another body of high-grade ore is reported near the small lake at the east of Mt. Hawkins. A shipment of ore from the deposits north of Boulder Creek, by Richard Denny, late in 1918 is reported. Both deposits are in the serpentine belt which occupies a large area south and west of Mount Stuart.

### *Alaska*

Deposits of chromite have been known in Alaska for a number of years and became of economic interest in 1917. Two deposits occur at the southwest end of the Kenai Peninsula—one along the north shore of Port Chatham and the other at Red Mountain, about 16 mi. (25 km.) to the northeast. Both deposits occur in bodies of altered peridotite. The ore is in lenses, none of which is more than 150 ft. (46 m.) in length, while most of them measure considerably less; they range in thickness up to 20 ft. (6 m.).

The deposits are being mined on the seashore, where the exclusion of the water increases the cost of mining. About 800 tons of ore containing from 46 to 49 per cent. of chromic oxide was mined in 1917, and in 1918 the output was about the same. There is much low-grade ore in the vicinity that can be concentrated to 50 per cent., and the Geological Survey reports (*Press Bulletin* 351) that material for 15,000 tons of such concentrates is available. The Red Mountain locality has not been worked. It lies 6 mi. (10 km.) over precipitous slopes from tidewater. One body probably contains nearly 1000 tons of high-grade ore, while the locality contains a much larger amount of low-grade ore. The freight rate on ore from Port Chatham to Seattle is $3.50 a ton, and thence by rail to an eastern smelter about $12.00 a ton.

### *Montana*

In southern Montana, between Red Lodge and Livingston, notable deposits of chromite have been discovered and recently attracted much attention. Those near Red Lodge are less important than those farther northwest, in the drainage of Boulder Creek, where there is a remarkable occurrence of chrome ore in a prominent dike of pyroxenite. The dike, running about N. 70° W. (Fig. 26), is commonly from ½ to 1 mi. (0.8 to 1.6 km.) in width and extends from Boulder Creek to Fish Creek, a distance of about 30 mi. (48 km.). It crosses plateaus and valleys cut in Archean or Paleozoic rocks to a depth of 4000 ft. (1219 m.) below the general plateau level. The pyroxenite is composed chiefly of enstatite, it is massive and fresh, with but little alteration to serpentine.

In the belt of pyroxenite the distribution of the body of chromite is like that of a dike intruded along the middle portion of the much larger dike of pyroxenite. Near this dike-shaped body of chrome ore are other smaller masses of chromite in parallel position. Westgate points out

that, although the great dike is mainly pyroxenite, the middle portion containing the chromite is rich in olivine, and the rock thus becomes peridotite or dunite. The chrome belt in the Bonanza claim has a thickness of about 90 ft. (27 m.), and its structure is broadly banded. The thickest band of chromite is 13 in. (33 cm.) and it lies near the middle of the belt.

Three properties were in active development during part of 1918. Considerable ore was mined (600 tons reported), but as the distance to Northern Pacific Railroad shipping points is about 35 mi. (56 km.), and the transportation facilities are not quite completed, there were no ship-

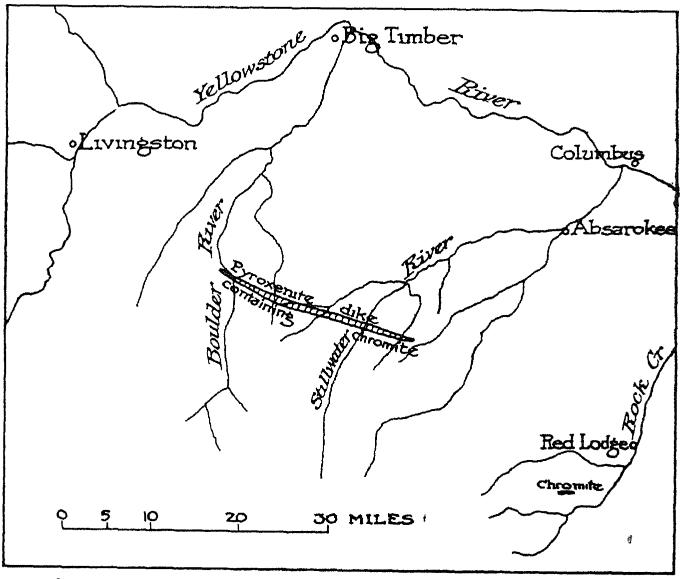


FIG. 26.—CHROME BELT OF BOULDER RIVER AND ROCK CREEK, MONTANA
(*L. G. Westgate* )

ments. In the Benbow mine the chromite occurs in a nearly vertical deposit from 3⅓ ft. to 5½ ft. (1.0 to 1.7 m.) in width. In places it is frozen to the wall rock, which may contain subordinate layers of chromite parallel to the main deposit. The depth of the ore deposit is not definitely known, but the exposures on the canyon walls suggests a depth of at least 700 ft. (213 m.) In length the deposit in the Benbow claim appears to be more or less continuous for 8000 ft. (2.4 km.), and the tonnage to a depth of 100 ft. is estimated at 250,000 tons.

The ore is black, granular chromite. Where the grains have not grown in contact with each other they are often octahedral, and it is possible to get specimens that are spotted with small but quite perfect crystals. According to Westgate, whose description of these deposits is being published by the U. S. Geological Survey, besides that in the orebodies, chromite occurs in grains and stringers in the country rock,

especially near the main bodies. In general, the orebody is dike-like in shape and lies near the middle of the pyroxenite dike. Smaller parallel bodies occur near the larger bodies, and a few smaller bodies occur near the borders of the pyroxenite dike, but they are not abundant near the contact between the pyroxenite dike and the adjoining Archean rocks, as has been noted by Lewis and others in North Carolina.

The chromite bodies are not true veins, but are believed by Westgate to be due to the forcing of a more basic portion of the magma into the partly cooled but still pasty pyroxenite. There are exceptional places where the orebody is more like a true vein; here the orebody is marked by sharp plane contacts on one or both sides, and often these surfaces are so smooth that they may be considered movement planes. The rock, both ore and enclosing pyroxenite, is here more crushed and serpentinized than is the rest of the belt, and the orebody is commonly wider.

The ore generally runs somewhat less than 40 per cent. chromic oxide and about 18 per cent. iron oxide. On the basis of 100 ft. depth, Westgate estimates the tonnage of the Benbow properties at 250,000 tons. He believes 375,000 tons is an underestimate for the belt as a whole, with 1,000,000 tons as possible. The chrome deposits of Stillwater and Sweet Grass Counties, although of too low grade for some purposes, and not readily concentratable to high-grade ore, are still of great importance.

### *Wyoming*

A chromite deposit is located on the left bank of Deer Creek in Natrona County, near the Converse County line, about 16 mi. (26 km.) southwest of Glenrock, the nearest railroad shipping point. It occurs on the steep slope of a canyon, about 400 ft. (122 m.) above the stream and 500 ft. (152 m.) below the gentler slope of the canyon rim, over which the road approaches.

At the time of my visit, Aug. 25, 1917, the road ended in a mass of serpentine and chromite lying between granite, toward the creek on the east, and hornblende schist on the west. The serpentine ranged from 20 to 30 ft. (6 to 9 m.) in thickness. It appeared to dip 45° or more to the west, and in the upper portion, adjoining the hornblende schist, contained a very irregular belt of chromite from 2 to 5 ft. (0.6 to 1.5 m.) wide, which was exposed at intervals for 150 ft. (45 m.). The best ore is fine grained to compact; at some places it is banded with serpentine, and much of the ore near the southern end is associated with micaceous minerals, in some places apparently interbanded with chromite and serpentine. The most common micaceous mineral is of a purplish pink color, and was determined by Mr. Schaller as chrome chlorite (kotschubeite or kämmererite); this forms veins and lines cavities in the ore. Several cavities contain a second lining of an earthy green mineral on the well

crystallized chrome chlorite; this appears to be wolchonskoite, which was recently identified by Schaller as a vein mineral in one of the mines near Ely, Nev.

The occurrence of these chrome silicates forming veins in chromite is interesting as showing the effect of hydrothermal action after the deposition of the chromite. Chrome garnet, as already noted, belongs in the same class, and its wide distribution with chrome chlorite in bodies of chrome ore on the Pacific coast strongly supports the view that chromite is of purely magmatic instead of hydrothermal origin.

The first ore was mined at this locality in 1908, and since then possibly as much as 700 tons has been taken out. In composition it ranges from 35 to 44 per cent. of chromic oxide with 16 to 20 per cent. FeO. Tests of the ore for furnace lining are reported to have shown that it is too easily fusible for that purpose. The property has recently been sold to the Ferro Alloy Co., of Denver, manufacturers of ferrochrome, who operate the mine to obtain ore for their plant at Pueblo. At the present time, it appears to be the only producing chrome mine in the United States.

*Pennsylvania and Maryland*

The chrome industry in the United States was started by Isaac Tyson, who discovered chromite in the Bare Hills, near Baltimore, in 1828, and the celebrated Wood mine,[37] of Lancaster County, Pa., was opened by him the same year. For many years it furnished the greater portion of the world's supply of chromite, with a total output of about 95,000 tons. The orebody was almost 300 ft. (91 m.) long, 10 to 35 ft. (3 to 11 m.) thick, and was proved to a depth of 720 ft. (219 m.). About 5 per cent. of the ore was crushed and washed; the remainder was pure enough to ship,[38] its average composition[39] being about 48 per cent. chromic oxide, although picked samples yielded as high as 56 per cent. Chrome sand on the surface near the Wood mine contained 46 per cent.

Other mines were developed in the same region and elsewhere, of which the State Line mine, on the border of Maryland, was one of the most important. In 1918 this region received much attention; some of the old Tyson mines were reopened, the Line mine was dewatered, and development had advanced into production when the war closed and operations ceased. The same occurred in the Soldiers Delight region, near Baltimore. Chrome-sand mining in that region will probably continue to furnish about 25 tons of the special high-priced stream-sand concentrates annually exported to Europe, where it is used to set the colors on fine porcelain ware.[40]

[37] William Glenn Chrome in the Southern Appalachian Region *Trans.* (1895) **25.**

[38] Persifor Frazer: Second Geol. Survey Pa., *Rept* CCC (1880) 192.

[39] E. C. Harder: U. S. Geol Survey *Mineral Resources*, Chromite (1908) 15.

[40] Joseph T. Singewald, Jr.: Maryland Sand Chrome Ore. *Econ. Geol.* (May, 1919) **14,** 189–197.

### *North Carolina*

Deposits of chromite in North Carolina were thoroughly investigated and described by J. H. Pratt and J. Volney Lewis as early as 1905.[41] The deposits occur in peridotite, with which corundum is associated, but the chromite and corundum appear to be mutually exclusive.

Pratt and Lewis found the deposits of chromite closely associated with the contact of the peridotite and gneiss. They attribute the origin of the chromite to crystallization from a magmatic condition; its segregation toward the border of the peridotite mass is regarded as due to gravitation and convectional movements induced by the cooling influence of the adjacent country rock.

Lewis classifies the deposits of chrome ore of North Carolina, according to origin and occurrence, under two heads, primary and secondary ores. The first he subdivides into (*a*) segregated lenses and pockets, (*b*) disseminated or partially segregated (banded) ore; the second into three groups, (*a*) float ore, (*b*) residual or lateritic ore, (*c*) alluvial or placer deposits. He remarks that "none of these occur in forms that may properly be called true veins."

The chrome ore is nearly uniform in general character throughout the entire area, being very hard and compact though often having a fine granular appearance. Masses of chromite are usually free from seams of peridotite or serpentine. Chromite pockets may be locally abundant, but they are not of large size; from one body 7 tons of ore was obtained, from another 12 tons, and a third yielded 25 tons. The ore in some places is high grade; selected samples contain as much as 63 per cent. chromic oxide; generally it is above 45 per cent., rarely as low as 39 per cent.

In 1918, operations were carried on at five places in three counties—Jackson, Buncombe, and Yancey—but the total output shipped was small, amounting to only a few hundred tons. Much of it was high-grade concentrate, containing 57 per cent. chromic oxide.

This interesting chrome field in North Carolina was reëxamined in the summer of 1918 by J. Volney Lewis, whose report will be published in the economic bulletin of the U. S. Geological Survey for 1918.

## Domestic Chromite Market

The effect of the war on the domestic chromite production is evident from Table 3. In 1913 the domestic product was only 255 long tons, and sold at $11.19 a ton. In 1918 the production had increased to 82,350 long tons, of which the average value was $47.82 per ton. Of the total amount mined in six years, 83 per cent. came from California, 16 per cent. from Oregon, and the remainder from Maryland, North Carolina, Washington, and Wyoming.

[41] North Carolina Geol. Survey (1905) 1.

TABLE 3.—*Marketed Output of Crude Chromite in the United States*

(Long tons[42])

| Year | California | Oregon | Wash., Wyo., Md., N. C. | United States,[43] Total | Average Price |
|---|---|---|---|---|---|
| 1913 | 255 | | | 255 | $11 19 |
| 1914 | 506 | | 85 | 591 | 14 75 |
| 1915 | 3,281 | | . . | 3,281 | 11 20 |
| 1916 | 43,758 | 3,099 | 178 | 47,035 | 15 44 |
| 1917 | 36,774 | 6,701 | 250 | 43,725 | 24 00 |
| 1918 | 63,064 | 18,455 | 831 | 82,350 | 47 82 |
| Total for 6 years | 147,638 | 28,255 | 1,344 | 177,237 | |
| Per cent. of whole | 83 | 16 | 1 | | |

Subject to revision. Brought up to date, April 24, 1919.

TABLE 4.—*Shipments, Stocks, and Values of Domestic Crude Chromite in* 1918

(Long tons)

| State | Shipments | Stocks on Hand, Dec 31, 1918 | Value of Ore Sold | Average Price |
|---|---|---|---|---|
| California | 63,064 | 34,510 | $3,060,585 02 | $48 53 |
| Oregon... | 18,455 | 6,833 | 855,049 88 | 46 33 |
| Md., N. C., Wash., Wyo. | 831 | 737 | 22,356 56 | 26 90 |
| Ga., Mont., Penn. | | 607 | | |
| Total, United States | 82,350 | 42,687 | $3,937,991 46 | $47.82 |

Subject to revision. Brought up to date, April 24, 1919.

TABLE 5.—*Equivalent Tonnages,* 1918 *Production*

| EQUIVALENTS | SHIPMENTS | STOCKS ON HAND, DEC. 31, 1918 |
|---|---|---|
| All grades—long tons, as above . . . | 82,350 | 42,687 |
| All grades—short tons . . . | 92,232 | 47,810 |
| 50 per cent. grade—long tons . | 66,554 | 34,043 |
| Chromium—long tons . . . . . . . . | 45,564 | 23,306 |

Of the above shipments of 82,350 long tons, 19,398 tons contained 45 per cent. or more of chromic oxide; 45,600 tons from 35 to 45 per cent.;

[42] Shipments of chrome ore by California and Oregon, Tables 1 and 2, are given in short tons according to railroad usage. In Table 3 the long ton is used for more convenient comparison with imports. The world's production is now generally given in metric tons.

[43] Pennsylvania, Georgia, and Montana have a mined production of 607 tons in 1918, but no shipments.

and 17,352 tons less than 35 per cent. chromic oxide; the average selling price at the railroad shipping station was $47.82 per long ton. The total shipments were therefore equivalent to 66,554 long tons of 50-per cent. grade, of which the value would have been $59.17 per long ton.

### Reserves

The market broke in August, 1918, when production was at its highest Much of the ore then on hand, as well as nearly all mined thereafter without contract, remained on hand at a loss to the miner. If to the tonnage of ore shipped we add the stocks on hand Dec. 31, 1918, the resulting 125,000 tons, the amount of ore mined in 1918, leads us to believe that the United States will be able to supply its own needs of chrome ore even during the severe demands of war. Recent investigations in North Carolina, Maryland, and Pennsylvania have not discovered large reserves, but in the West, especially in Montana, Oregon, and California, the reserves appear to be ample for several years' supply.

### Imports

According to the Bureau of Foreign and Domestic Commerce, the United States, in 1918, imported 100,224 long tons of crude ore. Imports from Canada, Cuba, and Brazil, 35,690 tons, ranged from 30 to 45 per cent. chromic oxide; the other 64,452 tons, coming chiefly from New Caledonia, contained 50 per cent. or more of chromic oxide; some ore from Guatemala contained 58 per cent. chromic oxide. The total of imports is equivalent to 92,681 long tons of 50-per cent. grade; its declared value being $2,892,825, the average price of 50-per cent. imported ore at the point of foreign ocean shipment would be $31.21 per ton, as compared with $59.17 per ton for the same grade of domestic ore on the Pacific Coast, at the point of railroad shipment.

As the total requirement of consumers in 1918 has lately been stated as not over 100,000 tons, it is evident that the imports more than equalled the requirements.

### Present Condition of Domestic Chromite Market

In 320 replies received by the U. S. Geological Survey to questionnaires concerning the production and shipments of chromite during the first quarter of 1919, 21 reported production. Of these, the six most important were:

The Ferro Alloy Co., operating in Wyoming

The Western Ores Co., concentrating in Butte County, Calif.; mining in San Luis Obispo County, Calif.

L. H. Butcher & Co.

S. L. Schwartz and P. A. H. Arata

The Neely Bros., in Trinity County, Calif.

All these report approximate shipments within the first three months of 1919, and were evidently supplying ore for their own use or for others under previous contract. There were 13 others reporting a production of about 800 tons and no shipments.

Of the 298 chromite correspondents reporting July 1, 1919, only one was at that time actually producing chromite. During the first half of 1919, 19,658 long tons of chrome ore, valued at $703,217 ($35.77 per ton in the country from which it came) was imported. The lower grade of the domestic ore and its reported greater cost to the consumer in the eastern part of the United States, as compared with the imported ore, is the reason why the eastern consumers prefer the latter, and there is no demand in our eastern market for domestic ore.

# Chrome-ore Deposits in Cuba*

BY ERNEST F BURCHARD,† M S, WASHINGTON, D C.

(Chicago Meeting, September, 1919)

A RECONNAISSANCE of the chrome and manganese[1] ore deposits of Cuba was made in the spring of 1918 by Albert Burch, representative of the U. S. Bureau of Mines, and the writer, representing the U. S. Geological Survey. The object of the study was to obtain authentic information for use of the United States Government as to the location, character, size, and availability of the Cuban deposits of the ores of these important steel-alloy metals. The Cuban Government courteously detailed Sr. E. I. Montoulieu, a mining engineer connected with the Treasury Department, to act as escort and associate throughout the work on the Island. Mr. George A. Wright, of Baracoa, Cuba, a mining engineer especially familiar with the chromite deposits of Oriente Province, was a member of the party from Feb. 25 to Mar. 20.

It was possible to visit most of the chrome-ore deposits that seemed to be of promise, but there was not sufficient time to visit all that were brought to our attention, some of which may have merit. Furthermore, no attempt was made to examine the extensive chromiferous iron-ore deposits of the Mayari, Moa, and Cubitas districts, as these deposits cannot be regarded as sources of chromium except indirectly.

At the close of the field work the essential data concerning the quantity and quality of ore probably available for shipment during 1918 and 1919, in the event of the continuation of the European war, and of the tonnage of ore in reserve, were furnished to such Government bureaus as were interested, including the Shipping Board and the War Industries Board, and as soon as chemical analyses of the chrome ores could be obtained, a press bulletin on the subject was published by the Geological Survey.

In the present paper the engineering data and tonnage estimates contained in the unpublished notes of Mr. Burch have been freely used, and their helpfulness is hereby gratefully acknowledged. Many thanks are due Senor Montoulieu and Mr. Wright for their valuable work in the field; to the officials of the Spanish American Iron Co. and of the various other companies; and to individuals holding chromite claims, for courteously facilitating the work of the party.

* Published by permission of the Director, U. S. Geological Survey.

† Geologist in Charge Section of Iron and Steel Metals, U. S Geological Survey

[1] See p. 51.

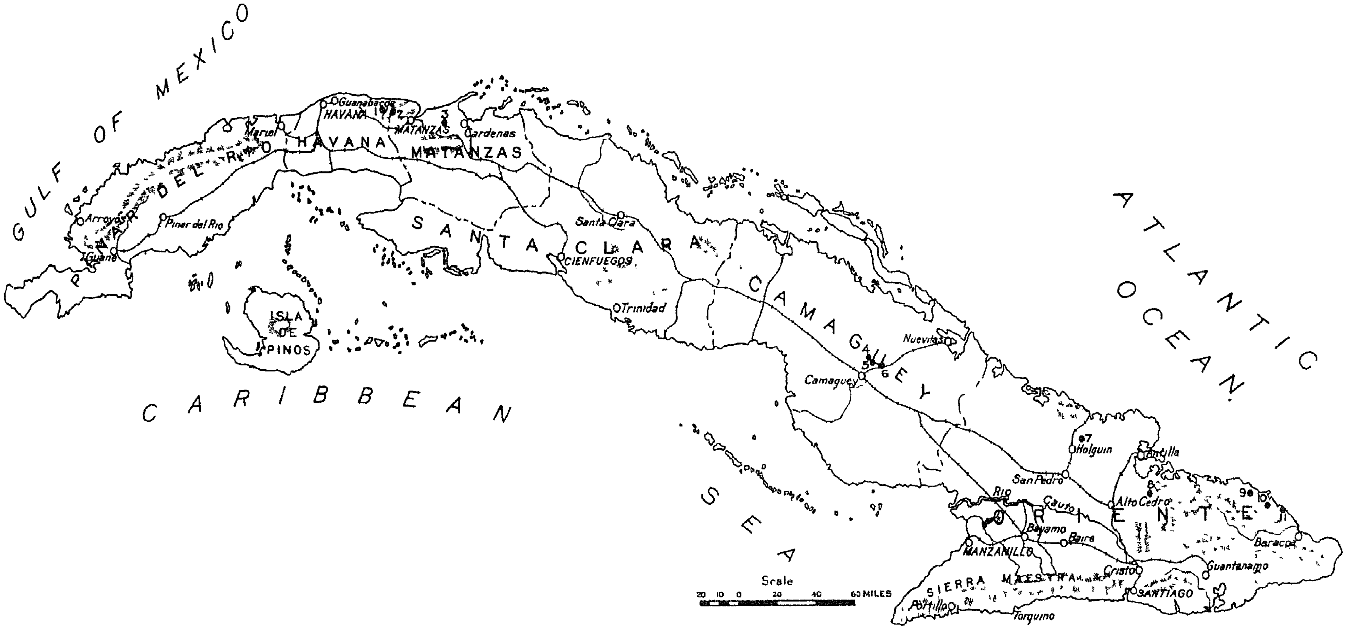


FIG. 1.—MAP OF CUBA SHOWING LOCATION OF PRINCIPAL DEPOSITS OF CHROME ORE. CLAIMS ARE INDICATED BY NUMBERS, AS FOLLOWS: .ELENA; 2, JACK; 3, ANA MARIA; 4, LEOCADIA AND NINAS; 5, NONA; 6, VICTORIA; 7, MARIA DEL CARMEN; 8, CALEDONIA; 9, CAYOGUAN, NARCISO AND CROMITA; 10, POTOSI; 11, CONSTANCIA.

In the study of material for the present paper, chemical analyses have been made by R. C. Wells and petrographic determinations and descriptions were furnished by Miss E. F. Bliss, both of the U. S. Geological Survey.

## Distribution and Nature of Deposits

About twelve groups of chrome-ore deposits in Cuba have thus far attracted attention, all being within 25 mi. (40 km.) and most of them within 10 mi. (16 km.) of the north coast; they display considerable diversity in quality, size, and accessibility. One of the most westerly deposits is in the eastern part of the Province of Habana, and two are in the Province of Matanzas. The next group toward the east is in the Province of Camaguey, a few miles northeast of the city of Camaguey. Other deposits are in the Province of Oriente, one near Holguin and another south of Nipe Bay, and there are three groups in the mountains near the coast between Punta Gorda and Baracoa. (See Figs. 1, 3, and 6.)

## Geologic and Topographic Features

Areas of serpentinized basic rocks are exposed in all the provinces of Cuba, generally parallel to and within a few miles of the north coast. Within these serpentine bodies are segregations of chromite. Small bunches of radial fibrous crystals, probably tremolite, commonly occur in the serpentine near the orebodies. The deposits are lenticular or tabular masses ranging in thickness from 1 ft. to more than 50 ft. (0.3 to more than 15 m.), and reaching a maximum length of more than 200 ft. (61 m.), but they may include small masses of serpentinized peridotite. Thin veinlike seams of ore have also been noted. The boundaries of the orebodies are fairly sharp, especially along their flatter contours, but chromite grains may be discerned with the unaided eye in the serpentine in places several inches from the main mass. Stringers of chromite branch out, especially near the ends of the bodies. The orebodies appear to be characteristic magmatic segregation deposits and resemble many of those found in areas of serpentine in California and Oregon.

Most of the mineralized masses are highly inclined and certain of them, which are exposed in ravines on steep hillsides in mountainous or hilly regions, dip at about the same angle as the hillsides. The deposits west of Nipe Bay are in areas of moderate relief; those near Camaguey are in an area of very low relief. The deposits in the eastern part of Oriente Province, which are the largest, are in mountainous country and are difficult of access.

The serpentine areas are characterized by thin, poor soils, lacking in potash, and support only a scant vegetable growth, consisting chiefly of low scrub palm, coarse grasses, and thorny weeds.

*Character of Ore.*—The ore is generally fine grained to medium-coarse grained, and varies from spotted material, consisting of black grains of chromite, ranging in diameter from 1/30 to 1/4 in. (1 to 6 mm.) embedded in white or light-green serpentine, to compact black granular material containing little or no visible serpentine. (Figs. 4, 5, and 7.) Fine seams of a green crystalline mineral, uvarovite, a calcium-chromium garnet, are occasionally found in the ore.

As a rule, the grade of the ore is not very high. Excellent specimens may be secured at almost all deposits and may be picked up in the float, but it would require highly selective mining at most places to secure a product that would average 45 per cent. chromic oxide.

The mineral chromite ($FeO.Cr_2O_3$), free from impurities, contains, theoretically, 46.66 per cent. chromium, or 68 per cent. chromic oxide. Representative samples from 18 deposits gave chromic oxide from 26 to 43 per cent.; iron from 10 to 13 per cent.; silica from 1 to 12.5 per cent.; alumina from 15 to 33 per cent.; traces, only, of sulfur, phosphorus, and nickel were found in some of the samples, but others contained none. The majority of the samples carried between 33 and 40 per cent. of chromic oxide (representing between 22 and 27 per cent. of chromium) and only two carried more than 40 per cent. chromic oxide. Ores of certain other deposits, reliable analyses of which are available, are of better grade, some of them showing as high as 48 per cent. chromic oxide and as little as 8 per cent. of iron. In most of the ores of which analyses are available the percentage of chromium is two or more times that of the iron, a relation of importance in connection with the manufacture of ferrochrome in the electric furnace.[2]

## Description of Deposits

### *Habana Province*

*Elena.*—The Elena claim, 2 mi. (3.2 km.) south of Canasi, near the eastern border of Habana Province, is owned by the Elena Mining Co., of Habana. The properties are reached from the railroad station of Jaruco by auto to Jibacoa about 12 mi. (19 km.), and thence by foot or horse 4 mi. (6.4 km.) over a hilly road. Shipments of ore can, however, be made by cart to Canasi, about 3 mi. (4.8 km.), and thence by boat to Habana; and another outlet may be afforded later by the Hershey Cuban Railroad to Santa Cruz del Norte, the route of which is reported to pass within 1 km. of the properties.

Serpentine here forms a chain of hills parallel to the coast and about 500 ft. (152 m.) in altitude. The serpentine is much fractured and slick-

[2] R. M. Keeney: Manufacture of Ferro-alloys in the Electric Furnace. *Trans.* (1920) **62**, 28.

ensided and is intersected in places by stringers of darker rock, probably peridotite that has not been greatly altered. Float pebbles of chromite are found on the slopes and in the water courses, and have led to the opening of several small lenticular bodies of chromite 5 to 10 ft. (1.5 to 3 m.) wide, with stringers of the mineral branching from them.

The workings consist of a cut 20 ft. (6 m.) deep, in a hillside terminating in a short tunnel which leads to a shaft. A tunnel had also been started toward the shaft at a point somewhat lower and several hundred feet eastward but at the time of visit had not reached the shaft, nor intersected any orebodies. Other openings have been made on the surface without disclosing any ore. The pockets of ore exposed by the open cut had been entirely removed except small stringers in the east side and south end. About 600 tons of ore of commercial grade were reported to have been obtained and shipped to Baltimore, Md.

Although conditions are favorable for the occurrence of more small orebodies in this vicinity, the cost of prospecting is so great, compared with the value of the ore that might be discovered, it would seem that only through accidental discovery is more chrome ore likely to be uncovered.

### *Matanzas Province*

*Jack.*—The Jack chrome-ore claim of the Compania Oriental de Minas is situated in the western part of Matanzas Province about 7 mi. (11 km.) northwest of the railroad station of Mocha, and 15 mi. (25 km.) by wagon road west of the City of Matanzas. The road to the railroad is rather rough and the cost of haulage of ore was estimated at $5 per ton, in the spring of 1918.

The country is hilly and rocky and locally it is underlain by the same belt of serpentine exposed at the Elena. The surface is sparsely covered by vegetation, characteristic of serpentine areas, so that outcrops of chrome ore and float are readily observed. A mile or more toward the south and east lie high ridges of limestone bordering the serpentine.

Five or six small orebodies have been opened on this property, all of which showed some trace on the surface, and considerable development work had been done, amounting in all to at least 30 openings, including tunnels, one of them 200 ft. (60 m.) long, several prospect cuts, shafts, and drifts.

The serpentine is much fractured, and slickensides are common. Stringers and larger masses of peridotite (?) containing brighter specks of bastite are cut in all the openings. The chromite occurs in stringers and lenses in the serpentine and peridotite, but the orebodies are not generally connected with one another. The two largest lenses were 35 to 50 ft. (11 to 15 m.) long, 13 to 20 ft. (4 to 6 m.) deep, and 1 to 3 ft.

(0.33 to 1 m.) wide. Asbestiform and talcose minerals have been developed in the serpentine near the chromite.

The chrome ore is fine to medium grained and is generally fairly compact and of good appearance, but it also contains masses of spotted ore composed of medium-sized grains of chromite in a gangue of altered serpentine. Some of the ore on weathering shows a faint brownish tinge, probably due to iron oxide.

Nearly all the chrome ore disclosed by the workings had been removed and was stored in two stock piles which at the time of visit appeared to contain about 440 tons of ore. Analysis of a composite sample of ore from the two bins gave the results in column 1, Table 1, while analyses *A* and *B* represent samples obtained by trenching separately the two bins of ore.

TABLE 1.—*Analyses of Chrome Ore from Stock at Jack Mine, Near Matanzas*

| | 1 | *A* | *B* | | 1 | *A* | *B* |
|---|---|---|---|---|---|---|---|
| $Cr_2O_3$ | 43.0 | 36 52 | 35.84 | S | Trace | | |
| Fe... | 13 0 | 12.3 | 11.85 | P | None | 0 04 | 0 03 |
| $SiO_2$ | 5.4 | 6.66 | 6.73 | Ni | Present | | |
| $Al_2O_3$ | 15 0 | | | | | | |

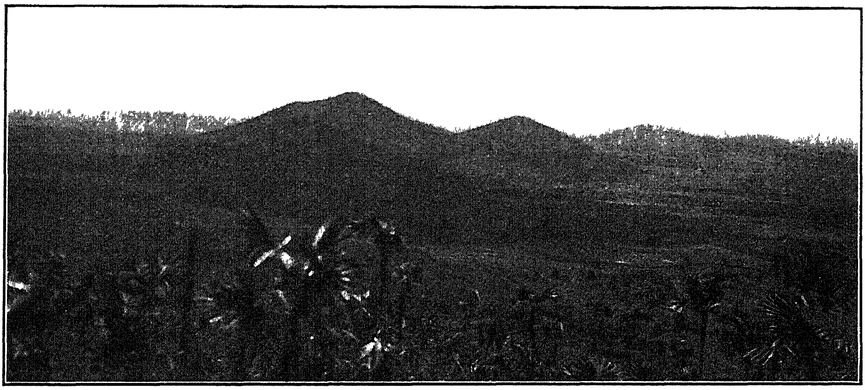

FIG. 2.—TYPICAL SERPENTINE HILLS IN VICINITY OF ANA MARIA CHROME-ORE CLAIM, BETWEEN CARDENAS AND MATANZAS.

*Ana Maria.*—The Ana Maria claims, 17 mi. (27 km.) by wagon road west of Cardenas, were held by Jorge Faz, of Cardenas. These claims are about 2 mi. (3.2 km.) from a sugar plantation railroad and about 1.25 mi. (2 km.) north of the Matanzas-Cardenas highway. They are also about 9 mi. (15 km.) south of a small cove on the coast, said to have 21 ft. (6.4 m.) of water. These claims are situated on a chain of cone-

shaped hills known as the Cordillera de Guamacara, which rise 650 to 750 ft. (198 to 229 m.) above sea level and 500 to 600 ft. (153 to 183 m.) above the neighboring valley (Fig. 2).

Chrome ore shows as float on the north slope and on the lower spur to the northwest of the hill known as La Coaba, and a few outcrops have been found. The country rock is of much broken serpentine and the chrome-bearing zones follow a general east-west direction. The lower one, at an altitude of about 385 ft. (117 m.) had been opened by a shallow trench about 100 ft. (30 m.) long, showing chromite in several small seams mixed with serpentine. The ore shows in places on both sides of the trench, and at the middle of the opening was a pillar from which good lump ore might have been hand-cobbed. The ore is fine grained and of high grade in spots, but is seamed and specked with serpentine to a considerable extent; thus, to obtain a high-grade product will require concentration, and this might be difficult on account of the fineness of grain. The chrome-bearing material is sparsely and unevenly distributed and constitutes only about 10 per cent. of the rock as disclosed in the cut. Two small piles of milling ore, about 2 tons in all, had been taken out here. On the north slope, 35 ft. below the top of the neighboring hill, La Coaba, at an altitude of about 630 ft. (192 m.), five or six small cuts had been opened, the largest of them 30 ft. long, 8 to 10 ft. deep, and 12 ft. wide; this cut showed a seam 1 to 5 ft. (0.3 to 1 5 m.) wide of fine-grained, speckled ore somewhat mixed with rock. The seam dips steeply toward the north and its depth has not been determined. Three other cuts showed a small seam of ore-bearing material. Much high-grade float chromite was observed on this hill below the cuts and a few fragments above them. Some deeply weathered lumps showed stains of iron oxide. Picked samples of the ore are reported to have analyzed 52 per cent. of chromic oxide. About 50 tons of ore was in sight in the two large cuts at the time of visit.

Other chrome claims reported to have been denounced in this vicinity, on the basis of discovery of float ore, are the Louisiana, in the Camarioca range of hills about 1 mi. east-southeast of the Ana Maria, and the Sultana, on Maceo hill about 1½ mi. (2.4 km.) to the west.

*San Miguel.*—Since the visit of the writer, Senor Montoulieu reports the occurrence of a chrome-ore deposit in Matanzas Province near the sulfur springs of San Miguel de los Baños, north of Coliseo, some 50 miles (80 km.) northeast of Matanzas, in the same range of coastal hills on which the Ana Maria deposits near Cardenas are located, but farther inland.

Large amounts of fine-grained chromite float led to the discovery of an ore outcrop. The deposit is reported as a large lens over 20 ft. (6 m.) thick, lying almost horizontal. Good shipping facilities were available through the completion of the new macadam road from Coliseo to the

sulfur baths at San Miguel. The deposit was leased to the Brittania Mining Co. of Havana, which commenced active mining in the summer of 1918, and it is reported that an output of 30 tons a day was reached during the last days of the war. The total shipments are understood to have been over 700 tons of chrome containing over 42 per cent. of chromic oxide.

*Camaguey Province*

The deposits of chrome ore examined in Camaguey Province consist of three groups which lie along a narrow zone beginning about 9 mi. (14.5 km.) northeast of the city of Camaguey and extending south-

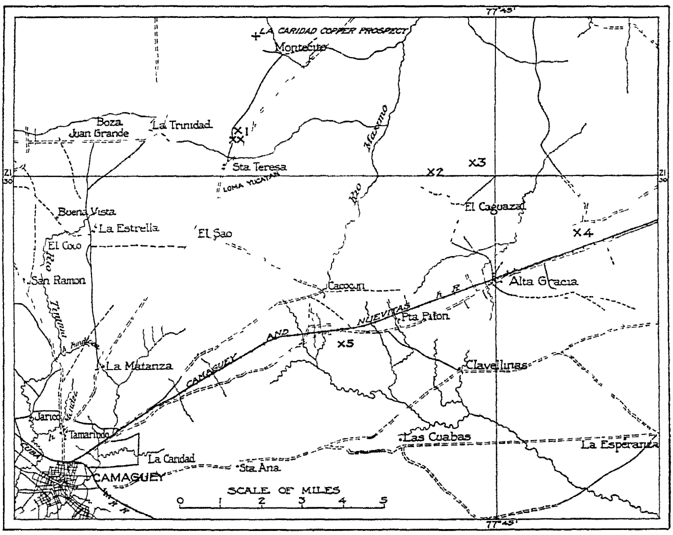


FIG. 3.—MAP SHOWING LOCATION OF CHROME-ORE DEPOSITS IN VICINITY OF CAMAGUEY. CLAIMS AND ORE LOCALITIES ARE INDICATED BY NUMBERS, AS FOLLOWS: 1, LEOCADIA AND NINAS; 2, NONA; 3, FLOAT CHROME ORE, IGUANA HILLS; 4, VICTORIA, 5, FLOAT CHROME ORE.

eastward to a point near the Camaguey & Nuevitas Railroad, 2 mi. (3.2 km.) northeast of Alta Gracia station (Fig. 3). Immediately north of Camaguey is a nearly level plain developed on serpentine rock, which extends northward for several miles and is covered with a thin mantle of ferruginous clay and limonite gravel residual from the decay of the serpentine. The chrome-ore deposits lie along the border of this plain at its juncture with some low hills which are outliers of a plateau south of the Cubitas Mountains; hence they are easily accessible by wagon roads. The local rock is serpentine with masses of gabbro outcropping on the hills. Float fragments of chromite are found in places and

broken ore caps 10 or 12 small mounds which rise 5 to 50 ft. (1.5 to 15 m.) above the surrounding surface. In this zone there are also about 15 places where chromite float is so thick as to suggest that small bodies of ore in place are obscured by broken ore or rock débris. Several claims have been denounced here, among which are the Teire, Leocadia, and Ninas, near the west end of the zone, the Nona, near the middle of the zone, and the Victoria the easternmost.

*Leocadia, Ninas, and Teire.*—Denouncements bearing the first two names are reported to be holdings of the Cuban Mining Co., an American organization; the third is held by Camaguey parties. The claims are situated about 9 mi. (14.5 km.) northeast of Camaguey and 2 mi. (3.2 km) south of Montecito, near where the Caridad copper prospect has been opened. The deposits comprised within these claims are distributed over an area extending about ½ mi. (0 8 km.) south to north and about 1200 ft. (366 m.) from east to west at the widest point. There are here 8 or 10 outcrops of chrome-bearing rock in as many small mounds which rise 5 to 20 ft. (1.5 to 6.1 m.) above the surface of the serpentine plain, having axes which strike generally about N. 20° W. These mounds range from 50 to 100 ft. (15 to 30 m.) in length, and represent the loci of lenticular bodies of chromite. They are covered with broken boulders and fragments of the mineral, but show none in place. Float chromite extends several hundred yards from these mounds and can be seen along the Camaguey-Limones-Gloria wagon road about ¼ mi. (0.4 km.) southeast of the group. All the ore that can be considered as actually available here, unless prospecting should disclose bodies in place below the capping of broken material, is in loose fragments ranging in size from pebbles to boulders 2 ft. (0.6 m.) in diameter. The quantity of broken ore on the mounds ranges from 50 tons in the smaller ones to about 2000 tons in the largest. In addition, there are several thousand tons of ore in the form of loose float scattered over an area of about 40 acres (16 ha.)

The ore is not of uniform grade, being mostly of only fair grade although containing some high-grade lumps; some portions contain streaks of barren rock The grain is fine to medium, and much of the material is spotted, consisting of grains of chromite imbedded in a gangue of light green to white serpentine. Three representative samples of the ore (analyses 2, 3, 4, Table 2) show not quite 34 per cent. of chromic oxide, about 12 per cent. iron, 4 per cent. silica, and 30 per cent. alumina, whence it is evident that all the ore would have to be concentrated to the marketable grade of 40 per cent. chromic oxide.

Prospecting here, while yielding samples for analysis, has not shown the nature or extent of the deposits in place. Surface indications warrant more thorough prospecting, and should a large quantity of ore be proved there would be little difficulty in putting the road to Camaguey into suitable condition for motor trucks.

*Nona and Neighboring Deposits.*—The Nona claim includes chrome-ore deposits on a small hill and on the neighboring plain about 12 mi. (19 km.) northeast of Camaguey, and about 5 mi. (8 km.) east of the Leocadia group; it was reported to belong to the Cuban Mining Co. It is not so accessible from Camaguey as the deposits first described, but it was possible to reach it in a Ford car. The distance to the Camaguey & Nuevitas Railroad, at Alta Gracia, is about 4 mi. (6.4 km.) over a nearly level surface, and if a large quantity of ore were found here a cart road might easily be built to that point.

Ten to twelve small patches of broken chrome-bearing rock were noted here, extending a distance of about 200 ft (61 m.) northwest-southeast, the small hill being at the northwest extremity. These areas range in length from 10 to 25 ft. (3 to 7.6 m.), and in width from 2 to 10 ft. (0.6 to 3 m.). Like the other deposits in this district, the outcrops are obscured by broken ore and float and little can be ascertained about the actual dimensions or attitudes of the orebodies without making excavations. The ore is fine to medium grained, rather dull and lusterless on the outcrop, and does not appear to be of as good quality as that at the Leocadia claim, but analyses of three samples (No. 5, 6, 7, Table 2) showed 35 to 36 per cent. chromic oxide, which is slightly better than the Leocadia ore.

In the broad flat valley east of the Nona deposits there is considerable chromite float, up to 3 or 4 in. (76 to 102 mm.) in diameter, and also some chromite sand. The sand is derived from fine-grained, light-specked chromite-bearing seams ½ to 2 in. (13 to 51 mm.) thick, which outcrop in the serpentine floor of the valley, but the pebble float was traced to two sources near the Iguana hills, bordering the valley on the northeast. These deposits are respectively about ½ and 1 mi. (0 8 and 1.6 km.) northeast of the Nona deposits, and both of them show only as broken ore partly buried by débris of gabbro from the adjacent hills. The deposit farthest northwest is apparently small and unimportant, but the other is evidently of considerable length, since there is a line of float fragments for a distance of about 600 ft. (183 m.) along the southwest slope of the ridge. No estimate of tonnages was possible, as nothing could be seen of the orebody.

The ore is of fine to medium grain and appears to contain considerable impurity. Samples from both of these areas indicate a low grade of ore, carrying less than 30 per cent. of chromic oxide and more than 30 per cent. of alumina (No. 8, 9, Table 2). Still farther up this valley toward the southeast, and also south of the Camaguey & Nuevitas Railroad, abundant chromite float occurs in places, but probably most of this has originated in many small unimportant bodies of ore that have been completely broken down and eroded from their inclosing rocks.

*Victoria.*—The Victoria chromite claim, said to belong to the Cuban

Mining Co., is situated about 14 mi. (22.5 km.) east-northeast of Camaguey, and about 2 mi. (3.2 km.) northeast of the Alta Gracia station on the Camaguey & Nuevitas Railroad. It is only about ½ mi. (0.8 km.) in a direct line from the railroad. A good wagon road to Alta Gracia passes within ¼ mi. (0.4 km.) of the property, and a wagon road or spur track could without difficulty be built over nearly level ground from the nearest point on the railroad.

Chrome ore in boulders and lumps caps the flat top of a small mound which rises about 50 ft. (15 m.) above the surrounding plain. The approximate length along the top of this mound is 150 ft. (46 m.) and the width is about 70 ft. (21 m.). The greater part of the top of the area appears to be covered to a thickness of at least 5 ft. (1.5 m.) by chromite débris in lumps up to 2 ft. (0.6 m.) in diameter. The slopes are in part covered by 1 ft. or more of chromite débris, and float ore is distributed over the surrounding 20 acres (8 ha.) of gently rising plain.

There are no definite outcrops of ore in place. From the general distribution of the broken ore, however, it is suspected that the hill contains two or more steeply inclined thick ledges of chrome ore 150 ft. (46 m.) or more in length, and to the superior resistance of this chromite to erosion is due the existence of the present mound. The chromite is associated with masses of olivine gabbro, large boulders of which lie on the plain east of the mound. This gabbro is a medium to coarse grained, even textured, mottled green and white rock consisting essentially of 65 per cent. plagioclase and 35 per cent. olivine.

The ore is mostly coarsely granular spotted material, but some is fine grained. Some of the coarse-grained ore resembles the gabbro in texture, the chromite grains composing about the same proportion of the rock as the olivine in the gabbro. Two samples were taken from representative types of material, and both were found to contain about 34 per cent. of chromic oxide (No. 10, 11, Table 2). Probably all of the ore would need mechanical concentration; its texture appears favorable for this and a supply of water can be obtained from a small stream, called the Santa Cruz River, about ½ mi. (0.8 km.) distant.

A thin section of the ore is described by E. F. Bliss, as follows:

The rounded chromite areas are not massed in aggregates but are surrounded by an interstitial serpentine which is plainly seen to be derived from olivine

It would appear that the original rock had been a dunite in which olivine formed the interstitial material around a crystallization of chromite. The serpentine that fills the core of the olivines is surrounded by a narrow rim of material that is probably recrystallized serpentine and perhaps later than the olivine alteration product. Iron oxide is notably scarce in this section. There is a slight development, with the serpentinized olivine, of a cloudy white mineral similar to that in the Constancia deposit.

*Other Claims.*—In addition to the above described deposits, two large bodies of good chrome ore, claimed by the Cia. Nacional Minera, of

Havana, were examined by Senor Montoulieu in October, 1918. One of them was located about a mile west of the Nona claim near Alta Gracia, and the other about 300 meters south of a switch of the Senado Sugar Co., about two miles west of the town of Minas. Some samples taken at random assayed 38 per cent. $Cr_2O_3$, and one is reported to have been assayed a few years ago by a Mr. Dewey in the employ of the U. S. Government which exceeded 45 per cent. of $Cr_2O_3$ About 5000 tons of ore may be available in these two deposits.

*Résumé of Camaguey District.*—At the time of visit, no attempt had been made to mine or ship any chrome ore from the deposits near Camaguey, so far as could be ascertained. Rough estimates of the total quantity of surface ore in the form of broken blocks and coarse float indicate that there may have been 20,000 tons available, and if the deposits have not been completely eroded there may be nearly as much ore below the surface. The float ore, however, should be of a better average grade than the ore in place because the more compact ore disintegrates more slowly than that of lower grade, and travels farther in the form of lumps.

Ten samples of ore were found to contain 27 to 36 per cent. of chromic oxide (Table 2); only two of these samples contained less than 30 per cent. and few contained more than 35 per cent. The ore in these deposits is therefore of low grade, but it may be suitable for certain purposes. If it should require concentration, sufficient water is believed to be available in small streams within a mile of the deposits.

TABLE 2.—*Analyses* of Chrome Ore from Deposits near Camaguey*

| | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 |
|---|---|---|---|---|---|---|---|---|---|---|
| $Cr_2O_3$ | 33.7 | 33.7 | 33.8 | 35.2 | 36.3 | 35.0 | 27.4 | 29.1 | 34.2 | 34.1 |
| Fe | 12.2 | 12.3 | 10.9 | 11.8 | 10.6 | 11.6 | 10.7 | 11.4 | 11.1 | 11.0 |
| $SiO_2$ | 3.9 | 4.3 | 4.1 | 3.9 | 3.6 | 4.5 | 4.0 | 2.4 | 1.5 | 1.5 |
| $Al_2O_3$ | 29.8 | 30.7 | 27.0 | 27.4 | 26.2 | 26.7 | 30.2 | 32.9 | 28.3 | 28.7 |
| S | | | | | | | | | None | |
| P. | | | | | | | | | Trace | |
| Ni | | | | | | | | | ±0.05 | |

*Analyzed by R. C. Wells.

Samples 2–4 from the Teire, Leocadia, and Ninas claims, north-northeast of Camaguey; 5–9 from the Nona and an unnamed deposit, northeast of Camaguey; 10–11 from the Victoria claim, northeast of Alta Gracia.

As these deposits can be reached by wagon roads that are already in existence, or might be laid out over nearly level ground, they are of interest notwithstanding the low grade of the ore.

Besides the chrome-ore deposits we examined in the vicinity of Camaguey, others are situated about 20 mi. (32 km.) north of Camaguey and just north of the east end of the Cubitas field of surficial brown iron

ore. In this locality claims known as the Cid, Teyde, and Yunque, which were examined in 1907, by A. C. Spencer, of the U. S. Geological Survey, all show noteworthy quantities of chrome float, apparently of good grade, and the occurrence of tabular bodies of ore from 1 to 5 ft. (0.3 to 1.5 m.) wide is indicated. On the Cid claim, boulders of ore are distributed over a belt about 1700 ft. (518 m.) long, and on the Yunque the ore fragments are found in an area 150 by 250 ft. (46 by 76 m.). On the Teyde, five separate deposits lie within an area measuring 1200 by 3000 ft. (366 by 914 m.). These deposits, one of which seems to be continuous for 900 ft. (274 m.), strike N. 10° to 30° E.

### *Oriente Province*

*Maria del Carmen.*—These claims, owned by the Compania Nacional Minera, of Habana, are situated 7 to 8 mi. (11 to 13 km.) northeast of Holguin. The deposits extend for a distance of a little more than a mile (1.6 km.) from southwest to northeast and are reached from Holguin by a cart road about 8 mi. (13 km.) in length, rocky and hilly in places and fairly good in others.

Several small bodies of chromite outcrop on the northwest slope of a low ridge of serpentine which lies between two higher ridges of steeply inclined limestone standing about ½ mi. distant to the northwest and southeast. The serpentine in this locality is generally rather hard and brittle, light-colored, and fresher looking than much of the rock farther east in Oriente Province, and both the rock and the ore resemble those of certain deposits in California. The altitude of the serpentine ridge is about 500 ft. (152 m.) above sea level and it rises about 200 ft. (61 m.) above the nearby streams. Float pebbles of chrome ore are found on the slope below the pockets of ore, and in other places where no ore is known to be *in situ,* and much of this float has been gathered for shipment. Some of the specimens noted here were of compact, medium grained, material of very good quality, while others were mixtures of chromite and gangue minerals. Analysis of a sample of ore showed 30.6 per cent. chromic oxide. Several tests of ore reported by the company engineer averaged 34.37 per cent. chromic oxide, and later reports have been received to the effect that a body of ore carrying 40 per cent. was discovered in the summer of 1918.

At the time of visit the developments on this property consisted of an irregular open cut about 30 ft. long and 15 ft. deep, made about 40 years ago; a short cut 6 ft. deep about 150 ft. toward the northeast; and a short prospect trench about ¾ mi. northeast from the old pit. From the old pit had been taken about 150 tons of chrome ore of good appearance; when mined, this was mistaken for iron ore, but after being tested none was ever shipped. A body of ore 2 ft. wide crosses the neighboring open

cut from southwest to northeast, and the distant cut shows a 3-ft. layer of fair-grade material.

The ore indications observed at this locality do not warrant any estimate of ore in reserve. Geologic conditions are similar to those near Matanzas, where slightly larger bodies of chrome ore have been discovered. About 175 tons of ore were available for shipment in the spring of 1918.

*Caledonia.*—One of the larger deposits of chrome ore in Oriente Province is on the Caledonia claim, near the base of the south slope of the Sierra de Nipe, about 7 mi. (11 km.) southeast of Woodfred, the headquarters of the Mayari iron-ore mines. This claim is held by the Spanish American Iron Co. and has been sufficiently prospected to block out fairly well the outlines of the deposit. A zigzag trail of easy grade, descending 1000 ft. (305 m.) in 3½ mi. (5.6 km.), has been cut along a spur from the plateau down to the deposit, and at the time of visit ore could be carried out by mule pack, loaded into auto trucks at the top of the trail, carried 4½ mi. (7.2 km.) to the railroad at the Mayari mines, and thence by rail to Nipe Bay. Routes had been surveyed and cleared for a 6000-ft. (1.8-km.) aerial tramway which would take the place of mule transportation.

The upper part of the orebody crops out of serpentine on a steep hillside southeast of and about 300 ft. (91 m.) above a mountain stream which flows into the Pinos, a small tributary of Mayari River. The country rock along the stream below the chrome-ore deposit is generally dark green in color, and dense and dull in appearance except where crystals of bastite are conspicuous. A thin section of this rock is described by E. F. Bliss, as follows:

> Serpentine derived from a peridotite, probably wehrlite variety. A medium-grained dark-green rock composed of serpentine, in part a dense variety and in part a fibrous bastite The original rock appears to have been largely olivine with considerable pyroxene, probably a mixture of diallage and bronzite. The pyroxene, which can be recognized by the characteristic cleavage lines, has now altered to the brown, fibrous variety of serpentine known as bastite Alteration in the olivine seems to have proceeded along the characteristic network of cracks. The original rock does not appear to have contained much iron and iron oxide is not abundant In some places it has been liberated in the central portions between the cracks, while in other places it follows the line of cracks. In a few places the original olivine core still remains

At the upper outcrop of the orebody the rock is light or dark green to brownish, dense, fine-grained material of duller appearance than the rock lower down. A thin section is described by E. F. Bliss, as follows:

> It is very similar in character to the rock above described, composed chiefly of serpentine altered from olivine and of a fibrous bastite derived from an original pyroxene. A light green variety of serpentine fills cracks which cut across the original constituents of the rock. Iron oxide occurs both in the core of the olivine crystals between the cracks and along the cracks.

Within tunnel No. 2 barren rock is found in places separating portions of the orebody. In the hand sample this material is a dense, dark-green rock, partly covered with a coating of light-green, bluish, and white minerals. In thin section it exhibits the following characteristics:

This rock is very similar to the two specimens just described but it shows less evidence of original pyroxene. The characteristic reticulate texture of olivine is evident even in the hand specimen on examination with a lens

In places the original olivine core remains, but the alteration has generally been complete, liberating iron oxide in small amounts in the core, which is composed of serpentine Iron oxide is also concentrated to some extent along the cracks.

The bluish-white coating on the exterior of this specimen seems to be due to the development of a soft, cloudy white aggregate probably of similar composition to the white interstitial and coating material occurring with the ore at the Constancia claim

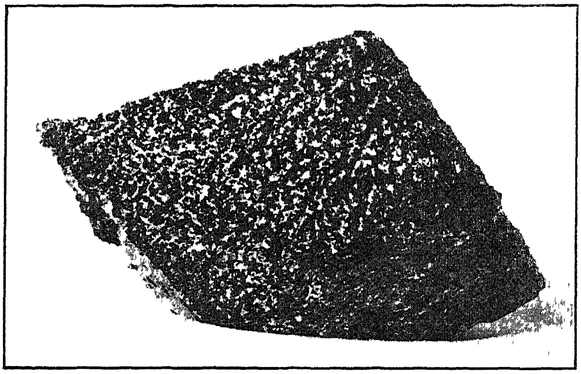

FIG. 4.—MEDIUM-GRAINED SPOTTED CHROME ORE FROM CALEDONIA CLAIM. SHOWS SLICKENSIDES AT LOWER LEFT EDGE. 0.55 NATURAL SIZE.

The ore consists mainly of moderately fine grains of chromite, ranging in diameter from $\frac{1}{50}$ to $\frac{1}{20}$ in. (0.5 to 1.25 mm.) or in aggregates of grains, mixed with more or less serpentine gangue. Some of it is compact chromite showing little serpentine and some is distinctly specked with yellowish green grains of serpentine (Fig. 4). At the mouth of tunnel No. 1 is a coarsely spotted ore consisting of rounded chromite grains segregated into bunches 0.1 to 0.4 in. (2.5 to 10 mm.) in diameter, separated by light-green serpentine (Fig. 5). The ore is closely jointed in many directions and in many places the ore fragments show slickensided surfaces similar to those of the serpentine, indicating that the ore existed prior to the fracturing of the rock.

The orebody is roughly tabular in form, and 10 to 30 ft. (3 to 9.1 m.) thick. It dips toward the northwest at about the slope of the hillside (40° to 45°) and where it does not crop out it lies 30 to 50 ft. (9 to 15 m.) within the hill. Two tunnels cut the ore at levels 100 ft. and 200 ft. below

the top of the outcrop. The orebody lies so close to the surface of the hillside that it might possibly be mined as an open cut.

The ore varies in quality, the better grade being in the western part of the deposit, where it is reported to carry as high as 48 per cent. of chromic oxide. Analyses furnished by the Spanish American Iron Co. show a range of 36 to 48 per cent. of chromic oxide, 7 to 15 per cent. of silica, and 8 to 15 per cent. of iron for the whole body. By cobbing or by simple water concentration it might be possible to maintain a shipping grade of ore containing 44 per cent. of chromic oxide. According to the company's engineers, the deposit has been calculated to contain about 50,000 tons of chrome ore, one-half of which should carry more

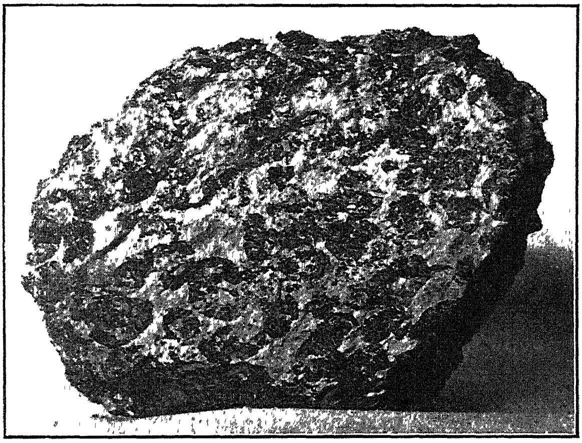

FIG. 5.—COARSELY SPOTTED CHROME ORE FROM CALEDONIA CLAIM. 0.66 NATURAL SIZE.

than 40 per cent. and the other half between 34 and 40 per cent. chromic oxide. By rough concentration, for which abundant water is available in the creek, the lower-grade ore should yield 15,000 tons of concentrates carrying 40 per cent. of chromic oxide; hence a total of 40,000 tons of ore of 40-per cent. grade may be available.

*Cayoguan Group.*—This group, which includes the Cayoguan, the Narciso, and the Cromita No. 1, 2, and 3, occupies both sides of Cayoguan River, about 5 mi. (8 km.) above its mouth in Moa Bay (Fig. 6). It is about 8 mi. (13 km.) by trail from an old wharf at Punta Gorda; at present the first 3 mi. of this distance may be traveled by scows on Cayoguan River, then a foot trail near the river is followed to a camp site about ½ mi. below the nearest deposit. In order to transport ore from the deposits, a road must be built down the Cayoguan, a difficult undertaking, because the river occupies a narrow gorge bordered at many places by steep cliffs. These claims were reported to be under

option to the Cauto Mining Co., which was at that time actively mining manganese ore in Oriente Province.

The Cayoguan claim is on the right side of the river, about 600 ft. (183 m.) above it and 750 ft. (229 m.) above sea level. An area extending around the hillside for 300 ft. (91 m.) horizontally, with a vertical range of 20 ft. (6 m.), has been prospected by a few small cuts which disclose massive chrome ore, generally of good appearance where freshly exposed. Boulders of ore also outcrop and there is considerable float on the slopes below the outcrops. Where weathered, the ore shows

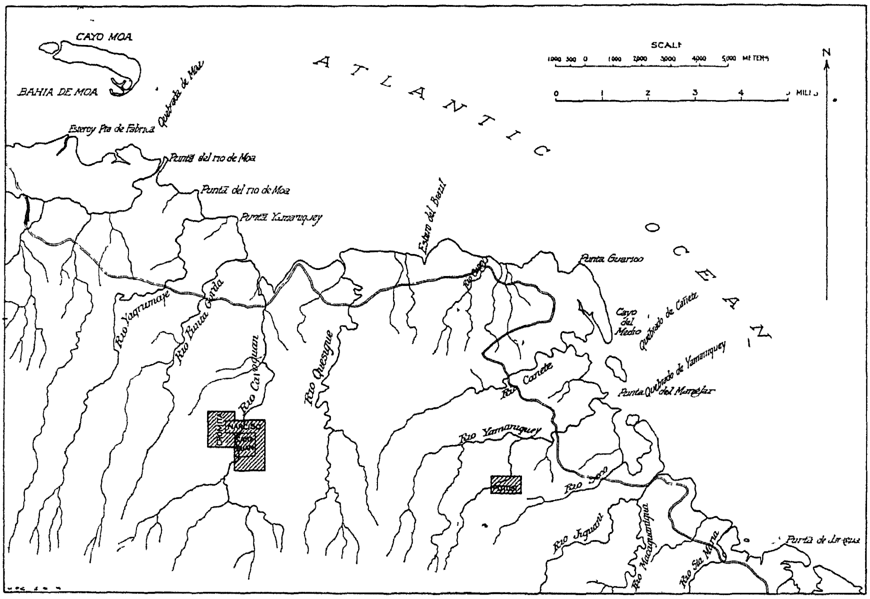


FIG. 6.—MAP SHOWING LOCATION OF CAYOGUAN GROUP AND POTOSI CHROME-ORE DEPOSITS

stains of iron oxide. The ore is mostly medium grained and black (Fig. 7), but carries a minor amount of spotted material. Thin films of emerald-green crystalline uvarovite occur in the chromite. A representative sample of the ore contained 38.1 per cent. of chromic oxide.

The Narciso claim, which nearly surrounds the Cayoguan, contains an orebody which outcrops on a steep hillside 500 ft. (152 m.) above the river, and at a distance of 600 ft. (183 m.) south of the Cayoguan prospect pits. It is accessible only by way of the trail to the Cayoguan claim and a climb down from the top of the bluff. The orebody is bordered by serpentine on the north and by a partly altered, dark basic rock on the south. The outcrop is in a steep watercourse which was dry at the time of visit, and the deposit was exposed to a width of 20

ft. (6.1 m.) and a height of 30 ft. (9 m.). The ore is medium-grained and appears to be mostly of fair grade, but some of it shows ferruginous stains and some is mixed with specks of serpentine. A sample from this outcrop showed 34.8 per cent. of chromic oxide.

The Cromita claims are situated on the left side of Cayoguan River, a short distance down stream from the Cayoguan location. These claims contain three known orebodies and hundreds of tons of boulder float strewn along the bottom of an arroyo for more than ½ mi. (0.8 km.). The float ore may have originated in an orebody now entirely eroded. The orebodies still in place are exposed in a river bluff at a height of 150 to 350 ft. (46 to 107 m.) above the river. The most northerly one

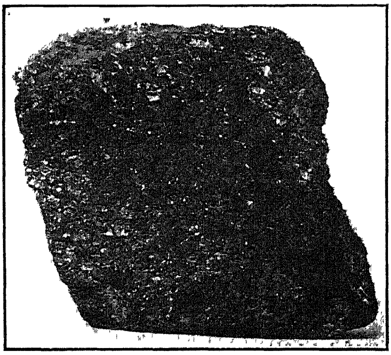

FIG. 7.—COMPACT GOOD GRADE CHROME ORE FROM CAYOGUAN CLAIM. REFLECTION OF LIGHT FROM BRIGHT SURFACES OF CHROMITE CRYSTALS GIVES THE SPECIMEN THE APPEARANCE OF CONTAINING MORE LIGHT-COLORED GANGUE MATERIAL THAN IS ACTUALLY PRESENT. 0.55 NATURAL SIZE.

shows a face 20 ft. (6.1 m.) wide and 15 ft. (4.6 m.) high, and has been prospected by an open cut, a sample of which contained 31.9 per cent. of chromic oxide. The middle body, which shows an outcrop 75 ft. (23 m.) long and 50 ft. (15 m.) high, has been penetrated 35 ft. (11 m.) by a cut and a tunnel. A sample from the fines in the tunnel dump carried 25.9 per cent. chromic oxide. The southerly orebody is exposed to a length of 60 ft. (18 m.) and a height of 40 ft. (12 m.), but its thickness had not been determined. A sample from the outcrop contained 40.5 per cent. chromic oxide. The alignment of the outcrops, their similarity in strike, and other geologic relations suggest that the middle and southern orebodies, which are about 75 ft. (23 m.) apart, may possibly be parts of one lens of ore and thus be connected within the hill. The chromite on these claims is of medium grain, but a little coarser than that on the claims at the opposite side of the river. Seams of uvarovite are especially conspicuous in the chromite at the north prospect opening.

The analyses, by R. C. Wells, in Table 3 show the general composition of the ore in this locality:

TABLE 3.—*Analyses of Chrome Ore from the Cayoguan Group of Claims, near Moa Bay*

| | 12 | 13 | 14 | 15 | 16 |
|---|---|---|---|---|---|
| $Cr_2O_3$ | 38 1 | 34 8 | 31 9 | 25 9 | 40 5 |
| Fe | 11 7 | 10 2 | 11 9 | 13 0 | 11 2 |
| $SiO_2$ | 0 9 | 2 4 | 4 3 | 12 5 | 1 9 |
| $Al_2O_3$ | 27 0 | 29 0 | 25 2 | 20 1 | 25 7 |
| S | . | .. | ... | Trace | None |
| P. | ... | | | Trace | None |
| Ni. | ... | ... | Present | Trace | Present |

Sample 12, Cayoguan claim; 13, Narciso claim; 14, Cromita claim, north body; 15, Cromita claim, middle body; 16, Cromita claim, south body.

The deposits of the Cayoguan group contain probably 22,500 tons of available chrome ore but may possibly yield 60,000 tons or more. These estimates include 2000 tons of float ore in Cayoguan River and a tributary arroyo.

*Potosi.*—The next deposit toward the east, which was examined, is on a claim known as the Potosi, on Saltadero Creek 4 mi. (6.4 km.) above its mouth. This creek is a tributary of Yamaniguey River, which enters Canete Bay 4 mi. below the mouth of the creek. In going to the deposit, the party was carried by scows about 2 mi. up Yamaníguey River and thence proceeded on foot by a trail to a camp site on the river just below the mouth of Saltadero Creek. From this point the deposit was reached by following the bed of Saltadero Creek, which it is generally possible to wade during the dry season. The Potosi claim is reported to be held for a company manufacturing refractory material in the United States.

The orebody is a lens dipping 50° to 70° toward the northwest, reaching a depth of more than 100 ft. (30 m.), and at one place has a thickness of 25 ft. (7.6 m.). The upper outcrop is on the steep mountain side 325 ft. (99 m.) above Saltadero Creek and 600 ft. (183 m.) above sea level. The orebody has been prospected by two drifts started respectively 50 and 100 ft. (15 and 30 m.) below the outcrop, and by a crosscut 80 ft. (24 m.) distant from and a little lower than the upper drift. All these openings reach the orebody within short distances but do not intersect it in such a way as to show definitely its dimensions. There is much float ore on the slope below and also down the stream bed, some of it being in the form of boulders 8 ft. thick and 12 ft. long (Fig. 8); probably 2000 tons of float ore could be recovered here.

The ore is medium to coarse grained. Some of the material in the drifts is spotted, but most of the outcropping and float ore are black and of good appearance. According to analyses accompanying the report of G. W. Maynard, who prospected the deposit in 1903, the representa-

tive ore contains 35 to 41 per cent. of chromic oxide, 1.4 to 15 per cent. of iron, 1.5 to 5 per cent. of silica, 5 to 17.5 per cent. of magnesia, and 25 to 29 per cent. of alumina. The orebody contains small masses of peridotite, which may reduce materially the quantity of ore available, as well as seams of serpentine and of olivine. This deposit contains 10,000 to perhaps 20,000 tons.

The work of getting this ore to the coast involves a difficult problem in transportation. The gorge of Saltadero Creek is too narrow and winding, and in places too steep, to permit the construction of any kind of road except at great expense, and even if a road could be built down to the mouth of Yamaniguey River it is doubtful whether steamers of proper draft could enter Canete Bay. The only feasible plan appears to be

Fig. 8.—Boulders of chrome ore in Saltadero Creek 300 ft. vertically below orebody on Potosi claim.

that of constructing an aerial tramway, about 3 mi. (4.8 km.) long, from the deposit over the mountain and down to a point on the coast 2 mi. (3.2 km.) southeast of Canete Bay, from which a cart road or light tramway may be built 9 mi. (14 km.) southeastward to Taco Bay, where there is fair anchorage for steamers.

*Constancia.*—This claim is situated 3/4 mi. (1.2 km.) south of Navas Bay and 100 ft. (30 m.) above sea level. It is reported to be owned by Tomas Gomez of Baracoa, Cuba, and is said to have been denounced originally as iron ore. There is no road to the claim at present, but one could easily be constructed.

A small body of chrome ore outcrops on this claim. It appears to

extend 50 ft. (15 m.) along the face of a gently sloping hill and has been opened by a cut 25 ft. (7.5 m.) long and 5½ ft. (1.8 m.) deep. A few float boulders were noted in a gully 75 ft. northeast of the prospect, at about the same level. The ore is not of uniform quality; it is mostly low-grade and spotted, that is, chromite mixed with much serpentine gangue, but 6 ft. of fair ore is exposed in the cut. The mixed ore occurs in aggregates of chromite grains intergrown with quartz, magnesium carbonate (?), and serpentine. Thin sections of the two grades of ore, (*a*) the better grade, and (*b*) the average grade, are described by E. F. Bliss, as follows:

(*a*) A heavy black to green rock composed chiefly of lustrous black chromite in rounded grains, embedded in a gangue of light green serpentine and partly covered with a white material which penetrates the rock in veinlets. The chromite occurs in masses of dark brown color and the material that penetrates and surrounds the chromite is seen to be composed of an aggregate of two white, cloudy, feebly birefracting minerals, one with low index, and one with a fairly high index, probably some varieties of serpentine. Associated with this birefringent material are two clear isotropic minerals One has an index slightly higher than Canada balsam and contains isotropic grains with index considerably higher than balsam. The section shows a small amount of colorless to green serpentine.

(*b*) Similar in character to the better-grade ore, but the proportion of serpentine gangue to chromite is larger. The original character of the olivine is evident under the microscope and a few cores of original olivine still remain, whereas in the high-grade ore the original character of the gangue material is almost completely obscured The contact between the olivine and chromite is often marked by a narrow rim of what is probably a recrystallized form of serpentine. This material, which appears to be possibly of later origin than the serpentine gangue, penetrates the serpentinized olivine in minute stringers along the cracks.

Analyses by R. C. Wells, of representative samples of the average ore and of the better ore are given in Table 4.

TABLE 4.—*Analyses of Chrome Ore from Constancia Claim, near Navas Bay*

| | 17 | 18 |
|---|---|---|
| $Cr_2O_3$ | 27.6 | 39.4 |
| Fe | 11 9 | 11.5 |
| $SiO_2$ | 8 9 | 4 9 |
| $Al_2O_3$ | 25 3 | 20 5 |

Sample 17, mixed ore, 18, clean ore.

This material might be concentrated. No estimate of the quantity of ore could be made; very little float was seen near it and there are no indications of a large deposit. Water for concentration is available nearby in Navas River, and a road could easily be built to Navas Bay, which, however, is not deep enough for steamers, so that the ore would

have to be lightered 4 mi. (6.4 km.) northwestward to Taco Bay, or 10 mi. (16 km.) southeastward to Baracoa.

It is credibly reported that a body of at least 10,000 tons of material similar to the low-grade ore at the Constancia claim lies in the mountains 8 mi. (13 km.) south of Navas Bay, but this deposit could not be examined within the time available.

*Other Deposits.*—The occurrence of a number of other chrome-ore deposits in the mountains of eastern Oriente Province has been reported. One of these, on the Desiada claim, is in the vicinity of the Caledonia. On Yamaniguey River deposits have been reported on the Esperanza and on the Delta claim. Low-grade ore was reported on claims known as the Tinta, San Marino, and Tivoli. Near the Potosi claim others had been denounced for chromite, including the Cuba, Suerte, Republic, and Carmen. None of these reported deposits could be visited, but from inquiry it appears that none of them is considered to carry exceptional material, and all of them are disadvantageously situated with regard to economical exploitation and shipment of ore.

## Chromiferous Iron Ores

None of the extensive deposits of chromiferous iron ore that are found on the north coast of Oriente and Camaguey provinces was studied, but it is possible that a brief reference to them may be of interest to one who desires to look further into the subject.

Brown iron ores carrying appreciable percentages of chromium occur as lateritic mantles overlying serpentine throughout large areas near the north coast of eastern Cuba, in the provinces of Oriente and Camaguey. The ore is in the form of ferruginous yellow clay with a top layer of spongy limonite and small hard pellets of limonite. The thickness of the deposits varies, the pellet or "shot" ore generally not exceeding a few feet, but the ferruginous clay is in places more than 50 ft. (15 m.) deep. Locally the deposits contain abnormal percentages of chromium, such portions probably representing residual accumulations from broken-down bodies of chromite, but the raw ore as at present mined in the Mayari district carries about 1.5 per cent. chromium. In nodulizing, the greater part of the water is driven off and the percentages of the metallic constituents are increased proportionately so that the chromium rises to about 2 per cent. Between 0.5 and 1 per cent. of nickel is also present in the ore. Both the chromium and nickel contents are utilized in making "Mayari" steel, in which 1.3 to 1.5 per cent. of nickel and 0.3 to 0.5 per cent. of chromium[3] are present.

The principal areas containing these chromiferous, hydrous iron oxides are the Mayari district, 12 mi. (19.3 km.) south of Nipe Bay, the

[3] Communication from Bethlehem Steel Co, Feb. 14, 1918.

Moa district, near Moa Bay, about 50 mi. (80 km ) east of Nipe Bay, both in Oriente Province, and the Cubitas district, 15 mi. (24 km.) north of the city of Camaguey, in the province of the same name.

A series of papers describing some or all of these deposits was published in the *Transactions*[4] in 1911 and included contributions by J. S. Cox, Jr., C. K. Leith and W. J. Mead, A. C. Spencer, C. W. Hayes, W. L Cummings and B. L. Miller, D. E. Woodbridge and J. E. Little. Three other papers[5] have appeared elsewhere.

## Production and Reserves of Chrome Ore

Under normal market conditions there was little stimulus to encourage the exploitation of chrome-ore deposits in Cuba. The reported market prices for carload lots of imported chromite *ex* ship at New York, in 1912, were $14 to $16 per ton. The occurrence of large deposits near the sea in foreign countries, notably Greece, New Caledonia, and Rhodesia, has enabled chromite to be delivered and sold at low prices along the eastern coast of the United States. War conditions changed the situation materially, increasing the demand for chrome ore as well as the costs of transportation and labor. Import restrictions and a shortage of vessels also made supplies of chromite from distant sources difficult to obtain. Prices therefore rose rapidly, and in 1918 domestic ore of 38-per cent. grade was quoted at $47.50 per ton at railroad shipping points in California; low-grade ore from Canada was valued at an average of $31.56 per ton, and the average value of high-grade ore at shipping ports in Rhodesia, New Caledonia, and Brazil was $28.17 per ton.[6] These prices stimulated chrome mining in Cuba, but not to the extent that might have been expected when the proximity of this island to the chrome-consuming centers in the United States is considered.

Prior to 1916 there are no available records of imports of chrome ore from Cuba into the United States, although it is probable that some ore may have been imported in earlier years. Shipments to the United States, moreover, do not represent the total production of chrome ore in Cuba, for a little may have been shipped to other countries and some stocks never have been shipped from the mines. The imports for 1918 were 8821 tons averaging 40 per cent. $Cr_2O_3$, or 7057

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[4] *Trans.* (1911) **42,** 73–169.

[5] A. C. Spencer: Three Deposits of Iron Ore in Cuba. U. S. Geol. Survey *Bull* 340 (1908) 318–329

J. F. Kemp: The Mayari Iron-ore Deposits, Cuba. *Trans.* (1915) **51,** 3–30.

C. M. Weld: The Residual Brown Iron Ores of Cuba. *Trans.* (1909) **40,** 299–312.

[6] Statistics communicated by J. S. Diller of the U. S. Geological Survey.

tons carrying an equivalent of 50 percent. $Cr_2O_3$. More than half of this quantity came from the Caledonia mine. It seems probable that the total production of chrome ore in Cuba to the close of 1918 has amounted to between 7500 and 8000 gross tons of the equivalent of 50 per cent. $Cr_2O_3$.

The reserves of marketable chrome ore in Cuba were estimated by Mr. Burch and the writer to range from 92,500 to 170,000 gross tons, practically all of it in Oriente and Camaguey provinces. The largest known deposits—those of the Caledonia, Cayoguan group, and Potosi claims—are near the northeast coast of Oriente, in a region difficult of access. Together they may yield 72,500 to possibly 130,000 tons of material, most of which can be brought to present commercial grade by simple concentration. With suitable transportation facilities and mining equipment, and sufficient labor, most of this ore could be mined and shipped within two years after these conditions had been established. At the time of the examination, only one deposit was ready for production and that on a very small scale, but its rate of production could have been greatly increased by the employment of more miners and pack animals or by the use of an aerial tramway. It would probably require more than a year to put the other deposits in eastern Oriente into shape for production. The next largest known group of deposits is near Camaguey. They are easy of access but are of lower grade than those in Oriente Province. They appear to contain 20,000 to 40,000 tons of ore, most of which can be gathered by hand from the surface.

Near Matanzas, Cardenas, and Holguin there are a few small ore deposits and it is possible that further search may reveal bodies of greater commercial importance. The ore near Holguin is of medium grade, but that near Matanzas and Cardenas is generally of lower grade. The expense of hauling is reported to have been almost prohibitive during the war and thus to have retarded production.

## DISCUSSION

E. I. Montoulieu, Havana, Cuba.—A very high grade of chrome deposit was discovered, shortly after Mr. Burchard's visit, north of Coliseo in the Province of Matanzas, and to the east of the city of that name. Over 500 tons of ore containing more than 50 per cent. of chromic oxide were shipped monthly until the signing of the armistice. Another large deposit was discovered in Camagüey Province, between the city of this name and Nuevitas. Development was just starting at the end of the war. A report of other chrome deposits in Santiago Province was also published, but no estimate of possible tonnage was available.

[7] U. S. Bureau of Foreign and Domestic Commerce.

J. A. Ede, La Salle, Ill.—What is the relation between the chrome-ore and the manganese-ore deposits; how do they stand?

E. F. Burchard.—All of the manganese deposits are found in later sedimentary rocks. The chrome ore is found in serpentine which has been formed by the alteration of basic rocks that have cut older rocks than those containing the manganese deposits. The manganese ores, for the most part, lie in rocks of upper Eocene age that may be said to overlap the serpentines. I am not familiar enough with the geology of Cuba to absolutely place the serpentines. They seem to be younger than certain limestones that have been classed as Jurassic.

# Magnesite: Its Geology, Products and Their Uses*

BY C D DOLMAN,† CHEWELAH, WASH

(Chicago Meeting, September, 1919)

SINCE the outbreak of the war we have discovered in the United States minerals of which there was no general knowledge, and which compared very favorably with anything that could be found in any foreign country. One of these was magnesite, or the carbonate of magnesium as it occurs in nature. This mineral crystallizes in the rhombohedral form, has a specific gravity of 3.00, molecular weight of 84.4, molecular volume of 28.1, and a hardness of 3.5 to 4.5.[1] Had it not been for the development of the California deposits of this mineral and the discovery of immense new deposits in Stevens County, Wash., the production of steel during the war would have been more seriously handicapped than it was. Now that the war is over, we face the possibility of competition with the cheap Austro-Hungarian magnesite; and while that may not be so cheap as it was before the war, adequate protection should be given the producers in this country to insure the full development of our own resources.

The consumption of magnesite in the United States in 1913, the year preceding the war, was 172,591 tons of calcined and 22,872 tons of crude magnesite. Less than 3 per cent. of this amount was produced at home. At the present time, practically all of the magnesite used in the United States is produced at home. Expensive plants have been constructed, extensive exploration work done, and production pushed to the utmost to supply the necessary requirements of the war. Thus a considerable industry has been built up within our borders. Some magnesite has been imported from Canada and Mexico, but the Canadian ore was of an inferior grade and could not be used satisfactorily. During the year 1918, production decreased in both the United States and Canada; the production in the United States was 225,000 tons in 1918 as against 315,000 in 1917, and in Canada 39,365 tons in 1918 as against 58,090 in 1917. This decrease was mainly due to over-production during 1917, the domestic demand being only 300,000 tons.

The material produced and sold in the United States must contain, in the crude form, not more than 3.5 per cent. silica and 2 per cent. lime;

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* Read at meeting of Columbia Section, under auspices of Spokane Engineering and Technical Association, March, 1919.

† Chief Chemist, Northwest Magnesite Co.

[1] U. S. Geol. Sur. *Bull.* 330, 351.

and in the dead-burned finished product, not more than 7 per cent. silica, 4 per cent. lime, and 8 per cent. iron and aluminum oxides.

Although magnesite is found in a number of different countries and seems to be widely distributed, there are only a few deposits of any great extent and capable of extended development. In the United States, there are two localities containing magnesite deposits capable of commercial development; namely, the California deposits and those of Stevens County, Wash. These deposits differ essentially and the Washington deposits are the more extensive and more easily mined. The California magnesite is similar to the Grecian, in being amorphous and occurring in serpentine rocks. The Washington deposits are similar to the Austro-Hungarian and Canadian ores and are of the massive crystalline variety.

The prevailing impurities in the rock that cause trouble are silica and lime mostly in the forms of talc, asbestos, quartz, and calcium carbonate.

TABLE 1.—*Analysis of Magnesite*

| Source of Magnesite | $CO_2$ | $SiO_2$ | CaO | FeO | $Fe_2O_3$ | $R_2O_3$ | MgO | $H_2O$ |
|---|---|---|---|---|---|---|---|---|
| Austrian, one sample | . .. | 0.55 | 4 95 | . | . . . | 4 40 | | |
| Napa County, Calif. | 50 65 | 1.99 | 0 67 | 1 49 | .... | .... | 45 01[a] | |
| Napa County, Calif. | 50 11 | 2 43 | 1 57 | 2 40 | .. . | . .. | 43.62[a] | |
| Napa County, Calif. | 49 05 | 4 35 | 2 11 | 1 71 | . .. | .... | 42 62[a] | |
| Napa County, Calif | 44 64 | 12.65 | 0 95 | 0.80 | .... | .... | 40.73[a] | |
| Finch Quarry, N W Magnesite Co, Stevens County, Wash | 49 70 | 1 85 | 1 74 | 0.88 | . .. | .... | 45.20[a] | |
| Allen Quarry, American Mineral Productions Co., Stevens County, Wash. | 47 91 | 5 33 | 0 35 | 1.01 | .... | .... | 43.52[a] | |
| Allen Quarry, American Mineral Productions Co, Stevens County, Wash | ..... | 2.73 | 1 77[b] | | | | | |
| Finch Quarry, N W. Magnesite Co, Stevens County, Wash, high-grade material | . .. | 1 56 | 0 31[b] | ... | .... | 1.28 | 45 24[b] | |
| Finch Quarry, N W Magnesite Co, Stevens County, Wash, Medium-grade material | ..... | 2 25 | 0 38[b] | .... | . .. | 1.75 | | |
| Finch Quarry, N W Magnesite Co, Stevens County, Wash, as used for ferro-magnesite | . .. | 3.28 | 2 45[b] | ... | .. . | 1.84 | 42.48[b] | |
| Austro-Hungarian magnesite | 50 44 | 0.45 | 0 97 | . . | 3 65 | 43 82[c] | | |
| Austro-Hungarian magnesite | 47 99 | 5.83 | 4 52 | ... | 2 02 | 39 54[c] | | |
| Grecian magnesite | 51 77 | 0 20 | 0.51 | ... | 0 40 | 47.11[c] | | |
| Grecian magnesite | 51 26 | 0 90 | 1 53 | .. | 0 86 | 45 45[c] | | |
| Grecian magnesite | 49 88 | 1.63 | 1 44 | .... | 1.36 | 45 75[c] | | |
| Hydromagnesite from Atlin, British Columbia | 35 98 | 1.86 | 2 04 | .... | 1 42 | 41.13[c] | | 18 02 |
| Hydromagnesite from Atlin, British Columbia | 36 17 | 0 54 | 0 68 | .. | 0 92 | 42.19[c] | | 19 05 |
| Magnesite from Bridge River, British Columbia | 47 28 | 7 46 | 0 46 | .... | 1 04 | 43 42[c] | | 0 68 |
| Magnesite from Bridge River, British Columbia | .. .. | 4 08 | 3 25 | . .. | 1.54 | 42 20[c] | | |

[a] Analyses from U. S Geol. Survey *Bull*

[b] Analyses made in Laboratory of Northwest Magnesite Co

[c] Analyses from Canadian Geol. Survey *Mem.* 98.

The iron is added to improve the finished product by causing it to sinter more easily, thus giving it better binding properties. The lime causes the most trouble as it is difficult, if not impossible, from an industrial standpoint, to remove it economically, while the silica can be removed to a certain extent without too much expense. A few analyses of the different important deposits of the mineral are given in Table 1.

With a production of 110,000 tons of crude ore from June, 1918, to April, 1919, only 20 per cent., or one ton out of five, mined at the quarries of the Northwest Magnesite Co. was waste, whereas, in the Austro-Hungarian quarries it is necessary to mine and sort five tons of crude ore to produce one ton of finished product, or, two tons out of every three mined is waste. From this it may be seen that with close sorting a product just as pure, if not more pure than the foreign material, may be obtained from domestic sources. No great difficulty has been encountered by the Stevens County producers in meeting the requirements of the refractory trade.

## Geology of Magnesite

The deposits of magnesite with which this paper deals more particularly are located near the town of Chewelah, Stevens Co., Wash., about 65 mi. (104.6 km.) northwest of Spokane. The geology of the district, as given by the U. S. Geological Survey, is as follows:

"The Stevens County magnesite occurs as a replacement of lenses of dolomite in sedimentary rocks, probably of pre-Cambrian age. Basic igneous intrusives occur above and below the magnesite in some places, but so far as observed not in contact with it. The dolomite is interbedded with schist, slate, and quartzite, and the lenses are from a few hundred to a few thousand feet long. Replacement by magnesite is variable from place to place, some parts of an original dolomite deposit being wholly replaced and others scarcely altered at all. The magnesite is crystalline, varying from very fine to coarse grain, and in color is gray, white, black, pink, and red. The magnesite deposits are in mountainous country, where forest cover and hill wash conceal most of the bed-rock, and as the outcrops are discontinuous and the depth of the deposits is unknown, estimates of the quantity available may vary extremely. Computations of the quantity of magnesite in these deposits are astoundingly large when compared with the quantity of magnesite found at other localities in the United States. On more than one of the properties an estimate of 1,000,000 tons of ore within 100 ft. (30.48 m.) of the surface is reasonable. It is safe to say that there are 7,000,000 tons of magnesite in the Stevens County district, and exploratory drilling may multiply this estimate many fold. The crystalline magnesite of Stevens County is suited to refractory uses and is being shipped to large manufacturers of magnesite products in the Central and Eastern States. The crude

material in Stevens County is abundant and of good quality and the new industry in Washington, if properly managed, should supply a large part of the domestic demand."

Referring more particularly to the Finch deposit, which is now being operated by the Northwest Magnesite Co., it may be said in general that the deposit is underlaid by serpentine and overlaid by quartzite. The magnesite is supposed to occur as a replacement in dolomite resulting from the action of circulating solutions rich in magnesia derived from the ser-

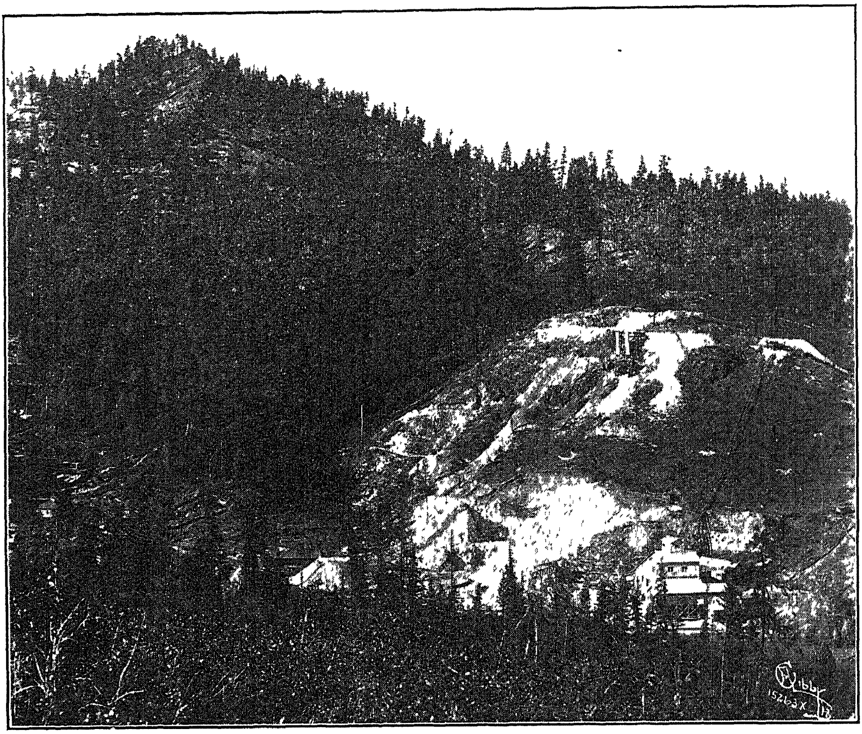

FIG. 1.—THE FINCH QUARRIES.

pentine. In contact with the serpentine, there is usually found a shale very high in lime. The lime content decreases as the distance from the serpentine increases and finally grades through dolomite to magnesite of varying purity.

On the Finch property there are four distinct zones in which the replacement has been sufficiently complete to result in commercial magnesite. The general strike of these zones is nearly north and south and the dip about 60° to the west. The deposit has been explored by diamond drilling to a depth of 300 ft. (91 m.). Portions of the magnesite are decidedly laminated; other portions are massive. This latter, in general,

is the purest material, due to the fact that the laminated portions contain thin layers of talc and siliceous material. The magnesite is of crystalline structure and varies from very fine to very coarse. The color may be white, gray, pink, or nearly black. It occasionally has a distinctly bird's-eye appearance, black spots occurring in a gray matrix. No amorphous magnesite is found in this locality. Magnesite, being less easily eroded than dolomite, is usually evidenced by bold outcrops. The portions of a deposit that are covered by overburden will, in general, be found to be high in lime.

The trend of the Stevens County deposits is from northeast to southwest, covering a total distance of about 10 mi. (16 km.) The Finch property is on the extreme northeast end. Next comes the Allen property about ½ mi. distant, then the Woodbury approximately 1 mi. (1.6 km.) from the Allen. There is then a gap of 4 mi. (6.43 km.) to the Keystone

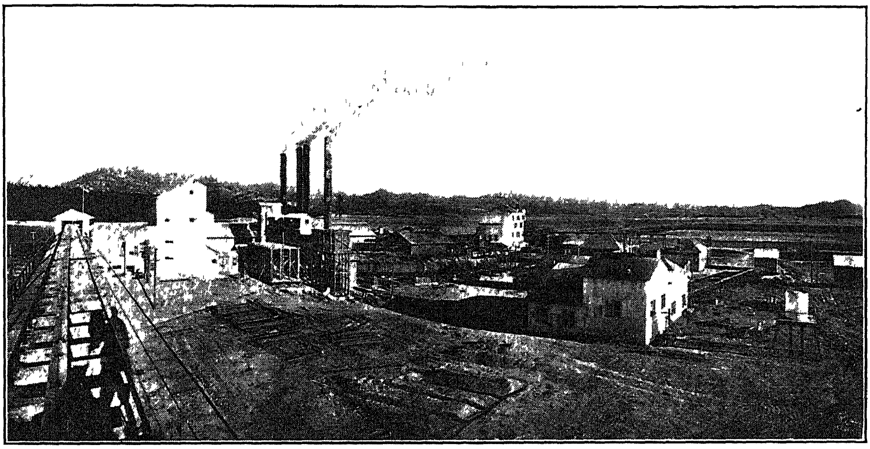

FIG. 2.—TRAMWAY, TERMINAL MILL, AND DEAD-BURNING PLANT.

property belonging to the Northwest Magnesite Co. About 2 mi. west of the Keystone is the Midnight claim, also belonging to the above company. Next come the Double Eagle and Red Marble, the latter being at the extreme southwest end of the zone.

The Keystone deposit contains a lens of brucite, the hydrated oxide of magnesia. It stands vertically in the magnesite, is about 30 ft. (9.14 m.) through by 100 ft. (30 m.) long and of unknown depth. It was discovered many years ago and thought to be marble. A company was formed to exploit the deposit but proved a failure as the material weathered badly on exposure. An investigation by the Geological Survey proved it to be brucite instead of marble and mentioned the association with crystalline magnesite. When magnesite was sought in this country at the beginning of the war, these old records were found and an investi-

gation of this deposit resulted in the discovery and development of the Stevens County magnesite.

## Products and Uses of Magnesite

By far the largest demand for magnesite at the present time comes from the steel makers, who use it in the form of dead-burned magnesite containing about 84 per cent. magnesia, 6 per cent. silica, 3.5 per cent. lime, 6.5 per cent. to 8 per cent. iron and aluminum oxides. It is also known as ferromagnesite. The bulk of this material is consumed by the basic open-hearth process of making steel. In this process, magnesia in the form of ferromagnesite is practically a necessity for the bottoms of the furnaces. Calcined dolomite can be used but it slacks so easily upon exposure to the atmosphere that it causes much trouble. In addition, it is said to be almost impossible to obtain a dolomite bottom that will not become detached in spots and float. In constructing the bottom of an open-hearth steel furnace, a few courses of magnesite brick are first laid on the steel shell forming the bottom of the furnace, then thin layers of grain magnesite are successively fused on this foundation. While thus partly fused, it is easy to shape the bottom as desired.

Magnesia brick are also used in the construction of soaking pits, metal mixers, billet and bar heating furnaces, copper converters and many special types of furnaces. In the form of grain and brick magnesite is also used extensively in copper reverberatories.

The plants installed in the Stevens County district are now equipped solely for producing the types of refractory magnesite used in the various furnaces. These plants are owned by two companies and when operating at full capacity are able to furnish a large proportion of all the magnesite required in the United States under normal conditions. The American Mineral Production Co. has four upright stationary kilns at one quarry (the Allen) and two at another (the Woodbury). Two large rotary cement kilns located at Irvin, Wash., are used by this company to make ferromagnesite. A standard gage railroad about 7 mi. (11 m.) long has been constructed from the quarries to Valley, Wash., where it connects with the Great Northern Railway.

The Northwest Magnesite Co., which is the largest producer in this district, has a completely equipped plant for grinding and burning the ore and the quarries are provided with the most up-to-date machinery and elaborate quarters for the workmen. These quarters include a steam-heated clubhouse and a bunkhouse provided with steam heat, beds and individual rooms. A mill for the preliminary crushing of the ore is also located at the quarry besides all the ordinary equipment necessary for mining. The dead-burning plant of this company is situated on the Great Northern Railway close to the town of Chewelah, Wash. The plant, together with the development and equipment of the quarries,

represents an investment of about $1,000,000. It consists of a Bonnot coal-pulverizing system with a capacity of 6 tons of pulverized coal per hour, a mill for grinding and mixing crude magnesite and iron ore, five 125 by 7½ ft. (38.1 by 2.28 m.) rotary kilns, a mill for crushing the finished product, storage floor and docks capable of storing 1000 tons of ferromagnesite, an electric power plant, office building, chemical laboratory, etc. The ore is carried from the quarries to the plant, a distance of 5 mi. (8 km.) by an aerial tram. This tram was built by the Riblet Tramway Co. of Spokane, Wash.; it has a capacity of 1200 tons per day and has operated with great efficiency. The capacity of the whole plant is about 7500 tons per month of dead-burned or ferromagnesite. The production for 1917 and 1918 was 61,805 tons and 80,430 tons of ore, respectively, calculated to a crude basis.

The materials produced are confined to ferromagnesite and calcined magnesite, exclusively, since the demand for these materials, until recently, required all that the quarries could yield.

Calcined magnesite is the crude magnesite burned at a high enough temperature to drive off the carbon dioxide (about 1200° C.); ferromagnesite on the other hand must be proportioned and burned with great care. The crude magnesite is first crushed very fine and is then mixed with ground iron ore of a high grade mined at Chesaw, Wash. This material is then passed through rotary kilns, where the temperature must be carefully controlled and comes out in the form of small dark red balls, which are thoroughly dead-burned and shrunk. This product is then crushed and screened, the coarser particles being sold for the manufacture of magnesia brick and the finer particles, called "grain magnesite," being sold in bulk without further treatment for placing in basic open-hearth steel furnace bottoms.

A dead-burned, sintered magnesite is sometimes made without the addition of iron ore. This product is used in making brick still more resistant to heat than the regular ferromagnesite. It requires about 2.2 tons of crude magnesite to produce 1 ton of dead-burned magnesite. Calcined magnesite also is used in the manufacture of magnesite brick.

Other refractory products manufactured from magnesite are magnesite crucibles, sintered magnesite tubing, magnesic porcelains and pure fused magnesia. Of these, the last two are the more important. Magnesic porcelains[2] are composed of a mixture of magnesite and alumina or various clays. The peculiar property of this mixture is that the proportions of magnesite and alumina can be varied over an extremely wide range and the product still have a refractoriness greater than 1710° C. and great resistance to sudden changes of temperature. In the manufacture of this

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[2] *Trans.* Amer. Cer. Soc. (1913) **15,** 606–619 and A. B. Searle: "Refractory Materials." London, 1917. C. Griffin & Co.

product, according to Heinecke's method, 72 parts of magnesite and 28 parts of alumina are mixed together, ground to a powder, burned at 1470° C., cooled, powdered, mixed with dextrine, molded and again burned at 1710°. This refractory should find many uses, especially in processes where siliceous vessels cannot be employed.

Pure fused magnesia is now being made electrically by the Norton Co. at Niagara Falls and is used as a basic refractory for electric-furnace linings. Another interesting use for this material that has come to the writer's attention is as an electrical insulator at high temperatures. The research laboratory of the General Electric Co. has worked out a method of employing fused magnesia in the manufacture of sheathed resistance wire for heating elements used in a large number of devices. The fused magnesia is applied as a coating on nichrome resistance wire and, the writer is informed, is the best substance available for this purpose, being a good heat conductor and conducting very little electricity even at high temperatures.[3] Pure fused magnesia is transparent and very hard.

While the development of the Stevens County deposits of magnesite has depended entirely on the use of magnesite as a refractory material, there are numerous other uses to which it can be put. One of the most important is for oxychloride or Sorel cement. This material consists of a mixture of caustic calcined magnesite (magnesite which retains from 2 to 8 per cent. of its carbon dioxide), magnesium chloride, asbestos, wood fiber, marble dust or other inert material. One process for its manufacture is as follows: with each 100 lb. (45.35 kg.) of sawdust 2 to 5 lb. (0.9 to 2.26 kg). of linseed oil is thoroughly mixed; and this mass is then thoroughly mixed with ½ to 3 lb. powdered caustic soda. Then 80 lb. of calcined magnesite and 40 lb. of dry magnesium chloride are added and the whole thoroughly mixed and reduced to a powder; enough water to reduce it to a plastic state is then added. Powdered marble or other hard substances may be used in the mixture in the proportion of 250 lb. (113.39 kg.) marble to 100 lb. sawdust, 100 lb. calcined magnesite, and 50 lb. (22.67 kg.) magnesium chloride. This product is much more widely used in Germany and Russia than in the United States. In Russia, it is made up into boards, which can be sawed, drilled, and used for the same purposes as wood lumber. While in the United States some is used for Pullman car floors and sanitary kitchen floors, its use is not widespread due mainly to ignorance, faulty construction, and the poor quality of material supplied. With proper material and methods, this product could be used for many building purposes because it is both fire- and water-proof, is easily laid and cleaned, germ proof, tough, elastic, takes a good polish, and can be permanently colored as desired. With

[3] Private communication from G. M. J. Mackay.

the new deposits of magnesite opened up in a vicinity where there is a great lumber industry, we have the two main ingredients at hand for making oxychloride-cement products; namely, wood fiber, in the form of sawdust, and magnesite. Magnesium chloride, another important ingredient, could be produced from the waste magnesite that is not fit for other uses.

Another substance that is being widely used and giving excellent results is magnesia insulation for boilers, steam pipes, etc. This substance, as now used, is composed of 85 per cent. light carbonate of magnesium and 15 per cent. asbestos, which is used only as a binder and not to increase the insulating qualities of the product. It can be readily produced from magnesite and if made properly is the best heat-insulating material known. This same compound, without the addition of asbestos, is used as a toilet preparation and for medicinal purposes.

A very important metal which during the war was in great demand for the manufacture of flares, illuminating bombs, signal lights, and light alloys for aeroplane construction is magnesium, also a product of magnesite, and now being made electrolytically at Niagara Falls and other places. The manufacture of magnesium in the United States began when the supply of the metal from Germany was cut off. It is now being used mainly as a deoxidizing agent, and alloyed with aluminum, copper, nickel, monel metal, brasses, and other metals[4] to make light strong castings. Two per cent. of magnesium doubles the tensile strength of aluminum, quadruples its resistance to shock, decreases its cost of machining over 50 per cent., and makes it take a better polish. As a conductor of electricity, magnesium-aluminum alloys are particularly serviceable for power transmission lines, having twice the strength of aluminum, and greater electrical conductivity, besides being lighter in weight. An alloy composed of 92 per cent. magnesium and 8 per cent. aluminum is said to be as strong as gun metal. Another alloy of magnesium which has remarkable properties is composed of lead 65 per cent. and magnesium 35 per cent.[5] When added to boiling water, this alloy will rapidly react to produce hydrogen gas useful for aircraft and other purposes, thus dispensing with the ordinary inconvenient cylinders containing the gas under pressure. After the reaction has taken place, very pure forms of magnesium and lead hydroxides remain, which can be made up into excellent paint, pure lead peroxide, litharge, and minium with excellent physical properties and by simple processes. The alloy containing 15 per cent. magnesium and 85 per cent. lead is used for abstracting oxygen from the air in the cold and is useful in creating a vacuum. The only drawback to the more general use of magnesium is

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[4] *Chem. & Met. Eng.* (Sept. 28, 1918) **19,** 525.
[5] *Chem & Met. Eng.* (Dec. 1, 1918) **19,** 776–7.

its high cost of production. Research, however, will probably overcome this difficulty in the near future.

The other products of magnesite, while not in such great demand, are still important. Among these may be mentioned magnesium sulfate, magnesium chloride, and magnesium hydroxide. Magnesium sulfate is used for medicinal purposes, tanning leather, warp sizing in cotton mills, fire-proofing, and in chemical laboratories. This salt can be made by treating magnesite with sulfuric acid and is being so made in the United States. Magnesium chloride is used for bleaching purposes, in the manufacture of oxychloride cement, in the manufacture of cold-water paint, and in the manufacture of cotton goods. This salt is now being manufactured in the United States by treating magnesite with hydrochloric acid. Magnesium hydroxide is used in sugar refining.

A considerable quantity of magnesite is used in the manufacture of chemically prepared wood pulp for paper manufacture. It is used in the form of magnesium bisulfite and is preferred on account of its greater stability and because of its greater solvent action on wood resins.

Minor uses for magnesite are: in the form of carbonate, in tooth paste and to prevent scale in boilers; as oxide, in dynamite and in the rubber industry; as organic compounds of magnesium, in medicine; as calcined magnesite, in filtering oils and neutralizing the contained acids and as paint pigment.

### Conclusion

In the foregoing the writer has tried to give as complete an idea as possible of the magnesite industry in Washington and to show some of its great possibilities. With abundant water-power available on the Spokane, Pend Oreille, and Kettle Rivers and with raw materials, such as clays, waste wood, and magnesite in large quantities close at hand, to say nothing of the other large mineral deposits, there are great opportunities for building up important and revolutionizing industries here in the Northwest.

## DISCUSSION

A. Malinovszky,* Belleville, Ill. (written discussion†).—I have been very much interested in Mr. Dolman's paper. We all realize, I think, that this question of developing our home industries and supporting the new industries which have developed during the great war is very important at present.

Mr. Dolman has well shown the quantity and quality of the raw magnesite in the United States, and the commercial and industrial possibility of our home magnesite and I believe, with him, that our home

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† Received Sept 3, 1919.

industries should receive adequate protection. There are two reasons, however, for believing that this may be difficult.

One reason is that much American capital is invested in the Austro-Hungarian magnesite industry. The raw magnesite can be quarried and prepared for the market in Austria with much cheaper labor expense than in the United States; large deposits which are very easy to quarry and good transportation facilities also contribute to the cheapness of production.

The second reason is the old saying that the magnesite found in the United States is not of the same quality as the Austro-Hungarian magnesite; this I doubt. Even if the magnesite found here does not in a raw state equal the Austro-Hungarian, it can be synthetized so that the finished article will have the same properties as the Austro-Hungarian finished magnesite refractory possesses. Raw magnesite from Greece is not the same as the Austro-Hungarian, but it has been demonstrated that a good refractory can be made from the Greek product.

The first reason can be overcome by the loyalty to the United States of the men who are interested in the foreign magnesite industry. Their choice should be not to develop foreign industries and a home market, thus depressing the home industries, but to develop the home industries and a foreign market, and thus help to build up our commerce with other nations. The United States leads in iron, steel and copper production, and therefore an enormous quantity of magnesite refractory will be consumed at home in many different industries for many years to come.

The answer to the second reason is that the well known Veitsch Magnesite Co., in Austria, has not won its name and place in the world market by the purity or special property of the raw magnesite, as many in the United States believe. The Austrian raw magnesite varies considerably in siderite (chalybite) whereas the marketed finished product is very uniform. The Austro-Hungarian magnesite refractory product has succeeded in conquering the world market by a high standard uniformity, by skilful technical management, by careful sorting, by calcining at high temperature, by careful grinding and sizing the high calcined product, by making into brick under a high pressure, by careful drying and then burning at a high temperature.

If the American raw magnesite is carefully sorted, dressed and mixed with the iron ore, and then calcined to a high prolonged temperature, carefully ground to the required size of grains, pressed under a high pressure, then carefully dried and burned at high temperature, a first class product can be obtained, equal to any magnesite refractory produced abroad. Homogeneous condition and good conversion of the magnesite to periclase by high temperature is of the greatest importance. After the raw magnesite is carefully sorted and mixed, it should be calcined to a high temperature (about 1500° C. or better) since this is the important

factor in transforming magnesite into a dense crystalline product of periclase; and a good quality of magnesite refractory brick should consist of a high percentage of periclase.

By this process of high calcination and conversion of the caustic magnesite to periclase, the maximum volume shrinkage is obtained, so that after grinding and careful sizing of the grains, the additional shrinkage in the final burning of the brick will be very small, and there will be no further shrinkage after the brick is placed in use in the furnace wall and kept in continuous furnace condition.

It is very important to standardize the finished product as the weakest point of most refractories is excessive shrinkage when in use, which leaves great spaces at the joints and causes the brick to give way.

The binder, which is also a very important factor, should be carefully selected and tested to determine the kind of binder to be used and the amount needed to give the best and most satisfactory result. The mortar in which the magnesite brick is laid in the furnace wall is just as important as the brick itself. Therefore care should be given to see that this material is as carefully prepared as the material for the brick.

After all, the aim of the American magnesite refractory makers should be to produce a uniform standard quality, so that the users of basic refractories can rely on the quality and get accustomed to it. This can be attained by skilled technical management at every stage in the manufacturing process, from the quarry to the final burning and cooling.

THEO. C. DENIS, Quebec, Can. (written discussion*).—In Mr. Dolman's paper I notice the following statement: "Some magnesite has been imported from Canada and Mexico, but the Canadian ore was of an inferior grade, and could not be used satisfactorily. During the year 1918, production decreased in both the United States and Canada."

As regards dead-burned magnesite made in the Province of Quebec, the two statements give a wrong impression. I have before me two incomplete lists of users of Quebec dead-burned magnesite to whom considerable shipments were made during 1918 and 1919. They comprise some thirty-five names, of which I will enumerate only a few: Bethlehem Steel Co.; Carnegie Steel Co.; Jones and Laughlin; Atlas Crucible Steel Co.; Halcomb Steel Co.; Ludlum Steel Co.; Algoma Steel Co.; Steel Company of Canada; Dominion Iron and Steel Co.; and others.

I have the assertion of the two principal Canadian magnesite companies producing dead-burned magnesite that from all the users of their products only one complaint was ever received, and this was that the nodules, or grains, of magnesite were rather coarse, a matter that was easily remedied.

I know personally one of the high technical officers of the Steel Com-

* Received Jan. 14, 1920.

pany of Canada, to whom I have frequently spoken of our Quebec magnesite, and he has at various times asserted that the Steel Company of Canada was getting quite as good results with our magnesite as with the Austrian article of pre-war days.

I wish also to point out that for 1917, when Quebec first began to sinter magnesite, the production of dead-burned of the Province was 3386 tons, and in 1918 it was 21,349 tons; that the clinkering of the magnesite has so far been done in old cement mills, at Hull and at Montreal, which means railroad hauls of 70 and 55 miles respectively for the crude magnesite and necessarily makeshift installations; that two companies are at present completing sintering plants at the magnesite quarries themselves, and that these facts do not bear out Mr. Dolman's statement that "the Canadian ore was of an inferior grade and could not be used satisfactorily."

C. D. Dolman (author's reply to discussion).*—The information published in the paper was received from one of the largest users of magnesite in the United States, as well as from other sources. In writing this paper it was not the purpose of the author to criticize Canadian magnesite, but to state the facts as he could determine them.

The U. S. Geological Survey *Press Bulletin* for August, 1918, says: "it (Canadian magnesite) is being substituted for the higher grade domestic material." Besides, it is generally accepted by manufacturers and users of magnesite in the United States that any dead-burned magnesite containing more than 4 to 5 per cent. calcium oxide is not satisfactory material for all refractory purposes. The author does not doubt, however, that Canadian magnesite can be profitably used for some purposes. The production of magnesite in Canada was obtained from *Mining and Scientific Press* of March 22, 1919.

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* Received April 5, 1920

# Correlation of Formations of Huronian Group in Michigan

BY R. C. ALLEN,* M A , LANSING, MICH.

(Chicago Meeting, September, 1919)

ABOUT four years ago the writer proposed a revision of the correlation of the Huronian formations in Michigan, and noted the bearing of the question on the correlations of the Huronian rocks in Wisconsin and Minnesota.[1] Dr. C. K. Leith opposed this revision and argued for the retention of the correlations of the United States Geological Survey, published in 1911,[2] for which he and Dr. Van Hise were mainly responsible. As late as June, 1917, Harder and Johnson, following Leith, thought it "premature to change radically a correlation based on years of careful field work."[3]

The work of Wolff on the Mesabi and Cuyuna Ranges, of Hotchkiss on the Gogebic Range (unpublished), and of the writer and assistants on the Michigan ranges, together with data developed by exploratory and mining operations, has introduced so many new facts not recognized in the scheme of the U. S. Geological Survey that the correlations of 1911 must be considered, in some important respects, obsolete. Indeed, new data are accumulating so rapidly that any classification, which for the moment may seem best to explain the facts, is almost immediately modified or refined by further observations.

The problem of Huronian correlations is difficult, but interesting and important. With the aid of fossils, stratigraphers of the post-Algonkian formations have been able to show that slight disconformities may separate strata of vastly different ages, and that within physically conformable strata may be concealed a hiatus of great magnitude. In the non-fossiliferous Huronian group, minor physical breaks in the strata are only beginning to attract attention, and it is too early to discuss what bearing these will have in correlating the formations of one area with those of another, but it is safe to say that the "lost intervals" which are not disclosed by physical relationships will remain forever lost; therefore we may not hope to discover the ultimate truth concerning the

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* State Geologist of Michigan.

[1] *Jnl. Geol* (Nov.–Dec., 1915) **23.**

[2] C. R. Van Hise and C. K. Leith: Geology of the Lake Superior Region. U. S. Geol. Survey *Monograph* 52.

[3] Notes on the Geology and Iron Ores of the Cuyuna District of Minnesota. U. S. Geol. Survey *Bull.* 660-A.

Huronian period, but this is not a reason why new facts should not be accommodated immediately in the general classification. There is much to be gained by continuous adjustment and readjustment of the correlations, to keep pace with new evidence, for these attempts not only bring us closer to the unattainable truth but develop new economic aspects of exploration for iron ore.

The writer's suggestions in 1915 were founded on discoveries of major unconformities in the then supposedly uppermost Huronian series (Animikie of the earlier writers) of the Gogebic, Crystal Falls, and Gwinn districts, of which no account had been taken in the most recent correlations of the U. S. Geological Survey. Since then a great unconformity in the heretofore supposedly uppermost series on the Menominee Range of Dickinson County has been brought to light. The amendments to the U. S. Geological Survey correlations which are now proposed are a consequence mainly of the following facts:

1. The Huronian series exhibits clearly three divisions or groups of similar formations separated by major unconformities in nearly every so-called "district" in Michigan, including the Marquette, Gwinn, Metropolitan, Menominee, Crystal Falls and Gogebic, and, so far as known at this time, not more than three major divisions in any district.

The most recently published correlation of the U. S. Geological Survey recognizes this succession in only a single district—the Marquette (the suggestion of a Middle Huronian series on the Menominee Range, in *Monograph* 52, 1911, was later withdrawn by its author, Dr. C. K. Leith).

2. In five of these districts the main iron-bearing formation is in the middle group. In the remaining district (Iron River-Crystal Falls) only a part of the iron formation is certainly in the middle group, while as to the remaining and greater part there is doubt whether it belongs in the middle or in the upper group.

The U. S. Geological Survey correlation places all of the iron formations outside of the Marquette district in the uppermost Huronian, or Animikie group; the iron formation of the Marquette Range alone is placed in the Middle Huronian group.

3. The recently discovered major unconformities are all within the Animikie or uppermost Huronian group of the U. S. Geological Survey, and without exception above the main productive iron-bearing member of this group. The term "Animikie," therefore, must either have a new definition or must be discarded. The Lower Huronian group of the U. S. Geological Survey remains as defined heretofore.

The writer's revision in 1915, on the basis of these facts, but not including the recent observations on the Menominee Range (which will be discussed in the second part of this paper) impressed Dr. C. K. Leith as "interesting possibilities" which he says "in the present state of knowledge cannot stand." The writer makes no claim of finality for his own

correlations, but it is difficult for him to reconcile the significant new facts just stated with "Arguments for Retaining the Present Correlations."[4] In dissenting from the suggested changes, Leith made no disposition of the new facts. If the facts are admitted, it follows that their consequences on the correlations cannot be denied. There is room for argument on the character and extent of the modifications themselves, but none, we think, on the necessity of accounting for the new facts in the classification of the Huronian strata.

The difficulties with the present U. S. Geological Survey classification have their roots in history. The field work which gave rise to that classification was finished prior to 1904. None of the six monographs on the iron ranges of Michigan, Wisconsin, and Minnesota, which were published at intervals from 1892 to 1904, contains any reference to more than two Huronian groups. The origin of the old classification seems to lie in the work of Alexander Winchell[5] who, by 1891, had divided Logan's Huronian sedimentary series of the north shore of Lake Huron (Original Huronian area) into two groups separated by a great unconformity; this is the origin of the terms Upper Huronian and Lower Huronian. Winchell's observations were confirmed by Van Hise and Pumpelly[6] in 1892, the year in which the first of the monographs was published. Subsequently, each of the districts south of the Canadian boundary was found by the geologists of the U. S. Geological Survey to contain two Huronian groups, and the subject of correlations was, therefore, apparently treated as one of great simplicity. The lower and the upper group of one district were correlated respectively with the lower and the upper group of every other district, including the Original Huronian Area. This disposition in some cases effected a union of dissimilar sedimentary groups in contiguous small areas.

The first modification of the dual classification appeared about 1904, when A. E. Seaman found a great unconformity which had been overlooked in the Huronian of the Marquette Range. The Middle Huronian was then established to accommodate this local situation. This event had as much importance to pre-Cambrian geology as the discovery of a new series, possessing as much significance as is now accorded to the Cambrian or the Devonian, would be to Paleozoic geology. No extensive re-examination of other Michigan districts in search of possible middle Huronian correlatives was undertaken, and the dual classification in Michigan, outside the Marquette Range, was carried into the final monograph (No. 52) which summarized the geology of the Lake Superior region. It should be said, however, that additional field work on the

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[4] C. K Leith *Jnl. Geol* (Nov -Dec., 1915) **23.**

[5] *Am Geologist* (1890) **6,** 360–70.

[6] *Am. Jnl. Sci.* [3] (1892) **43,** 224–32.

Michigan ranges prior to 1911 could not have added much to the correlation data Drilling and mining since 1911 has developed most of the information which it is desired to accommodate in this revised classification.

In the meantime, the writer and assistants had undertaken a systematic field exploration of the area lying between the Iron River and Gogebic districts of Michigan. Working westward from the Iron River region in 1910, the vicinity of Lake Gogebic was reached by 1913. Here, in T. 46 N., R. 42 W., just south of the lake, a situation developed which led directly to the discovery of a third division of the Huronian of the Gogebic Range, the Copps group. Crossing the northern part of this township is a belt of highly metamorphosed sediments comprising an iron formation, of thickness unmeasured, which is underlain by a graywacke-quartzite formation and overlain by several thousand feet of slate. This succession is similar to the Ironwood iron-bearing group of the eastern part of the adjacent Gogebic Range, as described by Van Hise and Irving, from which it is separated by a granite mass about 6 mi. (9.6 km.) broad, mapped and described by these geologists as Archean.[7] But we were unable to establish a satisfactory structural connection between the similar sedimentary groups on the opposite sides of this granite mass, on the assumption that it is Archean. Great masses of granite southward and eastward had been found to intrude the Huronian, and this led us to consider the possibility that Van Hise and Irving had misinterpreted the relation of the granite to the Huronian sediments of the east end of the range. In order to determine this matter, the Gogebic Range from Gogebic Lake westward for 12 mi. (19 km.) was examined during the summer of 1914, by a party in charge of L. P. Barrett, with the following results: The granite (Presque Isle) was found to intrude the iron-bearing series of the Gogebic Range, and to yield boulders and finer detritus to a conglomerate at the base of a thick graywacke-slate series (Copps Group) lying on the iron-bearing group with great angular unconformity, and overlain, with marked unconformity, by the Keweenawan series.[8]

This unconformity, it will be observed, splits the Animikie or Upper Huronian of Van Hise into two divisions separated by a major unconformity. We have suggested the name Copps for the uppermost group. It constitutes the uppermost Huronian group in this district, and therefore the Ironwood series, which is unconformably below it, can no longer properly be called Upper Huronian.

W. O. Hotchkiss has recently traced a well marked conglomerate at the base of the great Tyler slate formation for many miles in Wiscon-

[7] U S. Geol. Survey *Monograph* 19.

[8] R. C. Allen and L. P. Barrett: Michigan Geol. Survey *Publication* 18.

sin and Michigan; he thinks it marks an uncomformity of considerable magnitude. This discovery suggests the equivalence of the Tyler and Copps formations of the Gogebic Range. Lithologic characteristics broadly considered, particularly the occurrence of ferruginous beds in both formations, point in the same direction. The writer was inclined to consider the Copps and Tyler formations as identical prior to Hotchkiss's discovery, and is still so inclined, but Hotchkiss is noncommittal if not actually opposed to this assumption.[9]

Let us now consider the general bearing of this split in the old Animikie group on the correlations in Michigan. The dual classification was applied to the original Huronian area in 1891, to the Gogebic Range at about the same time, a few years later to the Marquette Range, and by 1904 it had been extended to the entire Lake Superior region. The single unconformity within the Huronian, which was identified by Van Hise and associates, now appears in some Michigan districts to separate the Lower and Middle groups, and in others the Middle and Upper groups. Thus it happened that the great iron-bearing series of the Marquette Range fell in the Lower Huronian and those of the other ranges in the Upper Huronian. The Negaunee iron-bearing series of the Marquette Range was set apart in the classification from similar series in neighboring districts until 1913, when the writer determined its equivalence with the iron-bearing series of the Gwinn district.[10]

It will be recalled that the setting apart of the Negaunee iron-bearing series in 1904 as the new Middle Huronian still left it without a correlative in the Lake Superior region. Since 1913, the unconformity within the Animikie has been traced to the northern Crystal Falls district (1914), Gogebic district (1914) and Menominee district (1917), and in each of these districts the iron-bearing series of the old Animikie now apparently constitutes the Middle Huronian group, with an unconformable sedimentary series above it and another below it; that is to say, its stratigraphic position is apparently identically that of the Negaunee iron-bearing series of the Marquette and Gwinn districts. The Middle Huronian group, therefore, seems to be present throughout the Huronian areas of Michigan, and carries the distinctive members of the old Animikie, the main iron-bearing series. Whether the name Animikie shall be retained for the portion remaining as the Upper Huronian, or shall follow its most distinctive members into the Middle Huronian, may be left for later decision.

The accompanying figure is a generalized graphic representation of the Huronian succession in Michigan, so far as it is known at this time.

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[9] Statement based on conversation with Mr. Hotchkiss.

[10] *Jnl. Geol.* (1914) **22**, 560–73.

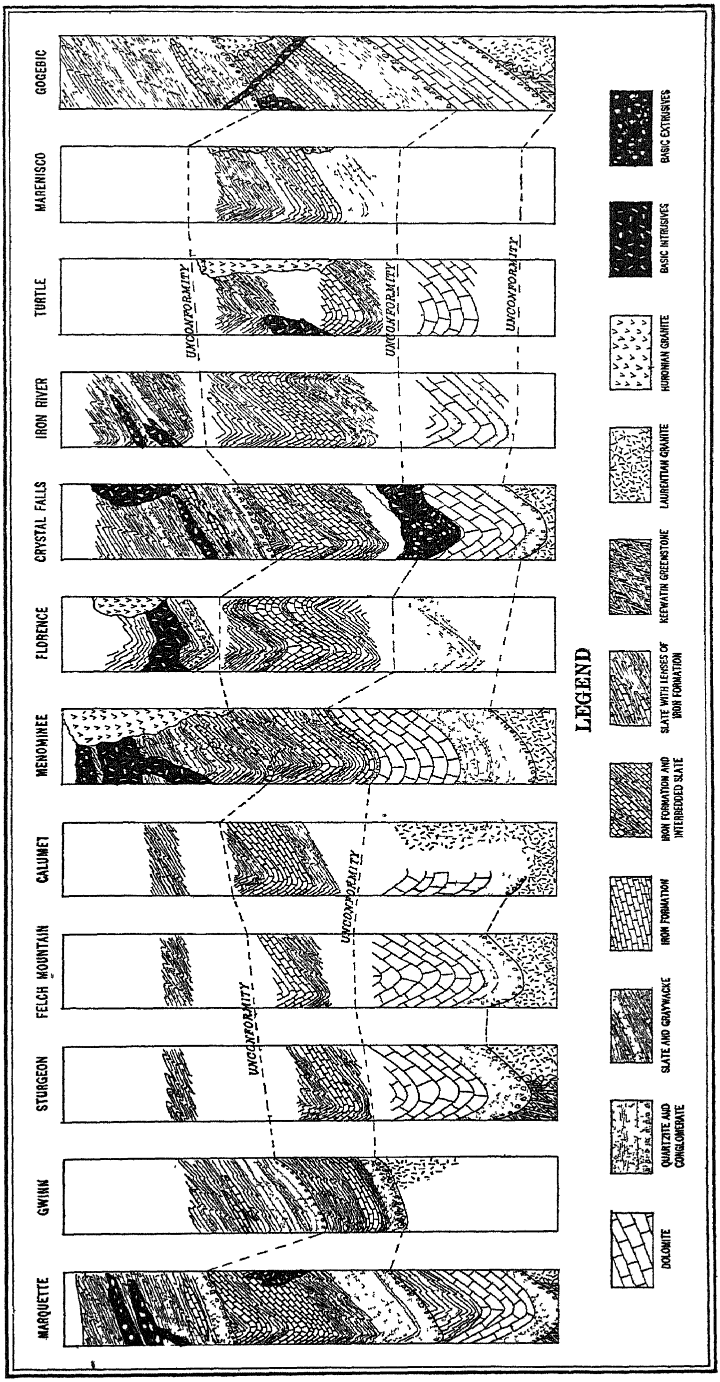


FIG. 1.—GENERALIZED SECTIONS OF THE HURONIAN IN MICHIGAN.

## Middle-upper Huronian Unconformity in Menominee Iron-bearing District of Michigan

In 1904, W. S. Bayley, under the supervision of Charles R. Van Hise, summarized the literature of the Menominee district and described the geology in great detail.[11] In 1911, Van Hise and Leith greatly condensed and to some extent revised Bayley's descriptions.[12] These writers present the following succession and correlation of the pre-Cambrian formations.

Archean System

| | | |
|---|---|---|
| Keweenawan Series | | Granite |
| *Eruptive Contact* | | |
| Huronian Series. | Upper Huronian | Quinnesec schist—basic eruptives<br>Hanbury slate, including iron-bearing beds<br>Vulcan formation, divided into Curry iron-bearing member, Brier slate and<br>Traders iron-bearing member. |
| | *Unconformity.* | |
| | Middle Huronian | Quartzite, not separated from Randville dolomite in mapping for most of the district |
| | *Unconformity* | |
| | Lower Huronian | Randville dolomite<br>Sturgeon Quartzite |
| | *Great Unconformity* | |
| Laurentian Series | | Granite and gneisses, cut by granite and diabase dikes. |
| *Eruptive Contact* | | |
| Keewatin Series | | Green schists (not separated in mapping from Laurentian) |

The above correlation departs from that of Bayley in three important particulars: (*a*) Bayley classified the Quinnesec eruptive green schists with the Keewatin, (*b*) the granite which intrudes them (Keweenawan in the above table) with the Laurentian, and (*c*) did not recognize a Middle Huronian. The true relations of the Quinnesec schists were determined by W. O. Hotchkiss in 1910.[13] In the same year, C. K. Leith placed the cherty quartzite-conglomerate, which lies on the Randville dolomite at Iron Hill, in the Middle Huronian, but after the publication of this view[14] he withdrew it[15] and returned to Bayley's classification of the Huronian sedimentaries.

[11] U. S. Geol. Survey *Monograph* 46. [12] U. S Geol. Survey *Monograph* 52.
[13] U. S. Geol. Survey *Monograph* 52, 344–5.
[14] U. S. Geol. Survey *Monograph* 52.
[15] Verbal communication to the writer.

It will here be shown: (1) that the Upper Huronian or Animikie, of Bayley, Van Hise, and Leith, is divided by a major unconformity representing a period of emergence and deep erosion; (2) that this unconformity lies at the base of the great Hanbury slate series; (3) that a part of the Hanbury, as identified by Bayley, is below this unconformity.

The existence of this unconformity was considered by Bayley in 1904, but after a careful review of the evidence on both sides of the question, he decided against it. The opposite view was advanced by Lane and Seaman in 1908[16] without much discussion of the evidence. The writer considered the subject in 1915[17] and was content to accept Bayley's interpretation of the evidence pending further field studies. In the fall of 1916, several weeks were devoted to a reëxamination of the east end of the Menominee range, and an examination of critical localities so firmly inclined the writer to the opinion of Lane and Seaman that he classified the Hanbury series as "Undifferentiated Middle-Upper Huronian" on a new geological map of Michigan[18] as an indication of his opinion that part of the Hanbury may be Upper Huronian and a part Middle Huronian. The results of recent explorations, which will be described, have decided the question beyond further doubt.

*General Description of the Huronian Sedimentary Formations*

For a complete description of the Huronian sedimentaries, the reader is referred to Bayley.[19] We will here discuss them briefly for the reader who is not familiar with the literature of Huronian correlations.

The Huronian series of the Menominee Range lies on the Archeozoic complex, from which it is separated by a great unconformity representing an interval of enormous duration. It comprises three separate series of sedimentary formations:

*The Lower Huronian group* contains two great, strongly contrasted formations. The lower one, the Sturgeon formation, is 1000 to 1250 ft. (305 to 380 m.) thick. It has a basal conglomerate composed of fragments of the Archean "Basement Complex," mainly granites and gneisses. Above the conglomerate, and associated with it, are arkoses and graywackes, and these in turn grade into the main phase, a massive quartzite. The Sturgeon formation is conformably overlain by the Randville, which is mainly cherty dolomite, interbedded with quartzose and talcose facies. It is about 1000 to 1500 ft. (305 to 457 m.) thick.

*The Middle Huronian group* is composed almost entirely of iron-bearing and slate beds, which were grouped by Bayley under the name

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[16] A. C Lane and A. E. Seaman: Michigan Geol. Survey *Annual Rept.*, 1908.

[17] C. K. Leith and R. C. Allen: Discussion of correlations, *Jnl. Geol.* (1915) **23.**

[18] Michigan Geol. and Biol. Survey *Publication* 23.

[19] U. S. Geol. Survey *Monograph* 46.

Vulcan formation; it overlies the Lower Huronian group unconformably. The lowest member is the Traders, an iron formation with a thin detrital base which is in some places conglomerate, in others quartzite, and in still others slate; in some localities the basal beds contain two or all of these facies. The Brier slate overlies the Traders member. It is a well banded, siliceous, ferruginous rock grading into the Traders below and into the Curry iron-bearing member above. The Curry member is in turn overlain by slate of variable thickness, up to probably several hundred feet, which was classified by Bayley with the Hanbury formation. The thickness of the Vulcan group, from its base to the top of the Curry, averages about 650 ft. (198 m.), the Brier slate 330 ft. (100 m.), and the Curry member 170 ft. (52 m.).

*The Upper Huronian Group.*—The Hanbury formation is in unconformable contact with both the Lower and the Middle Huronian groups. It is composed mainly of normal clay slate interbedded with layers and lenses of quartzite, graywacke, calcareous slate, graphitic slate and ferruginous, siliceous beds. The basal member is conglomerate carrying fragments of both of the underlying groups. The thickness of Hanbury is not known but it is doutbless several thousand feet and may exceed 5000 ft. (1524 m.).

The Quinnesec schists, volcanic extrusive rocks of great but unknown thickness, succeed the Hanbury formation and are to a certain extent interbedded with it. These rocks are intruded by a great granite mass lying south of the Menominee river in Wisconsin. Basic dikes, probably equivalent to the Quinnesec schists in age, intrude the Hanbury and younger formations.

## *Chronology of the Huronian Period*

All observations up to this time confirm the structural parallelism of the Huronian sedimentaries as described by Bayley. Their present distribution is therefore ascribed most largely to deformation in the Epi-Huronian interval followed by erosion terminating in the late Cambrian submergence. Post-Paleozoic erosion has not yet quite succeeded in re-exhuming the pre-Cambrian topography of the range and has had no appreciable effect on the distribution of the Huronian rocks. However, we shall see that erosion in the Epi-Middle Huronian interval has had a conspicuous effect, although the emergent interval separating the Lower and Middle Huronian periods has left no discernible traces in the present distribution of the rocks.

The Middle Huronian rocks are everywhere in contact with the Randville dolomite of the Lower Huronian. We conclude, therefore, that the Middle Huronian sea submerged a land surface formed exclusively of this formation, and that erosion in the Epi-Lower Huronian interval succeeded nowhere in cutting through it although it may have removed

completely other formations of which no trace remains. Bayley thought that the Lower Huronian originally contained an iron-bearing formation younger than the Randville dolomite, because pebbles of jasper occur in the basal conglomerate of the Middle Huronian. Similar pebbles occur in the basal horizons of other Middle Huronian iron formations, notably the Ironwood of the Gogebic and the Negaunee of the Marquette Range.

So far as may be inferred from the facts of observation it appears that the Epi-Lower Huronian interval was one of quiescence during which erosion worked slowly downward on an unwarped surface of Lower Huronian rocks. At the close of the interval the land surface was formed of Randville dolomite rather unevenly eroded into valleys and ridges with a maximum relief of several hundred feet, as shown by the progressive overlapping of the Middle Huronian formations.

The transgression of the Middle Huronian sea, as explained by Bayley, seems to have taken place gradually and very slowly. In the early stages the sea was dotted with islands but these were gradually submerged and finally disappeared. This interpretation rests on the observation of the manner in which the different members of the Vulcan group (Middle Huronian) come into contact with the Randville dolomite. Where one of them is absent it is always the youngest, or Traders; where two are absent they are the Traders and Brier, and where only one is present it is the Curry. Where the entire group is absent the Hanbury slate is in contact with the Randville formation. It was this phenomenon which led Bayley into the error of misinterpreting the relations of the Hanbury slate to the older rocks.

The Epi-Middle Huronian interval, like the one which preceded it, was unaccompanied by marked crustal disturbance. There is no measure of the extent of erosion during this interval. If the old Randville hills were ever completely buried by Middle Huronian sediments, which is extremely probable, they were at least partly uncovered during this emergent interval, for the total length of the Hanbury-Randville contact is apparently not less than 30 mi. (48 km.); however, it appears that the major distribution of the Huronian rocks in this district is due to faulting rather than folding, as Bayley thought, and that about half of this is fault contact.* But it is safe to conclude, on the basis of composition of the Hanbury basal conglomerate, to be described, that all of the Middle Huronian formations were cut through and in places probably obliterated by erosion in this interval.

*The Middle-Upper Huronian Unconformity.*—The Hanbury sea submerged a surface which was formed in part by the Middle Huronian (Vulcan) rocks and in part by the Lower Huronian Randville dolomite. Bayley considered two alternative explanations for this: (*a*) that the

* In a later publication the writer will discuss the structure of this range.

Vulcan sea was dotted with islands of Randville dolomite, which were only gradually submerged and not entirely covered until after the beginning of deposition of the great Hanbury slate series, by which, in the end, they were buried; (*b*) that erosion had removed the Vulcan formations in some places during an emergent interval preceding submergence by the Hanbury sea. By adopting the former explanation, he escaped the consequences of the latter, viz., the recognition of an unconformity at the base of the Hanbury series.

### *Relations Between the Hanbury Slate and Randville Dolomite*

It was just said that the total length of the Hanbury-Randville contact is probably not less than 30 mi. However, there are only two localities where this contact is exposed. For this reason, if for no other, these places possess unusual interest, but they are especially important in this discussion because Bayley described them with minute accuracy, in full realization of their bearing on the age of the Hanbury series, and naturally hinged his conclusions concerning the relations of the Hanbury series to the older rocks largely on the interpretation of these contacts. We can do no better than introduce at this point Bayley's descriptions of these localities.

*Bryngleson Shaft.*—At only one point has the actual contact between the Randville dolomite and Hanbury slate been seen. This is in a trench about 10 ft. long in graphite slates near the east line of Sec. 2, T. 39 N., R. 30 W., a few rods west of the Bryngleson shaft. A careful examination of the relations between the dolomite and slate was made, with special reference to their bearing upon unconformity and faulting. It was found that the dolomite projects slightly into the slates half way up the exposed portion of the contact, and recedes from them both above and below this point. The surface of the dolomite is minutely irregular, small projections and reentrants occurring throughout the entire line of contact. The slates, which are strongly graphitic, are interlaminated with cherty bands. They contain small fragments of dolomite and are badly shattered. A slate breccia is thus formed, which might be a fault or a brecciated conglomerate. There can be no doubt that there has been movement along the contact zone, for the bedding of the slate has been much disturbed for a distance of 8 ft. (2.4 m.) or more from the dolomite. Whether or not the movement was along a fault plane which cut out the Vulcan formation was not determinable from the exposures. The dolomite along the contact plane is not slickensided, nor is the rock near the contact greatly mashed, so far as could be observed. This may be thought to indicate that the movement was of slight magnitude, and that it was more in the nature of a differential movement of the slates near the contact than of faulting across the beds. If the absence of the iron bearing beds between the Hanbury slate and the dolomite is due to overlapping of the slates rather than to faulting, the lower layers of the slate should be coarse detritus, since the relation of the slate to the underlying rocks are those of a younger sedimentary series to an older series upon which the younger series is unconformable. The breccia between the slate and the dolomite referred to above may be a mashed conglomerate of this kind.

Bayley does not mention the possibility that this contact may represent an unconformity and is unable to decide between the alternatives of faulting on the one hand and overlapping of the Hanbury slate on the other. In the opinion of the writer, the Bryngleson shaft is on a fault plane and the locality has no bearing on the age relations of the Hanbury slate series.

*At Iron Hill*—At Iron Hill, in Sec. 32 T. 40 N., R. 29 W , the Hanbury slate is believed to lie immediately on the Randville dolomite although no contact between the two is seen. The dolomite ends in a number of small knobs having steep faces toward the south At the bases of the little cliffs is a swamp about 300 ft. (91 m.) wide and upon the opposite side of the swamp, on the north slope of a slight elevation, are several exposures of slate. The intervening swamp area has been tested at a number of places by augur borings, . . . and has been found to be underlain by slates.

The uppermost layers of the dolomite formation consist of white cherts, and these are beautifully brecciated. Below these in some places lies a conglomerate containing numerous large rounded boulders of dolomite, subangular fragments of chert, and an occasional pebble of quartzite in a matrix composed mainly of dolomite and chert debris Many of the pebbles are mashed and faulted, showing that the district was deformed after the conglomerate was laid down. Where the conglomerate is traced eastward to the end of the set of dolomite ledges, the relations between this rock and the chert are found to be very complicated. The conglomerate apparently grades into a breccia, and this in places is between beds of dolomite or layers of chert. At one place the conglomerate looks as though it were a breccia formed by crushing of the chert and dolomite; at other places it appears to be a layer of true conglomerate between layers of massive dolomite, and in other places it strongly resembles a conglomerate composed of fragments of the dolomite and chert lying above the dolomite. The conglomerate may be an intraformational conglomerate (one originally produced during the deposition of the formation, and therefore an integral part of it) whose complex relations to the remainder of the Randville dolomite are due to crushing and close folding; or on the other hand, it may be a true conglomerate at the base of the Hanbury slate, made to appear like an intraformational conglomerate by repeated close folding at the end of an eastward pitching anticline on which are superposed several minor folds. *The latter is thought to be the probable explanation.** If this be correct, the difference in composition of the conglomerate from the normal slates is explained by its being the first deposit along a shore line composed of dolomites and cherts. But the relations of the various rocks at this place are so exceedingly complicated that no unprejudiced observer would be willing to declare without reservation that the conglomerate is not a member of the dolomite formation, rather than the basal member of the Hanbury slate.

It is not surprising that Bayley reached no definite conclusion concerning the meaning of the phenomena which he describes so well in the above paragraph. The relations of the conglomerate here to the Randville dolomite below and the Hanbury slate above were puzzling to C. K. Leith and assistants, in 1910, and to the writer, on repeated visits since then. The discovery of the true relations of the Hanbury to

* Writer's italics.

older formations in 1917, by observations in another locality, is a strong confirmation of Bayley's belief that this conglomerate is the basal member of the Hanbury slate. However, this conglomerate and the apparent field relations of the Hanbury and Randville formations, as Bayley has said, are not proof of an unconformity at the base of the former even though they point strongly in that direction. But had Bayley been aware, at the time of this writing, of Seaman's discovery of the true stratigraphic position of the Negaunee iron-bearing series, it is probable that he would have attached more weight to the evidence, inconclusive though it was, indicating the same position for the Vulcan series of the Menominee district and confirming the tripartite division of the Huronian there as well as in the Marquette Range.

### *Relations Between the Hanbury Slate and the Vulcan Series*

Bayley says:

> The relations between the Vulcan formation and the overlying Hanbury slates are also those of conformity. The contact is usually very sharp. Little difficulty is experienced in defining the upper limit of the iron-bearing formation where exposures are plentiful. The slates, however, are often so very schistose on the upper side of the contact that their bedding planes cannot be recognized. In most places the overlying slates are strongly graphitic and often very fissile. Where not graphitic they are light silvery schists from which all bedding traces have disappeared. The bedding of the iron formation, on the other hand, is still almost perfectly preserved, and is parallel to the contact In one or two places the iron-bearing beds are separated from the slates by narrow bands of a soft earthy green rock containing remnants of plagioclase and a mass of green chloritic products such as usually result from the decomposition of a basic eruptive. The green rock has the appearance of a dike, which it probably is, that intruded the bedded series along the contact plane.
>
> At a few other places, as in the neighborhood of the Pewabic mine, faults may intervene between the iron-bearing formation and the overlying slate. They are apparently of little importance from a structural point of view but have not yet been sufficiently exposed to warrant any very definite statements as to their extent or position with reference to the adjacent rocks.

The above is an accurate description of the relations of the Vulcan and Hanbury formations so far as they may be inferred from a study of naturally exposed contacts and those which have been crossed by underground mine openings. But recent diamond-drill explorations have disclosed a remarkable conglomerate at the base of the Hanbury slate series, the discovery of which proves beyond any question that the periods of deposition of the Vulcan and the Hanbury series were separated by an interval of emergence and erosion. It also illustrates the extent to which a great unconformity may be obscured in highly deformed pre-Cambrian successions by the fortuitous exposure of only those relationships which are susceptible of two or more interpretations.

In 1915, E. E. White, who was then Chief Geologist of the Cleveland Cliffs Iron Co., stated to the writer that he had observed in drilling for iron ore on the Menominee range a conglomerate at the base of the Hanbury slate containing boulders and pebbles derived from the Vulcan slate-iron formation group and from the Randville dolomite, and that he considered this conglomerate to be complete evidence of an unconformity separating the Hanbury from the older formations. The first evidence confirming the observations of Mr. White was secured by the writer in the fall of 1916 when examining drill cores obtained by the Penn Iron Mining Co. from explorations in various localities on the "South Range" between Sturgeon River and Waucedah. In the summer of 1917, explorations conducted by C. H. Baxter, for the Loretto Iron Co., on the "South Range" just east of Sturgeon River, confirmed beyond doubt, the conclusions founded on the Penn Iron Mining Co. explorations.

*Explorations East of Sturgeon River.*—In Sections 13, T. 39, R. 29, and in Sections 17 and 18, T. 39, R. 28, on the "South Range," east of the Sturgeon River, a large number of diamond-drill holes penetrate the lower horizons of the Hanbury slate, the entire Vulcan group, and end in the Randville dolomite. The drill cores have been preserved and may be examined in the offices of the Penn Iron Mining Co. and the Loretto Iron Co. Half of them exhibit no evidence of unconformity between the Hanbury slate and the Vulcan group, a few show coarsely detrital beds at or very near the base of the Hanbury, and three show a well developed Hanbury basal conglomerate lying on a slate which is conformable with the Curry member of the Vulcan group. So far as these explorations have gone, it appears that the conglomerate is best developed in the S.W.¼ of the N.E.¼ of Section 13. Near the center of this description the Curry iron-bearing member of the Vulcan group is overlain conformably by a dark, distinctly banded, gray slate about 50 ft. (15 m.) thick. Lying on this slate is the basal conglomerate of the Hanbury formation. In one hole it was entered at rock surface by the drill and penetrated for a distance of 78 ft., corresponding to a stratigraphic thickness of about 65 ft. (20 m.); how much thicker it may be remains to be determined. It is composed of fragments of white dolomite, chert and quartz derived from the Lower Huronian dolomite (Randville) intermixed with red, black, and brown jasper, ferruginous slate and quartzite, in which are represented all of the members of the underlying Vulcan group of the Middle Huronian. Polished sections of the drill core exhibit a mosaic of great beauty formed by the heterogeneous mixture and strongly contrasting colors of the fragments from the different formations. The pebbles in this core show diameters varying up to 2 in. (5 cm.) and sections which are characteristically subangular to angular. The matrix is composed of the same material as the pebbles,

but more finely comminuted. There are thin beds and bands in which no pebbles are shown and some which are even slaty textured, but the main mass of the formation is conglomerate. Two other drill holes in this locality penetrate this conglomerate but in neither of them is it so strongly developed as in the one described above.

*Loretto Slate.*—The slate which lies conformably on the Curry iron-bearing formation has not been recognized heretofore as a distinct member of the Vulcan group. The name Loretto is proposed for it as a local term because it is best developed, so far as known, on the property of the Loretto mine. Bayley, of course, included it in the Hanbury formation. Before it may be discriminated everywhere, a reëxamination of the whole range will be necessary, including underground mine workings. There are many places where the Hanbury slate is in undoubted contact with the Curry formation in apparent conformity. There are doubtless also places where the Loretto slate is in apparent conformity with the Hanbury. Both of these relationships seem to be indicated in the explorations east of Sturgeon River.

The full thickness of the Loretto slate is probably nowhere preserved. At the Loretto mine it has a thickness of at least 400 to 500 ft. (122 to 152 m.) and resembles very closely the Brier slate, so closely in fact that it was mistaken for the Brier formation by the writer and others after careful examination. This mistake determined an erroneous conception of the structure of the mine, which was only corrected after the discovery of the Loretto slate as a distinct member of the Vulcan group in the Sturgeon River explorations described above.

### *The Epi-Huronian Revolution*

The Huronian period was brought to a close by a great outburst of volcanic activity and crustal disturbance of mountain-making order. A vast thickness of basic lava was poured out on the Hanbury slates and thrust into them in the form of dikes and sills (Quinnesec schists). Later, the country south of the Menominee range was invaded by great masses of granite which have been traced far to the southwest in Wisconsin; this granite intrudes the Quinnesec schist but its age is not definitely ascertainable. Bayley regarded the granite and green schist complex south of the range as Archeozoic, on the basis of its resemblance to the Basement Complex, but in 1910 Hotchkiss found that the outbreak of the basic volcanics began in Hanbury time and continued until after its close. The granite is therefore post-Hanbury, *i.e.*, later than the youngest Huronian sedimentaries. In 1911, Van Hise and Leith classified it with the Keweenawan. In 1915, the writer correlated it with the Middle Huronian granites of the country south of the Gogebic Range in Michigan and Wisconsin. This, of course, was prior to the discovery of the uncon-

formity at the base of the Hanbury slate. It seems now more logical to correlate this great granitic invasion with the Epi-Huronian revolution, rather than with the Keweenawan. It is not improbable that it was the source of the great thrusts which, directed from the South, built the mountains whose stumps remain in the folded and faulted Huronian rocks of the Menominee Range.

## *Pre-Cambrian Chronology*

The revised chronology of the pre-Cambrian of the Menominee Range is presented in the following table:

*Pre-Cambrian Chronology—Menominee District, Michigan,* 1919

| Eras | Sub-eras | Periods | Epochs | Stages | Description of Formations |
|---|---|---|---|---|---|
| Proterozoic | Algonkian | Keweenawan | | | |
| | | Epi-Huronian Revolution. Mountain building | | Emergent Interval | Granite |
| | | Huronian | Upper Huronian | Quinnesec | Eruptive contact. Basic volcanic extrusives of great thickness, sills and dikes |
| | | | | Hanbury | Great slate series with beds of conglomerate, quartzite, graywacke, ferruginous chert and impure limestone. Thickness not known. |
| | | | Emergent Interval | | |
| | | | Middle Huronian | Loretto | Slate, 400 + ft (122+ m). |
| | | | | Curry | Iron-bearing member, 100 to 200 ft (30 to 61 m). |
| | | | | Brier | Ferruginous, siliceous, banded slate, 300 to 400 ft (91 to 122 m). |
| | | | | Traders | Conglomerate, quartzite and iron-bearing formation 150 ft (46 m). |
| | | | Emergent Interval | | |
| | | | Lower Huronian | Randville | Dolomite, cherty dolomite and quartzose and talcose facies, 1000 to 1500 ft. (305 to 457 m.). |
| | | | | Sturgeon | Conglomerate, arkose, graywacke and quartzite 1200 ft (366 m). |
| | Great Archeozoic Interval | | | | |
| Archeozoic | Laurentian | | | | Granites and gneisses cut by dikes of granite and diabase. |
| | Keewatin | | | | Eruptive Contact. Green Schists. |

## DISCUSSION

ALFRED C. LANE, Tufts College, Mass. (written discussion*).—The attention of members may well be called by Allen to his discoveries, which affect not only correlations in Michigan but, as I have pointed out,[20] the use of names like "Animikie" which are widely applied. It shows that one disadvantage of the wide use of locality names is that local discoveries may show that they are inappropriate. The Wewe hills were found not to contain the Wewe slate! So we note that Allen, in his correlation, drops the term Animikie, which his researches indicate covers both middle and upper Huronian.

It would seem well, then, in dividing the Huronian or Proterozoic, to use, as Allen does, terms like upper, middle, and lower Huronian, or newer, middle, and early, which may be shortened into eo-, mio-, and neo-Huronian rather than to carry local terms far away; or we shall soon be in the condition of the zoologist, who has to add to a name the author whose usage he follows.

Comparing his correlation with previous attempts, as that of Lawson,[21] we find general agreement so far as the heavy dolomite of the eo-Huronian, so widespread and thick that its correlation is fairly easy. If, indeed, it marks the first efflorescence of lime precipitating vegetation, as the Carboniferous marks the first great efflorescence of higher vegetation, it may possibly be correlated throughout the world.

The heavy iron formations Allen places in the mio-Huronian, but they are supposed to be equivalent to the Animikie iron formations of the north side of Lake Superior. If this period marks the first great efflorescence of the iron precipitating forms, such as those recently described,[22] it, too, may be traced over the world. Allen makes, however, within the mio-Huronian two iron formations, the Traders and the Curry. A recent letter from Iron Mountain tells me that the existence of a separate Curry member is still doubtful.

There is one thing that makes it hard to be sure of the stratigraphy, worth remembering: block faulting and block fault mountains, if followed by a period of folding and granitic intrusion, are very hard to recognize. They are even more hard to prove and may produce apparent repetitions that are not real. It is noteworthy that the early Lake Superior monographs paid little attention to faults in the Huronian. It was in the "zone of flow." And yet to get there and back to the surface it must

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* Received Sept. 19, 1919.

[20] *Science* N. S. (1915) **42**, 869 and *Am. Jnl. Sci.* [4](1917) **43**, 42.

[21] A. C. Lawson: University of California Publications, *Bulletin* of the Department of Geology (1916) **10**, 1–19, reviewed by me in the *Am. Jnl. Sci.* [4](1917) **43**, 42, with his correlation plate marked over in free hand to show suggested changes.

[22] U. S. Geol. Survey *Prof Paper* 113.

twice go through the zone of fracture. Allen rightly recognizes much more faulting. So does Collins.[23] Still more will be found.

It is noticeable that Lawson's term Algoman, as applied to a mountain building interval between the neo-Huronian and the mio-Huronian, or as he calls them Animikian and Temiskamian, is not used by Allen. Now it is true that a number of the granites which Lawson placed with the Algoman, the Killarney, and Moira, for instance, belong later and Collins thinks them Keweenawan. Allen seems to put this group of granites later than the youngest Huronian sedimentaries, just before the Keweenawan. There are, however, at least two times in the Keweenawan when wide extended effusions of porphyries suggest the stirring of granitic magmas on a large scale, and one is reminded of the conditions around the harbor of Brest in France, where Barrois[24] points out that the effusives south of the harbor, including Kersantite dikes and a luster mottle ophite much like the Keweenawan ophite and the kersantites "are limited to Carboniferous synclinal regions; they are replaced, in the neighboring anticlinal regions, by massifs of grained (plutonic) rocks which represent the profound reservoirs of them." However, it may be that the relations of these granites and the Keweenawan are like those of the Late Carboniferous granites and the Triassic of the Atlantic coast. It must be remembered that I believe the Keweenawan is more or less Cambrian, and the discovery by Watson and Cline of amygdaloidal basalt that might well be Keweenawan in the Lower Cambrian of Virginia[25] at least weakens the argument of Van Hise and Leith[26] that the Keweenawan is pre-Cambrian because the Cambrian is "lacking volcanism."

But, I should not be at all surprised to find the Keweenawan composite, as the "Redbeds" have been found to be out West, yet it seems hardly possible to think that all the granites which Lawson classed as Algoman can be crowded into the Keweenawan or immediately pre-Keweenawan. The Presque Isle granite, which is pre-"Copps," must be placed in the "emergent interval" between the Hanbury and the Loretto, must it not? In that case the two "emergent" intervals separating the Huronian, while they may not be marked by granite intrusion and mountain building in the Menominee district may have been, and I believe were marked by these actions in other districts—actions that produced the widespread eustatic change of sea level, registered in the Menominee district. I am inclined to think that at least as near as Humboldt and Republic one will

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[23] W. H. Collins: Geol. Survey Canada, *Mus. Bull.* 22 (1916) and private communication from him and Morley.

[24] Ch. Barrois: *Guide Géologique de France*, Part VII (Bretagne) Internat. Geol. Congres (1900) 17.

[25] U. S. Geol. Survey *Monograph* **52,** 415, line 5.

[26] T. L Watson and J. H. Cline: Extrusive Basalt of Cambrian Age in the Blue Ridge of Virginia. *Am. Jnl. Sci.* [4] (1915) **39,** 665.

| Era | Period (System) | Epoch (Series) | I Rainy Lake | II Steeprock Lake | III Thunder Bay | IV Gunflint Lake | V Vermilion Lake | VI Mesabi | VII Gogebic | VIII Crystal Falls | IX Menominee | X Marquette | XI Lake Huron | XII Sudbury | XIII Cobalt | XIV S. E. Ontario | XV Adirondacks |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | NORTHWEST OF LAKE SUPERIOR | | | | | | SOUTH OF LAKE SUPERIOR | | | | NORTH OF LAKE HURON | | | SOUTHEASTERN DISTRICTS | |
| Paleozoic Era (Paleozoic) | Algonkian (Period System) | Keweenawan (Epoch Series) | Dykes | | Keweenawan | Keweenawan | Keweenawan | Keweenawan Embarrass granite | Keweenawan | A red land and volcanic formation Granite | | | Dykes Granite (Killarney) | Dykes and Laccolith | Sills | | |
| | Algonkian (Period System) | Animikian (?) (Epoch Series) Neo-Huronian | Allen's correlation* | | Animikian Slates, iron formation, quartzite | Animikian Rove slate iron formation | Animikian iron formation Rove slate | Animikian Virginia slate | Copps Formation Slate graywacke chert, conglomerate Tyler slate volcanics | Michigamme slate Paint slate lavas, Hanbury slate | Quinnesec schist Hanbury slate | Volcanics, Michigamme slate Bijiki I. F., Goodrich quartzite | Cobalt Series | Cobalt Series | Cobalt Series | Granite (Moira) | |
| | | | | | | | | | | | | | | Coleman's glacial conglomerate or tillite | | | |
| | Eparchean Interval | | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Unconformity | | Unconformity | Major unconformity | Major unconformity | Major unconformity | | |
| Archean Era (Colloozoic) | Algoman Revolution | | Granite, syenite, etc. | Granite gneiss | Granite | Granite | Granite (Giants Range) | Granite (Giants Range) | Granite (Presque Isle) | | | | | Granite gneiss | Granite (Lorrain) | | Granite (Picton) (?) |
| | Huronian (Period System) | Temiskamian (Epoch Series) Mio-Huronian | Seine Series Slate quartzite, conglomerate | Seine Series Slate, quartzite conglomerate iron formation | Graywacke slate conglomerate | Graywacke | Knife Lake slate, Agawa I. F. Ogishke conglomerate | Biwabik I. F., Pokegama quartzite Slate graywacke, conglomerate | Ironwood I. F., Palms | Vulcan I. F., Ajibik quartzite, Hemlock volcanics and slate | Vulcan I. F., quartzite Loretto slate | Negaunee I. F. Siamo slate, Ajibik quartzite | Temiskamian | Temiskamian | Temiskamian | Hastings | |
| | | | | | | | | | The great iron series | | | | | First great efflorescence of iron-ore precipitating forms | | | |
| | Huronian (Period System) | Adirondack Revolution | | Unconformity | | | | | Unconformity | Unconformity | Unconformity | Unconformity | Unconformity | Major unconformity Granite | Major unconformity Granite gneiss | Major unconformity Granite gneiss | Major unconformity Granite gneiss |
| | Huronian (Period System) | Bruce (Epoch Series) Eo-Huronian | | Steeprock Series Volcanics, limestone (fossils) quartzite conglomerate | | Early (algal?) limestone | | | Bad River limestone Sunday quartzite | Randville dolomite, Sturgeon quartzite | Quartzite Randville dolomite, Sturgeon quartzite | Wewe slate Kona dolomite, Mesnard quartzite | Bruce Series Quartzite limestone, graywacke limestone conglomerate quartzite | Grenville limestone metam. | Efflorescence of carbonate precipitating forms Grenville | | Grenville |
| | Epilaurentian Interval | | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | Major unconformity | | | | |
| | Laurentian Revolution | | Granite gneiss | Granite gneiss | Granite gneiss | Granite gneiss | Granite gneiss | Granite | Granite gneiss | Granite gneiss | Granite gneiss | Granite gneiss | Granite gneiss | | | | |
| Azoic | Ontarian (Period, System) | Grenville (Epoch Series) | | | | | | | | | | | | | | | |
| | Ontarian (Period, System) | Keewatin (Epoch Series) | Keewatin | Keewatin | Keewatin | Keewatin | Keewatin | Keewatin | Keewatin | Green stone schists | Keewatin | Keewatin | Keewatin | Keewatin | Keewatin | Keewatin | |
| | Ontarian (Period, System) | Coutchiching (Epoch, Series) | Coutchiching (?) | | | | | | No original red formations, nor great chemical concentration | | | | | | | | |

* Allen's work in Michigan suggests the very distinct possibility that the Animikian of the north shore is equivalent in part to the Middle Huronian of the south shore. The equivalence of the Ironwood Tyler of the Gogebic range and the Biwabik Virginia series of the Mesabi range has not yet been seriously questioned.

MODIFICATION OF LAWSON'S CORRELATION OF THE PRE-CAMBRIAN

find granites and basic effusives and dikes corresponding to these periods of disturbance, and I think that in the Menominee district, also, dikes and faulting belonging to these periods rather than to the epi-Huronian will be found. Lawson's correlation table would then be marked up somewhat as here shown.

W. O. Hotchkiss,* Madison, Wis.—When we began to do geological field work in the Lake Superior region, we found a correlation in existence, and, as youngsters, accepted all that as settled. Both Mr. Allen and I, since that time, have seen a great many things that were not apparent nor visible to Van Hise, and those who worked with him; we realize now that the correlation was not final and that much remains to be determined concerning geology of the Lake Superior region.

My attitude on this matter is that we should not publish an amended correlation as a final thing. We should avoid, in every way possible, foreclosing the minds of those who follow us in working in the geology of this region. As a matter of fact, no correlation can be final at present, because facts and data are accumulating so rapidly that every classification must be immediately modified to fit further observation. Consequently, my attitude in this matter would be, instead of saying that the iron formations of the Lake Superior region are all Middle Huronian, to say that there may be three Huronians—or four or five—we do not know.

I believe it is better policy to hold to our present correlation. I do not believe there is any economic benefit to the mining man or the geologist working on problems relating to explorations for iron ore that cannot be just as well solved by maintaining our present correlation, with a wholesome idea of our lack of knowledge and the necessity for further detailed work. Then, if any new horizons are discovered in any particular range, the minds of the investigators will not be foreclosed because some one has said that all the iron formation is in the Middle so that there is no use looking elsewhere for it. There is use looking elsewhere, in many cases, as the ore deposits become exhausted.

I am, at the present time, carrying on some very detailed investigations on the Wisconsin end of the Gogebic Range, and find throughout the whole extent of that range, in our state, unconformities marked by definite conglomerates, which can be picked up on nearly every exposure of the contact. Three are within the iron formation itself. Mr. Allen, by inference, has classed those as minor unconformities. They may be; but as we learn more about them they may become major unconformities, because the recognized great unconformity marking the hiatus between the Huronian and the Keweenawan took place without any material disturbance of the Huronian strata during that erosion period.

* State Geologist.

I am very unwilling to complicate our geological literature with something for which Mr. Allen himself does not claim finality. I want to emphasize rather the necessity for keeping our minds open. In every range we can divide the present Upper Huronian and the Lower Huronian. We can divide these into as many members as we please; we can say there are one or two, three or four in the Upper Huronian of the Gogebic Range.

My position is that we do not yet know enough to make that correlation; and, furthermore, it serves no valid economic purpose that cannot be served by the other plan, because these iron ranges are almost entirely separate from each other in areal extent.

E. F. BURCHARD,* Washington, D. C.—With regard to Mr. Allen's quotation that Harder and Johnson thought it "premature to change radically a correlation based on years of careful field work," it should be stated that the United States Geological Survey, in its pioneer work under the direction of Van Hise, aimed to point the way and leave the more detailed work to the State geological surveys and to the iron companies. The United States Geological Survey is not now in a position to carry on such extensive work as it was when the lake region was first studied. Plans were made, however, for correlation studies to be carried on this last summer by Mr. Harder, of the United States Geological Survey, in coöperation with the State Geologists of Michigan, Minnesota, and Wisconsin, but other work made it impossible for these plans to be carried out.

R. C. ALLEN.—As I understand Mr. Hotchkiss, he admits that in all but one at least of the iron ranges, or districts, in Michigan, there are two known major unconformities within the Huronian series. When, in 1904, it was found, on the Marquette Range, that instead of two unconformable series as theretofore supposed there are three, the United States Geological Survey properly set up a Middle Huronian to account for the newly discovered distinct and separable series. But the United States Geological Survey is now, in 1919, on record that the Middle Huronian is confined to the Marquette Range and exists nowhere else. What I have shown is that the Middle Huronian is widespread and the criteria of its recognition on the other ranges are precisely of the same order of significance as those by which it was first distinguished and set apart on the Marquette Range. I am not attempting to supplant old work in so far as it is in accord with present facts; therefore I do not understand the position of the State Geologist and certain former members of the United States Geological Survey. Are they going to adhere to a classification which they admit is wrong simply because they think it may be even more seriously in error than it yet appears to be? Will they delay

* Geologist, U. S. Geol. Survey.

a revision merely in the expectation that at some future time a further revision may be necessary? I think it is time that these facts should be taken note of.

W. O. Hotchkiss.—I do not agree with the policy of the United States Geological Survey to perpetuate what I believe to be a wrong policy. So long as we all agree that this correlation situation is in a state of flux and that probably, before many years, we will know more about it than we do now, I believe in letting the present correlation stand until we can correct the whole business to some adequate degree. I believe that, in the Marquette Range itself, there are more than three Huronian Series.

Leslie P. Barrett, Lansing, Mich.—Mr. Hotchkiss has said that the Michigan ranges should be considered as separate provinces.

W. O. Hotchkiss.—I admitted exceptions to that. One of the main facts that led us to the conclusion that there was something wrong about the correlation was the fact of the continual band of iron formation which comes down through the Marquette Range and around the oval in the Crystal Falls District. The United States Geological Survey mapped the basin on one side as Upper Huronian and on the other as Middle Huronian. Somewhere underneath this narrow syncline there was supposed to be a change.

Carl Zapffe, Brainerd, Minn.—I have always been willing to take as a guide whatever correlation was presented, that is, in any new field I happen to enter, but I keep my mind open, ready to find something new; and when I find something new I adopt it.

I began my work in the Cuyuna District ten or twelve years ago. The conditions I found did not seem to correspond with any I knew in the Lake Superior region, but because of my inexperience I did not dare upset the existing classification. The formations in the Cuyuna District were then known as Upper Huronian, but even by any other name it would have made little difference in our work; we were looking for iron ore and we followed the structural conditions, regardless of nomenclature of the formation. The Cuyuna District is disconnected superficially from any of the fields Mr. Allen has cited and yet the formations in the Cuyuna are such, as demonstrated by much mining and drilling, that you may establish a group there probably corresponding with his Copps formation. Rock outcrops between the Cuyuna and Michigan ranges are few and very far apart, and they have several times been raised in classification, from the Keewatin to the Upper Huronian Series. So there is always room for another possible change and Mr. Allen's Copps Formation may not be at all local in Michigan.

By altering classifications, I do not think you prejudice those who follow you any more than I was prejudiced by the teachings and classi-

fication of Van Hise and Leith. I am willing to accept their teachings, but that does not mean that because they have found something to classify it is unnecessary for me to keep my eyes open, and the younger geologists should and will do the same thing; if they do not, the probabilities are that they will go wrong on many other questions. I feel inclined to support Mr. Allen. I believe that the newer classification has advantages, and if Mr. Allen finds extensive and important differences not reported in the existing literature and does not make them known, I think science is the loser.

C. E. SIEBENTHAL, Washington, D. C.—Many years ago I attended some lectures by Professor Van Hise, and understood that, even in those days, there was not unanimity of opinion as to the classification. I gathered that, among others, Prof. A. C. Lawson differed with him. How does the more recent classification agree with the classification proposed by the Canadians at that time?

R. C. ALLEN.—After the publication, in 1914, in the *Journal of Geology*, of a classification which I offered as a substitute for the one existing, Lawson seized upon my classification of great batholithic granite masses of Northern Wisconsin as Middle Huronian, announced their equivalence to his Algoman granites of the north shore and revised the classification of the Huronian of the south shore to accord with his of the north shore on the assumption that his Algoman revolution was coextensive with the whole area. Soon after the publication of Lawson's view, I proved the unconformity on the Menominee range of Dickinson County, described in the second part of this paper, the result being that the granite which intrudes the Great Hanbury Slate series is proved to be later than the youngest Huronian sediments, that is to say, it is not Middle Huronian but Upper Huronian or later. It now appears that the granites of northern Wisconsin and Michigan which intrude the Huronian are of two different ages. There may be more than two periods of granitic intrusion here. It is quite likely that some of the granite is as late as Keweenawan. Lawson based the whole structure of his classification on the equivalence of these granites with his Algoman of the north shore, an hypothesis which is clearly untenable.

We people who are talking about classification need to stay pretty close to facts. Too much should not be hinged on the slender thread of hypothesis. Mr. Hotchkiss and I are not so far apart as we may seem to be. My position is this: When you have new facts that cannot be disputed it is time to account for them in a revised classification, but let us stop treating inferences as though they were facts.

The thing which I want to establish is this: The Huronian correlations of the United States Geological Survey, published in 1911, are in important respects wrong and need revision. Under this conviction,

last winter I invited the Director of the United States Geological Survey to coöperate with us in Michigan, and with the State Geologists of Wisconsin and Minnesota in a review of the present known facts with the end in view of issuing a revised classification of the Huronian that would be useful, at least for the moment, and with the full expectation that in a few years another revision would probably have to be made.

E. C. HARDER, Washington, D. C. (written discussion*).—Persons acquainted with the complexity of Lake Superior geology realize the value of the recent work on the iron-bearing rocks by R. C. Allen and his associates of the Michigan Geological and Biological Survey in the iron ranges of northern Michigan, by J. F. Wolff, of Duluth, and F. F. Grout and T. M. Broderick of the Minnesota Geological Survey, Minneapolis, in the Mesabi Range, and by W. O. Hotchkiss, State Geologist of Wisconsin, in the Penokee-Gogebic Range. One important point brought out by these investigations is the fact that the iron-bearing formations and associated rocks were undoubtedly deposited in shallow water and that during their deposition oscillations frequently brought them above water level. Conglomerate beds of greater or less significance are found at frequent intervals in these rocks. Hotchkiss in a recent paper describes five or six of them in the iron-bearing beds of the Penokee-Gogebic Range alone. Some of these represent unconformities of considerable magnitude, as is shown by the way in which erosion has affected the underlying beds; others are of minor significance. No great importance is attached to any of these unconformities by Hotchkiss, who simply uses them for the division of the iron-bearing layer into minor subdivisions. Similar conglomeratic layers have been found by Grout and Broderick in the Mesabi Range and by Johnson and myself in the Cuyuna Range.

Some of the unconformities found by Allen in his investigations are unquestionably of great significance, particularly that separating the Copps formation from the other Huronian rocks in the Penokee-Gogebic Range. Here a conglomerate successively comes into contact with various older rocks indicating an erosion period of some magnitude. Other unconformities, however, in my opinion have not been shown to be of sufficient importance to attach to them the significance Allen proposes. His description, for instance, of the conglomerate in the Hanbury slate does not appear to preclude the possibility of its being merely of local occurrence rather than representing a great unconformity. By successive overlap the Hanbury slate and the iron-bearing beds of the Vulcan and Traders formations below it come into contact with Lower Huronian dolomite. Why may not pebble beds, the material of which is derived from neighboring dolomite islands projecting up from the Hanbury sea, be found at various horizons in the slate? The occurrence of ferrugin-

* Received Nov. 20, 1919.

ous chert pebbles with those of dolomite might be ascribed to the presence of local ferruginous chert beds within the Hanbury slate. If Allen's basal beds of the Hanbury slate could be found in unconformable contact with the iron-bearing beds of the Vulcan or Traders formations, the significance of the conglomerate would be established.

As the detailed work progresses doubtless other conglomerate beds and unconformities will be discovered at various horizons in the Huronian rocks of the Lake Superior district, as well as further facts bearing on conglomerates already known. In the end a comparative study will have to be made of the relative importance of the various unconformities in relation to the major classification. Such a study should be undertaken by the United States Geological Survey in cooperation with the Geological Surveys of Michigan, Minnesota, and Wisconsin. Until such an investigation is completed, however, the United States Geological Survey must adhere to the classification and correlation established by the work of Van Hise, Leith , and other geologists of the Federal Survey

# The Wisconsin Zinc District

BY W. F BOERICKE,* A B , AND T H. GARNETT,* E M , GALENA, ILL

(Chicago Meeting, September, 1919)

THE Wisconsin zinc district, or the Upper Mississippi lead and zinc district as it is also termed, lies in the southwestern corner of Wisconsin, and embraces adjacent portions of Illinois and Iowa. It includes Iowa, Grant, and Lafayette counties in Wisconsin, and Jo Davies county in Illinois. In Iowa, Dubuque formerly produced considerable lead, but there has been little mining there in recent years.

The map, Fig. 1, shows the entire district, with roads, railways, streams, and the various towns, or "camps," around which the mines are clustered. The hatchwork indicates the areas in which lead and zinc have been mined. The district is about 60 mi. (96 km.) north and south and about 40 mi. (64 km.) east and west. Not all of the territory is mineral bearing, and there are considerable areas that are believed to be barren. The grouping of the mines is patchy and irregular, though certain broad zones of mineralization extend almost the full width of the district, as from Mineral Point west through Mifflin and Livingston, and from Shullsburg west through New Diggings and Hazel Green. It is worth noting that the best mines of the district have, to date, been found within 10 mi. east or west of the fourth principal meridian line, thus lying within a long, narrow, north-and-south belt extending the length of the district.

In a field as large as the Wisconsin, which has only within comparatively recent time been worked intensively for zinc, the chances are good for opening new and important orebodies in parts of the district not yet, or insufficiently, prospected. However, the prospecting has been most successful around, or adjacent to, the old lead diggings of nearly a century ago, and the camps that were well known then are still centers of activity. In their order of importance they are: New Diggings, Benton, Mifflin-Livingston, Galena, Linden, Hazel Green, Shullsburg, and Highland.

The district is one of the best farming districts in Wisconsin, with deep fertile soil. The farms are large and well improved, and the towns are modern and prosperous. Railroad facilities are good and proximity to Chicago is an asset. Mining is usually done on a leasing system of 10

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*Engineer, Mineral Point Zinc Co.

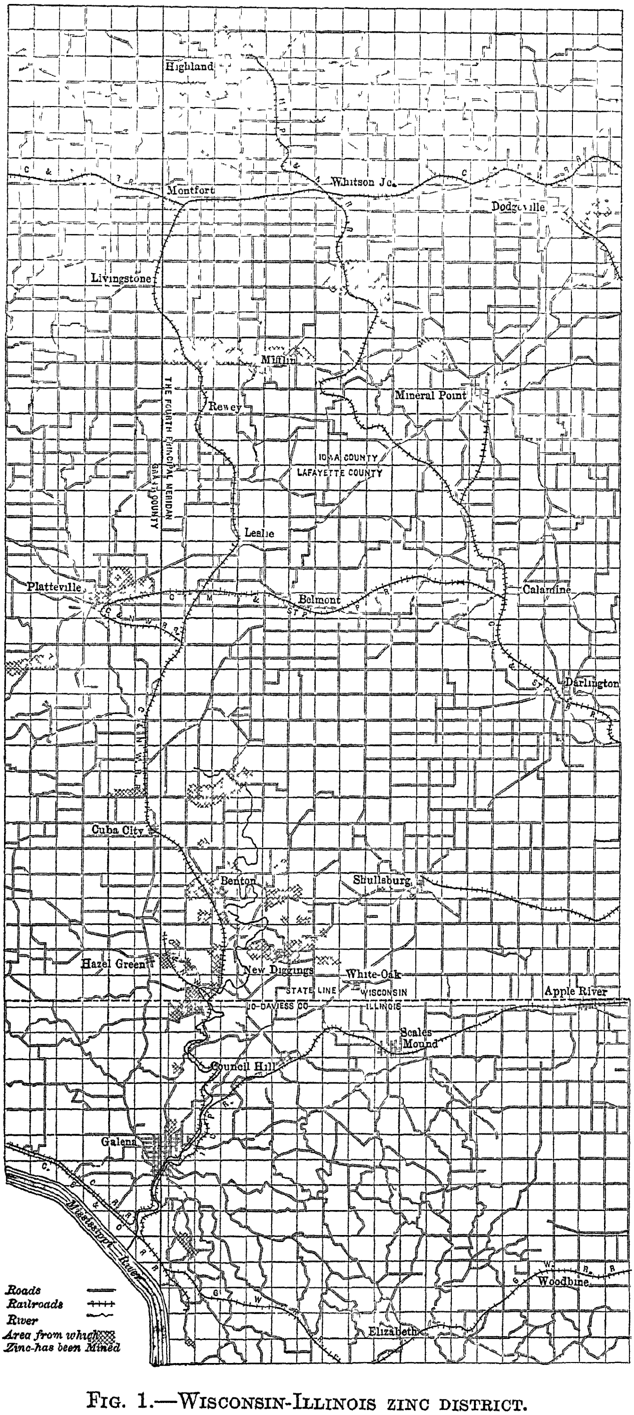



FIG. 1.—WISCONSIN-ILLINOIS ZINC DISTRICT.

per cent. of the gross return from ore sales, except with one company which generally buys the land outright. The leasing system has resulted in a close relationship between the land owners, or farming element, and the mine operators, causing the former to aid the operators, generally, in development and prospecting work and to curb radical labor agitators. from which the district has been comparatively free.

## History of the District

The Wisconsin district may fairly claim to be one of the oldest in America. As early as 1780, French traders, under Julian Dubuque, bought considerable lead ore from the Indians. In 1823, lead mining under white settlers began at Galena and Shullsburg; and by 1846, over 2000 lead leases had been granted to miners by the government. The first zinc ores were mined in 1862, principally around Highland. These were the carbonate, known locally as "drybone." Blende was of little value and generally discarded. About 1903, with advancing prices for zinc ore, attention was turned to the sulfides, and production increased rapidly with the introduction of modern mining and milling methods. The approximate lead production from the mines is as follows: 1822–1860, 205,000 tons; 1860–1900, 74,500 tons; 1900–1918, 94,000 tons. The approximate zinc production is given in Table 1.

Table 1.—*Approximate Zinc Production from Mines*

| Years | Tons | Years | Tons | Years | Tons |
|---|---|---|---|---|---|
| 1862–1890 | 283,500[a] | 1908 | 58,000 | 1914 | 134,500 |
| 1890–1903 | 126,000[a] | 1909 | 75,500 | 1915 | 164,000 |
| 1904 | 19,000[b] | 1910 | 102,000 | 1916 | 219,000 |
| 1905 | 33,000 | 1911 | 107,000 | 1917 | 250,000 |
| 1906 | 39,000 | 1912 | 119,000 | 1918 | 194,500 |
| 1907 | 53,000 | 1913 | 106,000 | | |

[a] Carbonate and blende. [b] Blende principally.

The sharp rise in the production in 1916 and 1917 was due to intensive mining on account of the high prices then prevailing for blende. The drop in 1918 reflects high costs and lower prices, causing many of the smaller mines to close. Figures for the three months of 1919 show the shipments to be at the rate of 140,000 tons for the year, but final figures are likely to be lower.

In 1916, there were over 80 mines reporting shipments. This number has been cut to a third on account of adverse operating conditions. As

might be expected, the years have shown that the large operating companies are producing a constantly increasing proportion of the ore of the field. The four largest companies are Mineral Point Zinc, Vinegar Hill, Wisconsin Zinc, and Frontier. Table 2 shows their shipments in comparison with the rest of the field.

TABLE 2.—*Shipments of Four Largest Companies*

| Year | Shipments of Entire District, Tons | Shipments of Four Companies, Tons | Per Cent Shipped by Four Companies |
|---|---|---|---|
| 1916 | 217,000 | 137,598 | 63 |
| 1917 | 250,000 | 176,413 | 70 |
| 1918 | 194,500 | 146,995 | 75 |

The success of the district in the main has been dependent on the working of large bodies of low-grade ore at a low cost. During the era of high prices for ore, in 1916, some extremely thin ground was carried at a profit. At the Penna mine, of the Mineral Point Zinc, the mill recovery of concentrates was only 2.0 per cent. during March, 1916. The concentrates averaged about 25 per cent. zinc. This meant only 0.5 per cent. zinc in the ore; or, assuming a 65 per cent. mill recovery, that the mine dirt carried only 0.77 per cent. zinc. This, of course, was exceptionally low. Under present conditions, the recovery in the district is about 7 per cent. and the concentrates average about 35 per cent. zinc, giving a figure of 3.7 per cent. zinc for the average of the ore in the ground. Both the mill recovery and the grade of the concentrates have been raised very much during the past year, through sheer necessity, by mining richer ground and by improving the grade of the concentrates. On a normal ore market, the recovery would be about 5–6 per cent. and the concentrates not over 30 per cent. zinc in grade. This would indicate 2.5 per cent. zinc in the ground.

## GEOLOGY

The geology of the district is relatively simple. It has been elaborately discussed by Whitney, Strong, Chamberlain, Blake, Winslow, and Bain, in numerous publications of the Wisconsin, Illinois, and U. S. Geological Surveys. Only a brief résumé will be given and acknowledgment is freely made to the foregoing for much of the data.

The region consists of unmetamorphosed, little disturbed sedimentary rocks of Paleozoic age. Igneous rocks are absent. A general section of the present rocks follows.

SECTION OF ROCKS OF UPPER MISSISSIPPI VALLEY[1]

| System | Formation | Character | Thickness, Feet |
|---|---|---|---|
| | | Alluvium | |
| Quarternary | | Terrace deposits | 5–70 |
| | | Loess | |
| | | Residual clays | |
| Silurian | Niagara | Dolomite | 150 |
| | Maquoketa | Shales | 160 |
| | Galena | Dolomite | 240 |
| Ordovician | Platteville | Limestone & dolomite | 55 |
| | St Peter | Sandstone | 80 |
| Prairie du Chien | Shakopee | Dolomite | 50 |
| | New Richmond | Sandstone | 10–40 |
| | Oneota | Dolomite | 200 |
| Cambrian | Potsdam | Sandstone | 800 |
| Pre-Cambrian | | Quartzite | |

Erosion has exposed all these formations, except the Potsdam, in different parts of the district. The full extent of the Galena formation is shown at Hazel Green, Galena, and points south and west, where the Maquoketa has not been eroded. North of Platteville only about 100 to 150 ft. (30 to 45 m.) of the Galena remains; and at Highland, erosion has exposed the Platteville over a large area. A deep well drillhole at the Clark mine at Highland shows the following section:

| | Feet |
|---|---|
| Galena dolomite | 88 |
| Trenton (Platteville) lime | 47 |
| St. Peter sandstone | 56 |
| Lower Magnesian lime (Prairie du Chien) | 204 |
| Potsdam, penetrated 86 ft. only in formation | |

## Topography

The deposits lie within the Driftless Area. There is a total relief of about 1100 ft. (335 m.), the bottom lands of the Mississippi lying about 600 ft. (182 m.) above sea level, and the tops of the highest hills above 1700 ft. (518 m.). In the northern part of the district, there is an east-and-west unbroken ridge, from which a north-and-south ridge extends through the middle of the district for about 30 mi. (48 km.). The ground slopes north and south from the north ridge, and east and west from the middle ridge to drainage basins.

## Structure

The rocks have a slight dip, to the southwest, of about 25 ft. (7.6 m.) to the mile. There has been some compression and folding, which has produced a series of structural basins. The main axis of most of these

[1] H. Foster Bain: Zinc and Lead Deposits of the Upper Mississippi Valley. U. S. Geol. Survey *Bull* 294 (1906).

basins is east and west, with a secondary axis north and south As these basins are intimately related to the orebodies, some further comment is needed.

Bain distinguishes four types: shallow basins, flat monoclines, asymmetric anticlines, and canoe-shaped basins. The last are the most important and, speaking generally, typify all the important ore deposits in the southern part of the field. They are closely connected, in theory, with the unique pitch and flat formation of the ore, which will be discussed later. These synclinal areas are true pitching troughs, with sides much steeper than the normal dip of the rocks, usually many times longer than wide, and with one end liable to be deeper than the other. There is no evidence of any faulting. It is within these basins, or along their sides, that the orebodies are found, and prospecting work is directed to determine from a close study of the structural contours of the oil rock where these basins are located, and to follow out their extent in the hope of finding a valuable ore deposit upon them.

Of late years a common occurrence in the rich New Diggings field has been the discovery of small, distinct basins, usually elliptic in shape with sharply rising sides, located along a general range, usually east and west or nearly so. There is no connection between these orebodies of workable ground, yet when mapped they lie along a fairly definite course, that would seem to indicate that the same original crevice was responsible for their formation.

Their distribution is not uniform throughout the field. It corresponds, so far as known, with the development of the mines. Over a century of prospecting (which until the last ten years has been mainly superficial) has failed to show any new developments outside of the areas adjacent to these camps. The origin of these basins, according to Chamberlain, was due to conditions of deposition modified in part by deformation due to pressure. Undoubtedly Bain's theory amplifying this has much to confirm it: that in certain favored places there was a settling of the oil rock underlying the Galena, due to consolidation of the former, with a consequent enlarging and deepening of the basin.

Ore deposits are confined entirely to the Galena dolomite, and the top of the Platteville formation, which is known locally as glass rock. The Galena dolomite is the main ore-bearing rock of the district. It is a granular, highly crystalline dolomite. Chert or flint is abundant in the middle part. The full thickness of the formation appears only in the southern part of the field; in the north only 100 to 150 ft. (30 to 45 m.) are present. The base of the formation is called locally oil rock. It is an impure lime impregnated with a large amount of organic matter, showing 18 to 40 per cent. of carbonaceous matter, which, when dry, will burn with a peculiar petroleum odor. The thickness varies from a few inches to several feet. As the rock is most abundant around the mines,

in drilling, its presence is always taken as a favorable indication of proximity to ore. Its importance geologically lies in its capacity to furnish a large amount of material especially well adapted for causing the precipitation of metallic salts as sulfides.

The glass rock, the only portion of the underlying Platteville formation known to contain workable ore, is a dense, hard, conchoidally breaking limestone, hence the name. It lies directly under the so-called clay bed that marks the base of the Galena formation. The glass rock varies in thickness from 2 ft. (0.6 m.) to over 16 ft. (4 m.) in the eastern part of the field. It is an important formation for ore in the Linden, Mifflin, and Highland fields, where glass-rock blende is frequently high grade. In the New Diggings field, where it attains its greatest thickness, it is usually unpayable in the top 6 or 10 ft., but in the lower part it frequently contains considerable ore, and may, as in the Winskill mine, be large enough to justify mining exclusively for itself. Generally, however, glass-rock ore is "spotty" and less regular in its extent and richness than the deposits in the overlying Galena. Recent developments at Shullsburg, however, in the eastern part of the field, have given glass-rock ore a large significance to the operators.

The base of the glass rock is the limit of workable ore in the field to date. Drilling in a few isolated holes has shown small amounts of zinc and iron throughout the Platteville formation, but never to any extent. A hole on the Fox land, which was sunk 30 ft. into the Platteville, showed ore assaying 2.6 per cent. Zn, 5.0 per cent. Fe. We think it unlikely that future work will show up anything below the now recognized limits of the ore horizon.

## Ores

The ores of the field consist of blende, smithsonite, and galena with associated marcasite and pyrite. The principal gangue mineral is calcite. Chalcopyrite and barite are occasionally encountered. Mineralogically the ores are the simplest. The complex sulfides of the west are entirely absent. The absence of cadmium in the zinc, and of silver (practically) in the lead, is noteworthy.

*Types of Ore Occurrence.*—Bain distinguishes four types of ore occurrence: crevices or openings, honeycomb or sprangle, pitches and flats, disseminated deposits. More simply, perhaps, we may classify these as top runs and bottom runs, meaning by the former, an orebody occurring in the top, or middle, part of the Galena, and not extending down to the oil rock, and by the latter, an orebody whose base rests on the oil rock, and which occupied a synclinal basin in the oil rock. The latter are, from a practical standpoint, of paramount importance in the field and are the only type worked by the larger operators on a larger scale. The top runs include crevice and honeycomb deposits. The

unique pitch and flat formation is sometimes seen in top runs, but is especially well developed in the bottom runs. Disseminated deposits are only found in or close to, the oil rock.

The crevices were worked mainly by the old lead miners; they comprise what Whitney calls "gash veins." The deposit occurs along a joint crack in the rock, which has been enlarged by the action of underground waters to form irregular cavities and openings, which may be from 1 to 4 ft. (0 3 to 1.2 m.) wide, 4 to 6 ft. (1.2 to 1.8 m.) high, and of considerable length. There may be more than one opening in the same vertical crevice, so that the miners speak of the first opening, second opening, flint opening, etc. Often these crevices may be traced for over a mile, but the openings are usually of limited extent. Lead was the main ore sought and practically the entire production of the field, in the early years of its history, came from surface deposits and crevices. As much as 1000 tons is reported from one of these openings, though the average recovered was but a small per cent of this.

Zinc often occurs in these crevices as a sheet of extremely pure blende, sometimes 3 or 4 in. (7 or 10 cm.) thick, lining the walls of the opening or occurring as a brangle in the brecciated portion of the rock. Considerable work has been done on such deposits, but the results have been disappointing. Although the resulting concentrates are high grade and bring satisfactory prices, on account of the narrow width and height of the orebodies it is impossible to break any tonnage, and costs are high. The limited extent of the deposits likewise prevent any considerable capital expenditure for mill and equipment.

Much of what has been said of crevices will apply to honeycomb deposits. They are relatively unimportant. Where conditions are right, a porous dolomite is partly filled with metallic sulfides and the resulting orebody is called a honeycomb, or sprangle deposit. They cannot be worked on a large scale.

Disseminated deposits consist of small crystals of blende in clay or shale. The ore is locally known as "strawberry jack" from the distinct crystals. The formation always occurs next to the oil rock and may be of considerable horizontal but small vertical extent. They were formerly of some importance in the north end of the field, but are not now worked.

*Pitches and Flats.*—These form vastly the most important orebodies of the district and are distinctly unique. The ores follow, in part, the horizontal bedding planes and, in part, the inclined breaks between these planes. The result is an orebody occupying a series of horizontal sheets, or "flats," and connected by a series of dipping sheets or "pitches." The shape of the orebody, when excavated, will be frequently that of an inverted trough or, where the pitches assume an elliptic or horseshoe form, an inverted bowl.

The classic illustrations of the pitch and flat formation is that given

by Chamberlain in the Geology of Wisconsin, which consists of a cross-section of the Roberts mine at Linden; it is here reproduced in Fig. 2. It will be noted that there is originally a vertical crevice in descending from the surface, which spreads out to form the so-called top flat. This, in turn, leads on either side to the two pitches, which dip in opposite directions and descend, by steps, until the glass rock is reached, marking the limit of the orebody. (These pitches usually terminate above the oil rock in the thin laminated layers under the blue ground, but at Linden they sometimes persist through to the glass rock.) The space between the two pitches is locally termed "core ground" and, as shown in the section, is more or less mineralized by flat sheets and parallel pitches leading off from the main pitch. In the ideal case, the entire core would be so completely mineralized that it would be possible to mine everything from one pitch to the other. Such cases have occurred, but it generally happens that the amount of mineralization in the core varies within rather wide limits, hence unpayable spots are frequently left, as pillars, if possible.

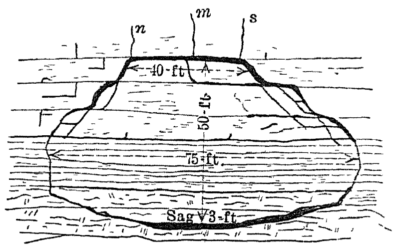



FIG. 2.—SECTION OF FLATS AND PITCHES OF ROBERT'S MINE, LINDEN, WIS. N, M, S ARE NORTH, MIDDLE, AND SOUTH CREVICES. (*After Chamberlain*)

Of the pitch and flat formation, there are many different types, some differing very markedly from the ore in Fig. 2. Chamberlain's illustration throws, perhaps, too great an emphasis on the presence of two pitches, each of equal strength. It seldom happens that both pitches are equally good; usually one shows a better defined hanging wall than the other. The mineralization also may be quite different; a high-grade blende on one pitch and predominance of marcasite on the other. Of such a character is the well-known Black Jack mine, or as it was formerly called, the Marsden. Here one pitch shows heavy black jack, with a firm, smooth, hanging wall from which, generally, the pitch ore breaks clean. The other pitch is heavy in marcasite and has no definite hanging wall. The rock thereon shows considerable evidence of metasomatism, and no clear line can be distinguished for the outside of the ore.

While the opposite pitch may be weak, as at the Black Jack, it is sometimes lacking entirely and the entire mining may be done on one pitch, with as much of the core ground as may be carried profitably.

In such a case there is no definite limit to the orebody in the core ground; the limit will be a function of the cost sheet. The best illustration we have of such a condition is at the Coker mines of the Mineral Point Zinc Co., shown in Fig. 3. A reference to the map will show the extraordinary length and persistence of the orebodies, showing today, with No. 1 mine, a mined and drilled out orebody of some 6900 ft. (2103 m ). No. 2 orebody, if the mines of the Vinegar Hill Zinc Co. are included, will show a continuous length of some 7500 ft. (2286 m.). The dip of the pitch at No. 1 mine is to the north; at No. 2, to the south. In theory, there should be a south leg for No. 1, and a north leg for No. 2, but drilling and prospecting has failed to show them.

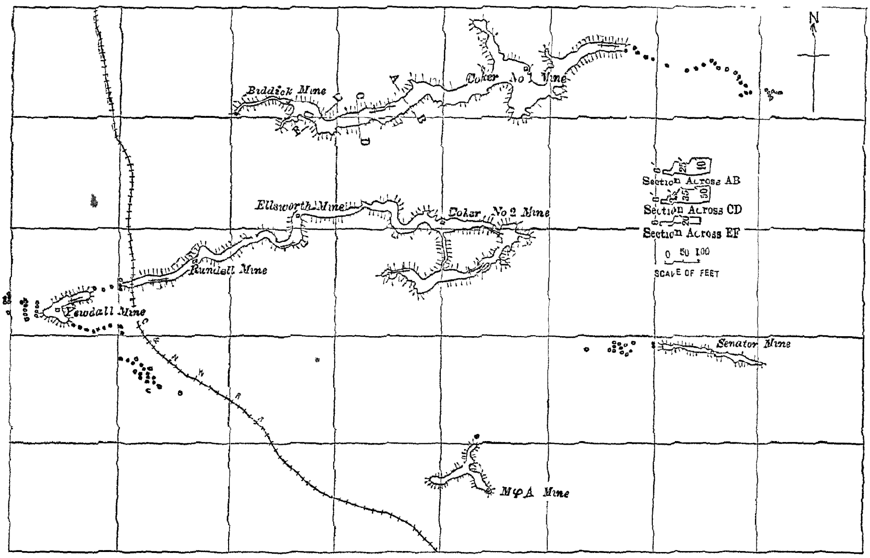



FIG. 3.—COKERVILLE MINES, LIVINGSTON, WIS.

This is one of the oldest and most productive orebodies in the district. A noticeable feature is the wavy line that the range takes, maintaining, however, a generally westerly direction. These swings of the pitch are frequently quite sharp, and on the Coker 2 map, it will be noted that in the space of only 100 ft. (30 m.) the pitch swung around in direction 90°. Sections of the orebody are shown across lines *AB*, *CD*, *EF*. The number of flats, height of pitch, and floor elevation vary within considerable limits, but its generally uniform width is remarkable. The mines show very well, as in section *AB*, the several steps of the pitch and flat, a phenomenon rarely seen in the mines farther south, where steep pitches, with no break until the top of the orebody is reached, are the rule. The Coker mines furnish the best illustration of the "flat" running off at right angles to the strike of the pitch, sometimes for 50 ft.

(15 m.) or more. Such a flat, besides giving heavy ore for the mill, furnishes an excellent chance to drive ahead rapidly and open up an orebody for future production, and the system has been used with good effect at the mine.

The Coker mines furnish the best and simplest illustration of the single pitch and flat formation, and any layman can grasp the geology after a single visit to the mine. The mines in the southern portion, however, do not form along such simple lines. Bain, in his monograph written in 1906, noted the significant fact that besides the presence of two outwardly pitching orebodies parallel to the trend of the main crevice, we may find other pitches, the strikes of which are at right angles, or nearly so, to that of the former, which often tends to give the orebody a semicircular shape, something like a flatiron. Since the monograph was published, later work has tended to emphasize the presence of right-angled or quartering pitches in limiting the extent of the ore along the strike of the main pitch.

As noted elsewhere in this paper, there appears to be a tendency of the main "range" to continue, though often so tight as to show no ore, and some hundreds of feet distant from the first orebody, to open up again with a similar series of pitches, and make a second orebody. The best example of this is at the old Lucky 12 mine at New Diggings, where there are three separate orebodies lying along the same general course, connected with each other by drifts that show no ore. It is believed that future drilling work, which necessarily must be guided more by geological theory and less by surface indications, will show other orebodies lying on extensions of ranges now thought to be worked out.

Fig. 4 shows a map of the well-known Kennedy mine of the Mineral Point Zinc Co. This, including the Mills and Winnebago mines, which are an integral part of the Kennedy orebody, is undoubtedly the largest deposit of ore yet found in the district. The mines have been worked since 1860, or earlier. Exact figures of the production are not available, but it is likely that about 2,500,000 tons have been mined, yielding 150,000 tons of concentrates assaying 35 per cent. zinc. The range, as mined today, has a total length northwest and southeast of 3000 ft. (914 m.) and a maximum width of 1500 ft. (457 m.); the mined-out area is approximately 20 acres (8 ha.). The general course of the range is northwesterly and the mine shows the double pitch of Chamberlain's section, though on a vastly larger scale. On the northeast side, the pitch dips about 60° in the direction shown by the arrows, and continues northwest from the Winnebago shaft for 1000 ft. (304 m.), showing a strong, well-defined hanging wall and rich ore, which extended in places clear across the core over to the south pitch and frequently showed stopes over 60 ft. (18 m.) high. South of the Winnebago shaft and north of the Mills, the pitch formation is not so well defined, due no doubt to

crossing pitches which carried the mine away from its normal course and 500 ft. (152 m.) to the east. The influence of the latter are plainly seen on the south, or west pitch, which makes almost a 90° turn at a point

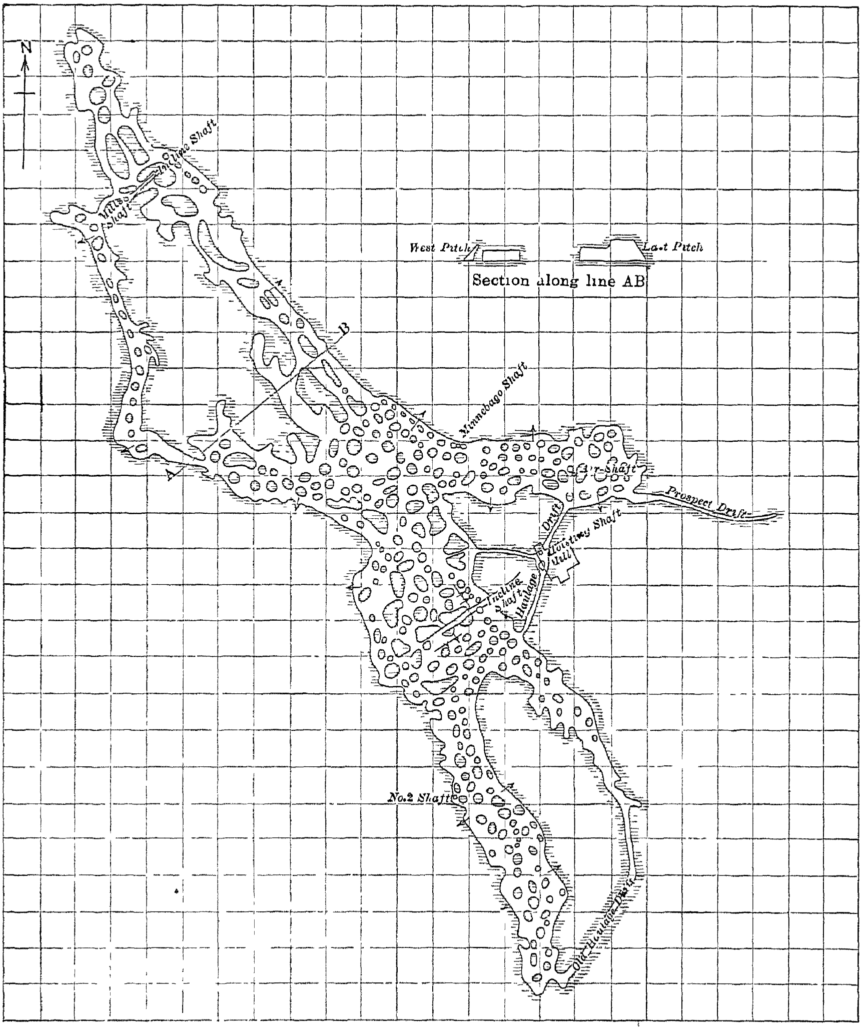



FIG 4.—KENNEDY MINE, HAZEL GREEN, WIS.

650 ft. (198 m.) distant from the east pitch, and thence converges rapidly toward it until, at the Mills shaft, they are only 300 ft. (91 m.) distant.

Where the Kennedy incline was sunk, a third pitch carrying rich ore was cut, bearing about northeast and lying at right angles to the strike of the two main pitches. This resulted in a phenomenally rich orebody

for several hundred feet around the incline. This third pitch did not cut off any further extension of the ore to the southeast, as has sometimes been the case elsewhere. It did, however, affect the formation to a marked extent; and although the orebody persisted for 1000 ft. (304 m.) south of the incline, the pitches and hanging wall became less defined and the stopes dropped down in height appreciably. The ore, though continuing rich, became more of a flat deposit than the pitch type.

*Genesis of the Ores.*—It is now generally admitted that the original source of the ores was from the surrounding rocks and concentration effected by descending solutions. That any theory of deep-seated origin and ascending solutions is untenable is shown: by the absence of any true fissure veins, by the presence of stalactites in all openings; and by the tendency of the ore to flatten out in broad sheets along on every impervious bed, for instance the clay bed, as well as by other evidence.

A résumé of Bain's theory of the origin of the ores is as follows:

1. The ores represent a concentration of material originally disseminated through the surrounding rocks.

2. The zinc and lead originally present in the crystalline rocks of the Lake Superior district were delivered by rivers to an especially favored part of the Ordovician coast, where they were concentrated by evaporation and precipitated in quantity.

3. The agents of precipitation were hydrocarbons or $H_2S$ derived from decaying masses of algæ within certain shallow basins on the sea floor.

4. As the algæ matter became compressed, the beds above broke and settled leaving open spaces of peculiar form for the ultimate reception of orebodies.

5. Deep joint cracks developed.

6. The area was subjected to peneplanation and gorge cutting, when the overlying shale was cut away.

7. Vigorous underground circulation was brought about.

8. This circulation resulted in a concentration of disseminated material, and with a fall in groundwater level the ores were reconcentrated, the favored areas being the present structural basins, where, it logically follows, then was originally the greatest depth of oil rock, which, under compression, would have yielded the largest amount of precipitating matter.

## Prospecting

The early miners, who were entirely concerned with lead, sunk shallow test pits at favorable points on the hillsides or where float lead occurred and drove narrow drifts to prospect between the shafts. Many of the hillsides around Galena are literally pock-marked with these old pits. These shafts were rarely sunk below water level, as pumps were generally lacking, and as the galena was usually found above the water

level, there was no incentive for going deeper. Zinc at that time was practically valueless and regarded as a detriment, necessitating careful sorting to free the lead from it. Hence in many cases the old timers would actually stop sinking on top of a good sheet of jack, which often was coincident with the groundwater level. With the development of zinc mining in the district, therefore, the obvious thing to do was to sink these old lead shafts deeper, wherever the dump showed zinc ore, and proceed with development. The well-known Marsden Black Jack mine was originally discovered as a lead mine and worked for lead alone for years.

It was early learned, however, that some other method must be used to prospect for zinc, as shaft sinking was too slow and costly and, in all probability, new and valuable zinc deposits would be discovered that might not be at all related to present lead mines. Hence churn drills were introduced about 1900. These were mostly light, portable machines, with small string of tools and not adapted for drilling deep holes or very hard ground.

The development of drilling from a mechanical standpoint alone, is interesting. Bain gives the cost, in 1906, as 75 c. to $1 per ft. In spite of the greatly enhanced costs of labor and material, the cost now varies from 60 to 75 c. per ft., due entirely to the use of large, well-designed machines and very heavy strings of tools. Where formerly a 3½-in. (9-cm.) bar was considered enough for average work, now a 32 ft. 4¼-in. (9 m. 10 cm.) "string," weighing about 1500 lb. (680 kg.) is customary. Formerly 20 ft. (6 m.) per day in hard ground was a day's work; now 35 to 40 ft. (10 to 12 m.) is looked for. Nearly all drilling is contracted, the large companies having found this more satisfactory than owning and operating their own drills. Hence there are in the district a number of experienced, efficient drillers who understand the ground and rarely lose a hole through carelessness or lack of knowledge of the business. In spite of its relative cheapness as a prospecting method drilling is expensive, and can cost a great deal of money in a very little time. Of late years, the importance of careful supervision by experienced men has been recognized by the operators and practically all drilling is now done under the direction of a "drill engineer," who spots the holes on the basis of his study and knowledge of the geology of the field, examines all cuttings, checking his judgment of those showing ore by assays; and if an ore deposit is found, blocks it out by properly spaced drilling until enough ore is shown to justify the sinking of a shaft or the erection of a mill. In early years, there was little if any supervision of the drilling; hence much money spent on this work was practically wasted, so far as providing permanent authentic records of the ground is concerned. Holes were spotted without surveying or level notes and the driller's report was taken as final on the worth of the cuttings.

During the brief, though hectic, boom that the district had in 1905–6, there was considerable wildcat drilling and, it is reported, some salting of drill holes. Many shafts were sunk and some mills built on utterly insufficient showings. Of late years, under proper supervision, there is no case, that we can recall, of deliberate salting of drill holes. The incentive is usually lacking for such dishonesty, as most lands are leased and not purchased outright for their mining value; or if they are, the drilling is done subsequent to such purchase.

When zinc mining was first started in the district, the favored places for drilling were naturally adjacent to the old lead ranges and shafts, particularly where these showed jack on the dumps. It was early recognized that the zinc deposit, where such existed, was rarely directly under the lead range but usually to one side, and sometimes removed hundreds of feet from such ranges. With a clearer understanding of the geology of the field, the aim of the drill engineer is to pick up a structural basin adjacent, if possible, to these lead ranges, and follow it down to its bottom; or failing to find any ore there, to follow it out along its longest axis. This necessitates careful surveying and leveling of all holes and accurate platting of the oil rock. With the exploitation of the known lead ranges during the last few years, it has become more and more necessary to use all possible knowledge of the structural basins for a guide in drilling, as surface indications, lead pits, crevices, etc. that used to be depended on are now largely lacking in the new fields.

When a good hole has been found, the blocking out and extending of the orebody is not especially difficult, if enough money is spent in drilling and, of course, if there is any size and height to the ore. The usual procedure is to cross-cut the range first in order to determine the width of the orebody, and the direction of the pitch. When this is known, the range can be more easily drilled out along its course. Our practice, when the dip of the pitch and the width of the range are known, is to place holes in zig-zag positions along the course of the range, thus gaining information on any change in height of pitch, as well as proving up more territory. The amount of drilling deemed necessary by the operators to prove up a mine will vary with the richness and quantity of the cuttings. Some companies believe in much closer drilling of an orebody than others, and take many cross-sections through the stopes. These sections are carefully platted, and subsequent mining and development are planned from a study of them. It is debatable whether this practice cannot be overdone; or in other words, if much of the information supposed to be gained from drill-hole sections would not have been had in the regular course of mining. Moreover, it should be emphasized that a drill log, no matter how carefully taken, must be interpreted in the light of experience with former mining conditions, and will frequently fail to tell the whole story; or more yet, may present it in a distorted or exaggerated

light. Mr. H. C. George, of the Wisconsin Zinc Co., in a paper presented before the Institute in February, 1918,[2] has two sections of drill holes showing very clearly how cuttings may be lost in an open crevice cut by a hole, with the result that the hole might be reported blank, or again, ore may be carried down from a top flat in soft ground, and so salt all lower runs. Every drill engineer has had dozens of such experiences and would hesitate before drawing conclusions from any one hole.

From the assays of the sludgings, where ore occurs, the proper calculations are made to convert them to sulfides, and thus the grade of the dirt and grade of concentrates can be figured. Mill recovery is usually taken at 60 to 70 per cent. of the zinc and 50 per cent. of the iron; but these factors are largely empiric and are affected by the system of milling, as well as the character of the ores. For instance, soft marcasite is slimed away in milling to a remarkable extent and the concentrates will be much higher in grade than might be expected from the iron assay.

Some companies make tonnage estimates of ore from the sections and records of drill holes. Mr. George, in the paper referred to, gives a full example and figures of such a problem. Unless the drilling has been exhaustive, such calculations are only approximations.

### Operating

*General.*—The mining problem in Wisconsin is to handle large bodies of low-grade ore at a low cost. It is, therefore, essential to handle as large a tonnage as possible in order to cut down overhead and fixed charges. It is interesting to note the increase of cars per man shift with the increase of cars hoisted per day. A mine hoisting 320 cars per day produced 4.6 cars per man shift, but with better labor conditions following the close of the war the same mine hoisted 520 cars per day with an increase to 6.2 cars per man shift. Another mine hoisting 790 cars per day produced 10.6 cars per man shift; and upon increasing the cars hoisted to 1012 produced 12.9 cars per man shift. With the larger number of cars per man shift, it is evident that the labor cost is correspondingly reduced. It is also true that supply and power items do not increase in anything like a direct ratio to cars handled.

Orebodies are from 5 to 100 ft. (1.5 to 30 m.) high and from 20 to 300 ft. (6 to 91 m.) wide. In a low stope, a simple heading is carried on the rail-fence plan. For high stopes, underhand mining in open stopes is the method used throughout the district. Pillars 10 to 30 ft. (3 to 9 m.) in diameter are left for the support of the roof. Spacing between pillars is from 20 to 50 ft. (6 to 15 m.), depending on the character of the ground and roof to be supported. The pillars are left in lean ore wherever possible. Limestone bedding is horizontal and bedding planes vary from a few inches to several feet thick. One of these bedding planes at

[2] *Trans.* (1918) **59**, 117.

the top of the ore is selected for a roof, or cap, and is kept unbroken between pillars, this insures a solid top and safety for men working beneath. Location of pillars is determined largely by the character and shape of the body; the ground stands well and there is very little trouble from this source.

Where there is a glass-rock or oil-rock flat, a row of pillars is left at the edge of the flat, or bottom of the pitch, in order to hold up the brow formed by the change from low ground to high. Other pillars are left in the high part of the stope, as required by the width of the stope and character of the ground.

*Shafts and Hoisting.*—Location of shafts is determined by the prospect drilling. They are generally located as centrally as possible and, if practicable, at a low point in the orebody. The depth ranges from 60 to 200 ft. (18 to 60 m.), and a sump 10 to 15 ft. (3 to 4.5 m.) deep is left below the working level for water. From the collar down to solid rock the shaft is either cribbed with timber or lined with concrete. Cages and cars are used for hoisting, and shafts have either one or two cage compartments with an extra compartment for ladders and pipes. Cages weigh 1300 lb. (589 kg ), cars 600 lb., and, in single-cage hoisting, a 1500-lb. counterweight is used. The hoisting rope is of ¾ in. (19 mm.) plow steel. Size of hoisting motors varies from 35 to 52 hp. and the hoisting speed is about 600 ft. (182 m.) per min. At one mine having a 35-hp. motor and single-cage shaft and a hoisting depth of 195 ft. (59 m.) 530 cars have been hoisted in a 9-hr. shift; this is 10 cars less than one car per minute for the entire shift, and a little more than one car per minute after taking out time for lowering and hoisting men. In a two-compartment shaft, this work has been equaled for a period of a month. Two men cage the cars at the bottom and two men on top pull the cars off into a self-dumping cradle above the ore bin and then cage the empty car. For safety at the top landing, a wooden hood is built over the shaft to work up and down on the guides. When the cage comes up, it raises this hood and lowers it into place again on the down trip, this avoids the loss of time where the topman must open and close a gate.

*Breaking Ground.*—The method used in breaking ground is one of underhand stoping. A heading 5 to 6 ft. (1.5 to 1.8 m.) high is carried at the top of the stope. This is broken in the usual rail-fence shape, three holes, one above the other, being necessary for breaking each slice. Air-hammer drills mounted on columns are used in the headings. The lower part of the stope is broken by horizontal holes acting as lifters, each hole having 10 to 15 ft. (3 to 4.5 m.) of ground on it. These holes are not placed on a regular system, but are drilled as determined by the shape of the breast to give the best results. They are drilled with an air-hammer drill mounted on a tripod or with Jackhamers. Broken dirt rolls to the bottom and is shoveled into cars. The heading at the top of

the stope is pushed ahead and the lower part of the stope is only drilled and shot when dirt does not roll to the bottom. Most stope holes are squibbed with a few sticks of powder. Contract drillers receive 4½ to 6 c. per car broken. The following analysis of drilling and blasting figures gives a fair average of work done. Stopes in mine No. 1 are only 8 ft. (2.4 m.) high, while those in mine No. 2 are 20 to 70 ft. (6 to 21 m.) high.

| | Mine No 1 | Mine No 2 |
|---|---|---|
| Average depth of holes, in feet | 9 5 | 13 6 |
| Holes per machine shift | 2 9 | 2 64 |
| Feet drilled per machine shift | 27 5 | 36 0 |
| Cars per hole | 34 8 | 60 0 |
| Cars per machine shift | 100 9 | 158 4 |
| Cars per pound of powder (40 per cent gelatine) | 1 4 | 1 83 |

*Shoveling.*—Shoveling is the highest labor item appearing on the cost sheet and for this reason has been given close attention. Aside from the item of expense, shoveling is the limiting factor in increasing mine production. There are only a limited number of places to shovel from and by increasing the number of cars from each shoveler the coal mine tonnage is increased.

Cars having a capacity of 16.5 cu. ft. (0.46 cu. m.) are used. The car is low and the top area is large so that the shoveler has no trouble in putting all of the dirt into the car. Tracks are carried up to the broken dirt so that the car is as close to the work as is convenient. A stope 30 ft. (9 m.) wide between pillars will accommodate three tracks. In a high stope, a shoveler will handle 40 to 50 cars; one heading produces 120 to 150 cars per 9-hr. shift. Each shoveler fills his car and trams it to a sidetrack and brings in an empty. The sidetrack is kept within 100 ft. (30 m.) of the heading. In some of the mines a mule brings in two cars to the heading. The shoveler turns these over to one side of the track to let the mule pull out the full cars and then turns one car back on the track, fills it, and pushes it back a few feet past the other car, which is then turned up on the track and filled. The cars are light and easy to handle and it is quicker than tramming to a lay-by.

The dirt, as broken, contains many large boulders, some of which are drilled and shot; others are broken by the shoveler with rock hammers. Boulders containing ore must be broken and sent to the mill. Large boulders of pure limestone are loaded in a separate car and thrown over the dump. From 5 to 15 per cent. of the dirt broken is sent over the dump as boulders. Several attempts were made to do this sorting on grizzlies on the surface, but it was found to be too costly. It was also difficult to keep the grizzlies cleared when cars were coming fast. Bedding planes in the limestone generally give a smooth shoveling surface but flat sheets are used in most places.

No. 2 D-handle scoops, holding approximately 27.5 lb. (12.5 kg.)

are used; weight of load depends on richness of ore and amount of water in the dirt. The men are paid on a contract basis of 12 c. per car. In small stopes with poor working conditions shoveling is done by men at $3.50 per day. Results in a mine having poor shoveling conditions show that 67 per cent. of the shoveling was done by contract labor. The average per shoveler was 27.5 cars, and the average for contract shovelers was 40.8 cars per shift. Another mine with better than average conditions shows that 95 per cent. of the shoveling was done by contract labor with an average for all shovelers of 46 cars, and an average for contract shovelers of 46.6 cars per shift.

*Haulage.*—Where headings are not more than 300 to 400 ft. (91 to 121 m.) from the shaft, hand tramming is the most economical and a trammer can handle from 100 to 200 cars per shift. For longer trams, mules weighing from 800 to 1100 lb. (362 to 499 kg.) are used. Few of the mines have hauls of over 2000 ft. (609 m.) and mechanical haulage has been used but little.

Tracks are of 16-lb. (7.25-kg.) rail, 22-in. (55.8-cm.) gage, and are laid on the mine floor without any particular attention being paid to grade. The floor of the mine follows the ore, so it is frequently necessary to lay the tracks on grades as high as 15 per cent. These dips are eased up as soon as conditions will permit and grades on main haulage tracks are kept below 4 per cent. A mule can handle two cars for short distances on 6 or 7 per cent. grades, or one car on a short 15 per cent. grade. No brakes are used on cars. If the grade is too steep, one car wheel is spragged; or with empties, the driver holds cars back by placing his hand on the mule's rump. Where the room is low a space is hollowed out between ties for the mule.

A mule pulling two cars for a distance of 1200 ft. (365 m.) can handle dirt from one shoveler; for distances greater than this, it is necessary to put in a side track. The first mule pulls to the side track and a second mule pulls from there to the shaft. The track from the side track to the shaft is kept in good condition and the second mule pulls four cars and takes cars from two heading mules. A mule will make from 12 to 20 ton-miles per shift at a cost of 25 c. to 35 c. per ton-mile.

*Pumping.*—Mine water varies from a few gallons per minute up to 3000 gal. (11,355 l.) or more. Pumps used include various types of cross-head pumps, motor-driven triplex pumps, and motor-driven centrifugal pumps; later practice inclines to triplex and centrifugal pumps. The triplex are the most efficient but the centrifugal pumps have found wide use on account of being light and easily installed and moved; the pump capacity can also be regulated to the mine flow by throttling the discharge. Sumps are generally small, a very convenient feature. As the headings progress, there are places where the bottom dips; small centrifugal pumps are placed in these low spots to lift the water over the rise.

*Ventilation.*—Air in the mines is generally good. Drill holes that have been put down in prospecting give a circulation; if this is not sufficient, a blower is attached to a drill hole and air blown in from the surface. When headings have advanced several hundred feet from the main shaft another shaft is sunk; in addition to furnishing fresh air, this provides a safety exit from the mine in case of accident to the main shaft. High stopes tend to ventilate themselves much better than low ones.

*Labor.*—The district is located in one of the best farming districts of southern Wisconsin and there are numerous small towns within a few miles of the mines. Good public schools are also near by. A number of the men reared in the district and farm hands and farmers work in the mines in the winter. Also many foreigners work there, mostly as shovelers and trammers. Men with families live in the nearby towns, though the mining companies have built a large number of houses for their men.

## Milling

The milling of the ore is generally similar to Joplin practice but is somewhat simpler, in that no attempt usually is made to use tables or to save the slimes. To a casual observer, all the mills look about the same in appearance and design, but there are important differences when the flow sheet is consulted. They are often classified as one-, two-, and three-jig mills, or sometimes designated as 75-, 100-, or 200-ton mills, the latter referring to the number of tons supposed to be put over in 9 hr. This figure is more or less arbitrary, and in the same mill will vary under a number of factors, as the grade of the dirt, amount of water, and supervision of the millmen.

The milling problem is relatively simple. There are no complex sulfides to deal with, and much technical skill is not demanded in the work. Only three products are made, galena, blende, and tailings. Galena is often absent in the ore and the blende necessarily includes marcasite along with the black jack, the difference of specific gravity between the two being very small. It is this fact that brings the average grade of the Wisconsin concentrates down to 30-35 per cent. zinc, with many mines operating (with normal markets) on concentrates assaying only 20-25 per cent. zinc.

As is well known, there are two iron sulfides, marcasite and pyrite, identical chemically, but with some important mineralogical differences. The specific gravity of marcasite is slightly less than pyrite, which is locally known as "hard sulfur," while marcasite is called "soft sulfur." Pyrite, when present in quantity, was sometimes drawn off the jigs immediately after the lead, in an endeavor to raise the grade of the zinc concentrates. This practice was never very successful, as considerable amounts of jack were usually drawn off with the pyrite, and the product, generally assay-

ing 10 per cent. zinc or so, was sold for a nominal figure only to acid works. It has since been discontinued at all the large mills.

When marcasite is present in the ore as a pure mineral, it is very often slimed away in the milling to a surprising extent and a much higher grade concentrate obtained than would be expected from assay of the mine dirt. In most cases, however, the marcasite occurs as "sulfur rock," an impregnation of the dolomite with fine stringers and bunches of the mineral, the specific gravity of which will vary with the amount of $FeS_2$ ranging from 3.0 to 3.9. This sulfur rock often constitutes one of the most difficult problems in milling, especially with low-grade ores as, aside from the iron, it tends to run up the lime content of the concentrates, which is objectionable. Inasmuch as the sulfur rock is more of an impregnation of the dolomite than a free milling mixture, finer crushing often fails to cause the marcasite to break free from the lime, and there is no change in the specific gravity of the smaller particle but a distinct chance for loss in sliming the valuable blende and galena by the finer crushing.

One-jig mills are often constructed for the small mines, where the orebody is limited and large capital expense is not justified. They are better adapted for concentrating high-grade than low-grade ores, provided they are not overcrowded. The supervision, however, is apt to be lax and the tailings losses are generally higher than with two- or three-jig mills. Sometimes a mill is constructed with a single jig of six or seven cells, and if development work in the mine shows up well, a second and third jig are added. All of the large operating companies use two- or three-jig mills. The labor cost is usually the same for a large as for a small mill, unless the recovery is very high, when an extra helper is needed. The usual crew consists of a crusher-feeder, millman, and "backer."

Where there is no provision made for treating the slimes, the logical treatment is to avoid sliming as much as possible, which means crushing and jigging as coarse sizes as the nature of the ore will permit. The Hoskins mill, one of the newest and best equipped in the district, has been successful in milling over 26,000 cars a month, or at the rate of 1000 cars per day of two-mill shifts, with a good recovery, making a high-grade low-leaded ore, with not over 3 per cent. lime in the concentrates. Tailings losses are held below 1 per cent. zinc. A flow sheet of the mill is shown in Fig. 5. From the ore bin, with a capacity of 200 tons, the ore goes to an 18-in. breaker, thence to a trommel, where the undersize is delivered directly to the rougher jig, in order to avoid loss by any sliming through further unnecessary crushing. The oversize goes to a set of 14 by 36-in. (35 by 91-cm.) rolls. It is then elevated and put through a second trommel, having the same size holes as the first, ½-in. (12.7 mm.), where the undersize goes to the rougher and the oversize is returned.

The rougher consists of eight cells, 30 by 40-in. (76 by 101-cm.)

sieves, screen sizes varying from 7/16 to 1/4 in. (11 to 6.3 mm.). It is run at 185 r.p.m. Lead is recovered from the first cell and coarse concentrates drawn off from above the hutch. The hutch product from all cells, except the first, goes to the eight-cell cleaner jig. Clean concentrates are obtained here from the hutch, screen sizes being 1/8 in. (32 mm.) on the first, where lead is obtained, and 3/8 in. (9.5 mm.) on the others. This jig sends its overflow from the last cell to a small trommel, with 3/8-in. holes, where the undersize is sent to a four-cell sand jig and the over-

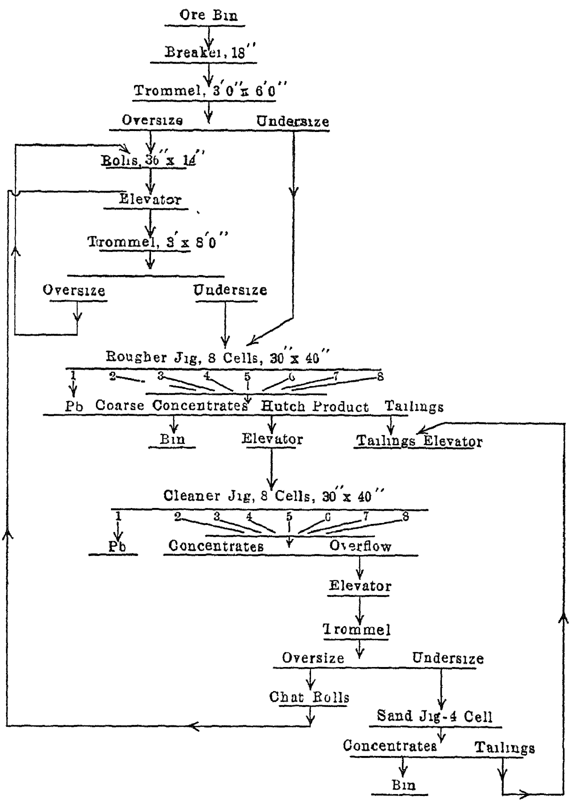



FIG. 5.—FLOW SHEET OF HOSKINS MILL, NEW DIGGINGS, WIS.

size back to a set of 10-in. (25 cm.) chat rolls, from which they are sen over the rougher once more. The sand-jig sieves vary from 3/16 in. to 5/16 in. (4.7 to 7.9 mm.), and are run with a fast light stroke at 220 r.p.m. The product is a fine, clean concentrate of high grade.

Electric power is used on a 100-hp. motor. About 700 gal. (2649 l.) of water per minute are required. As the mine provides only 200 gal. per min., the balance is obtained by saving the water from the cleaner jig, which runs to a settling tank thence to a mill pond, where it is pumped back to the mill.

A feature of the mill is the use of a solid concrete floor, all launders and elevator boots being built in concrete as well. In this way, a fruitful source of waste and loss in the old mills is eliminated, where rich mill feed was lost on the ground when a launder overflowed or an elevator broke.

No figures are on hand to show the efficiency of the mill recovery, as accurate sampling of the heads has not been attempted. The tailings loss, however, is held to a low figure, and considering the tonnage put through, the amount of slimes produced is comparatively small.

During the era of high prices, some attempts were made to remill the tailings piles of the worked out mines, where these piles showed over 2 per cent. zinc. In no case were operations undertaken on a large scale, nor was there any considerable amount spent for construction or equipment. The work brought out some interesting points in milling. It was possible to obtain exact samples of the heads and tails, and so figure the recovery, which in most cases amounted to only 50 per cent. of the zinc and 30 per cent. of the iron. This is considerably less than the recovery of the primary ore in the mill, which is estimated at 60-70 per cent. of the zinc, and 40-50 per cent. of the iron. The reason, no doubt, is that much of the zinc is in the form of fine flakes, or "floaters," which are too apt to be floated off in jigging without getting a chance to settle, particularly if much top water is used. Most of the iron is present as "sulfur rock" with a difference of specific gravity not sufficiently great to bring it down in the concentrates. Another thing found was that a high zinc assay of the pile was not so important as knowing that the jack was in recoverable form—free particles, rather than chats.

## Conclusion

The Wisconsin zinc district is a big field that is far from having been completely prospected. The possibilities of the field are best appreciated by noting, from the production table, how the output has grown from 19,000 tons, in 1904, to over 250,000 tons, in 1917. It is only within the last decade that intensive mining, large-tonnage production, and low-cost methods have been introduced, and the results have been highly satisfactory. Intelligent prospecting has succeeded in blocking out, in many cases, a supply of ore sufficient for heavy output for many years and the cost of prospect drilling is very much less than in other fields. Considering these factors, the Wisconsin field seems assured of a steady growth in importance and output for a considerable period.

## DISCUSSION

J. H. Polhemus, New York, N. Y. (written discussion*).—The Joplin mining and milling practice has largely influenced operating methods in the Wisconsin district. Milling equipment is essentially of the same type

*Received Sept. 4, 1919.

with the notable exception that no tables are used. The treatment of the fine sands has been attempted several times but without commercial success. Table material coming from any one mill is low both in tonnage and zinc content. The iron is comparatively high as the pyrite-lime "chat" is much more prevalent than the zinc-lime attached particle. In the finer sizes the zinc occurs mainly as free mineral. These conditions make the production of good-grade table concentrates very difficult. The accompanying tabulated results from several mill tests indicate the character of the mill tails. I think the average mill tails in the district assay nearer 1.5 per cent. zinc than under 1 per cent., as stated by Messrs. Boericke and Garnett.

| Mesh | Mill No 1 | | | Mill No 2 | | | Mill No 3 | | |
|---|---|---|---|---|---|---|---|---|---|
| | Per Cent by Weight | Zinc Per Cent. | Iron Per Cent | Per Cent by Weight | Zinc Per Cent | Iron Per Cent | Per Cent by Weight | Zinc Per Cent | Iron Per Cent. |
| On 10 | 42 4 | 0 7 | 5 3 | 73.4 | 0 9 | 2 9 | 66 0 | 1.7 | 4 4 |
| − 10 + 20 | 20 2 | 1 3 | 5 2 | 7 7 | 1 5 | 3 8 | 15 3 | 1 9 | 4 5 |
| − 20 + 40 | 9 6 | 1 7 | 4 9 | 3 7 | 1.5 | 3 5 | 6.8 | 1 9 | 3 6 |
| − 40 + 80 | 8 3 | 1 5 | 4 4 | 3 8 | 1.9 | 3 1 | 4 0 | 2 4 | 3 1 |
| − 80 | 19 5 | 2 5 | 6 2 | 11 4 | 4 2 | 6 4 | 7 9 | 3 5 | 5 2 |
| Total | 100 0 | 1 31 | 5 3 | 100 0 | 1 32 | 4 13 | 100 0 | 1 91 | 4 37 |
| Feed to mill | | 4.7 | 6 7 | | 6.3 | 3.8 | | 6 5 | 7 5 |

The fines offer a more promising field for increased recoveries than do the table sizes. Flotation tests have been made with very promising results. High recoveries and good-grade concentrates are indicated, the pyrite and marcasite largely remaining in the tails. The general mill practice now is to make as little fines as possible, keeping the feed to the jigs as coarse as will produce concentrates fairly low in lime. Very little regrinding is done, the coarser middlings from the rougher jig being the main product so treated. This practice often results in concentrates carrying comparatively high lime contents, the lime being present as attached particles and very fine sand or fines. If flotation could be adapted more regrinding could be done and the fines classified out of the jig feed. This would result in better grade concentrates and increased recoveries, as the fine mineral being lost under present mill practice would largely be recovered.

The majority of the mines still stick to hoisting in buckets as in the Joplin district with the hoist placed in the top of the derrick. The tendency, however, is toward the use of cars and cages, as described by the authors, as the cars are better adapted to power and mule haulage and the use of power shovels underground. The haulage problem has not received the attention it deserves, but rising labor costs and lengthened

hauls will probably lead to the discard of the mule in the near future in favor of the locomotive. With proper tracks, the ton-mile figure of 25 to 30 c. given by the authors should be almost cut in two. The scarcity of shovelers has led to the trying out of several types of underground shoveling machines. The successful use of such machines is intimately tied up with the installation of a good transportation system and a well defined mining plan. If the capacity of the shovel is held down by inability to keep enough "empties" on hand or if the shovel has to be constantly moved to be supplied with ore, there is little chance for success. A fairly accurate knowledge of the orebody, as obtained by drilling, is of material assistance in laying out the mining work. Obtaining this knowledge by the actual mining out of the ore adds to one's hindsight, but a little foresight is worth a lot of the other. Drilling will show where the mine bottom will be, thereby doing away with much "taking up of bottom," whether or not the stopes will be of sufficient height and size to warrant consideration of power shoveling, whether sublevel work will be most advantageous, and whether cheaper mining methods may not be possible. The inherent irregularities in the orebodies make any knowledge gained by drilling very valuable in preventing irregularities of operation.

In certain orebodies where minable ground from 50 to 80 ft. (15 to 24 m.) high is found, and where the ground stands well, other mining methods than those now being used should be seriously considered. If the mine is not so wet as to cause cementing of the broken ore, shrinkage stoping might be successfully applied. The necessary drifts and raises would be largely in ore and the dead work expense consequently low.

The points touched on here are not new but merely emphasize certain possibilities for lowering costs and improving recoveries.

F. J. De Wilde, Galena, Ill. (written discussion*).—As it is impossible to spare the time for a lengthy discusson of this excellent paper, I am dwelling only on points where either the information is not clear or opinion differs radically from that of the authors.

On page 218, relative to oil rock, the authors state that the presence of oil rock is always taken as a favorable indication of proximity to ore. As thick, highly carbonaceous, oil rock has been found quite often in isolated patches outside of the district proper, the mere presence of oil rock is not always an indication of the propinquity to ore deposits.

Regarding glass rock, I agree with the authors, that, generally speaking, sphalerite in the glass rock is spotty; however, they fail to mention that nearly all mines contain more or less ore in the glass rock, the usual location being near the toe of the pitch.

*Received Sept. 19, 1919.

On page 220, they say that disseminated deposits are only found in, or close to, the oil rock. To refute this statement, I refer them to a typical disseminated deposit situated at the Adam and Eve shaft of the Frontier Mining Co. This deposit is located in the central horizon of the Galena dolomite, that is, in those sedimentaries containing flint nodules. Perfect crystals of sphalerite are found disseminated throughout the rock, and are commonly known by the miners as strawberry jack. I believe all top runs, or upper deposits as described above, should be classed as disseminated; physically the ore makes its occurrence in just such a manner.

Honeycomb deposits are those that have been formed through changes in the circulation and solution of the underground waters acting upon the sheet deposits. Proof of this is the presence of remaining typical sheets within the deposit; often deposits are part sheet and part honeycomb. It might be advisable to mention that sheet and disseminated deposits have been observed as adjacent deposits, either in higher or lower horizons, and the one may lead to the discovery of the other.

On page 221, it is stated that pillars are frequently left in unpayable spots. By this, I assume the authors mean either rock barren of sphalerite, or so highly impregnated with iron sulfide that the mining of it would materially lower the grade of the concentrates.

Chamberlain's illustration of a typical sheet cross-section showing the two pitches is an exception rather than the rule; I have yet to see a typical double-pitch deposit.

The Coker mines furnish the best illustration of a flat sheet running off at right angles to the strike of the pitch. This occurrence is also true at the Blewett mine of the Blewett Mining Co. Here the flat, which is located midway between oil rock and capping, makes a right angle turn to the strike of the pitch, and carries a large sheet of jack for about 50 ft. (15 m.) the width being about 2 feet.

Extensions to orebodies, as illustrated by the Lucky Twelve mine, could also be illustrated by deposits found in Calvert ground, of the Frontier Mining Co. Four times the Calvert mine was thought to have been worked out, only to come to life again by finding another deposit separated from the last orebody by a stretch of barren limestone.

The authors say, on page 227, that when a good hole has been found, to develop the orebody the usual procedure is to cross-cut the range first, in order to determine the width of the orebody and the direction of the pitch. This statement is misleading as the direction of cross-cutting is unknown. I think we try to cross-cut the range to determine the strike of the orebody, but it is seldom that this occurs as most deposits are too irregular in height, width, direction, etc., to follow their course by this method.

Reference is made to the location of pillars as determined largely by

the character and shape of the orebody, etc. At the mines of the Frontier and Burr mining companies, pillars are left as often as it is deemed necessary to support the cap rock, regardless of whether they contain much or little ore, although barren ground or that containing a high percentage of marcasite is favored as much as possible, all things considered. Where the cap rock shows numerous crevices, pillars must necessarily be left more frequently to support the broken area. Often faulty cap rock delays operations for days, as it must be taken down, trimmed, and arched to the succeeding layer of rock above.

*Shafts and Hoistings.*—Buckets, designated as cans and varying from 1000 to 1200 lb. in capacity, are used more extensively than cars and cages; not that it is ultimately less expensive to do so but because the first cost of installation is considerably less. They are more elastic in their application to small mines. Balanced hoisting with cars and cages, as operated by the Mineral Point Zinc Co., no doubt is the cheapest and best method to use where the deposit is large.

*Breaking Ground.*—Air hammer drills mounted on columns are fast being discarded for the more easily hand-held and more mobile Jackhamer. With the Jackhamer, a drill man will drill more footage and, consequently, break more ground than with the mounted machine, as no time is lost in setting up and taking down a machine. Contractors prefer the Jackhamer. The consumption of air and the upkeep are less than with the mounted machine. The machines are best oiled by forcing oil into the air line under pressure, the operation being automatic.

*Ventilation.*—It is very seldom that a drill hole by itself will prove an efficient means for providing air to a mine heading. Blowers are used to force air into the mine workings and the larger the diameter of the hole the greater is the volume of pure air to mix with the vitiated atmosphere underground, thereby making the mine a desirable place for the men to work in, and resulting in more efficient labor. It is usual, where the workings are low and the ventilation consequently poor, to ream a 6 in. hole to 10 or 12 in. in diameter and then install a large blower to force air into the mine. Where a drill hole has been left without casing, it is seldom efficient as a means of ventilation. All drill holes should be cased from top to bottom, as often open flats will deflect the air and allow little air to arrive at the mine headings. It has been my experience that suction fans, in this district, have been uniformly unsuccessful.

*Haulage.*—It seems to me that the gasoline and storage-battery locomotives have been slighted by the authors and favoritism shown to the mule. The authors state that haulage costs when using mules are from 25c. to 35c. per ton-mile. This is excessively high when compared to mechanical haulage where the cost is seldom above 24c. per ton-mile with a limited tonnage. Where large tonnages are hauled, the average is about 16c. per ton-mile. When first used, the cost has been as low as 8c.

per ton-mile These figures include the cost of maintenance, repairs, oil, gasoline, and depreciation. The exhaust from gasoline locomotives vitiates the air considerably and it always takes some time for the men to become accustomed to the fumes. The mines must be kept well ventilated for good results.

*Concluding Remarks*—The conclusions drawn regarding the life of the field seem to me to be slightly optimistic. I agree that there are still large areas remaining unexplored, but we do not know that these areas will prove as prolific in mines as those that have been prospected thoroughly in the past. Very few sheet deposits have been discovered during the past two years, which fact is causing grave concern among the larger operators. This is a serious problem and there is no doubt in the minds of most managers that only better prices for zinc ore will stimulate prospecting and make possible the exploitation of large deposits of low-grade ore.

The phenomenal rise in production during the past five years has been due almost entirely to high prices; and only high prices will spell a steady growth in production for the district.

G. H. Cox, Rolla, Mo.—Practically all the ore of this district occurs in solution openings. Joints and crevices have controlled, to a considerable extent, the positions of the orebodies, but the cavities have been formed by solution.

It is generally assumed that thick oil rock is a favorable indication when searching for ore. The chief reasons for this are the facts that the oil-rock distribution coincides rather closely with the general mining district and the thickest oil rock is found in the central portion of the area where, in general, the mines have been the best. A careful investigation[3] of the thickness and distribution of the oil rock showed that, other than as outlined above, the evidence was not favorable to this idea. If one will measure the total thickness of oil rock in a vertical section (omitting the beds of shale and limestone with which it may be interlayered) at no place known to me will it exceed 5 ft. (1.5 m.). The average thickness for the district is less than 1 ft. There are whole townships in the vicinity of Lancaster which are practically barren of ore, yet others near Highland, underlain by a less thickness of oil rock, are heavily mineralized. The thickness of the oil rock at Fennimore and at Dodgeville is practically the same, yet little ore has been found near Fennimore while the Dodgeville area has been a large producer. Many mines are underlain by thick oil rock and many are not.

We are all interested in seeing the application of theory to practice. There are some places here where ideas as to origin are of value in prospecting. Irrespective of the source of the ore, it must be admitted that it

[3] Made in Wisconsin for the Wisconsin Geological and Natural History Survey and in Illinois for the Illinois State Geological Survey.

was carried by surface waters in their downward or laterally moving courses. It therefore follows that prospecting is a search for water courses.

Water courses are controlled chiefly by: (1) topography; (2) crevices or fissures; (3) the impervious clay and oil-rock beds; and (4) structural conditions. The topography causes surface concentrations of the water, the crevices furnish underground passageways, the impervious clay and oil rock limit the downward movement of the water, and the structure or dip determine largely the direction in which the underground water will attempt to move. At places crevices may be found by means of soil discoloration or by irregularities in topography and surface drainage. The dip may be found by actual measurement, by leveling, or by a study of the topography, for the latter is somewhat controlled by and expressive of the rock structure.

The water in the Galena tends to flow down the dip into the oil-rock basins, then down the pitch of the basins to the point of outlet. In many cases, such as the old Empire and East End mines near Platteville, it seems apparent that the ore-bearing solutions have moved along crevices, changing from one to another whenever the latter permitted them to move more nearly in the direction of the local dip, then into a crevice parallel to the first when the cross crevice terminated.

Wm. Kelly, Vulcan, Mich.—A topic that is of interest to all mining men is the proper amount to invest in the equipment of a property to accord with the character of the deposit and the value of the ore. Some 30 or 35 years ago, the Calumet and Hecla Mining Co. spent large sums of money for hoisting engines of great efficiency; some of these hoisting engines are operating today and are still efficient. In that case, the amount of ore hoisted by these machines has been very great and the cost per ton practically negligible. On the other hand, we all have to contend with the proposition as to what is the right amount to spend for equipment and the right method of operating. I have had to combat the proposition of tramming by electricity underground for the reason that our ore comes from many different shafts and several levels in each shaft, so that the amount that has to be handled on any one level does not, in my opinion, justify the expenditure necessary for an electric plant although electricity is used for all other purposes.

In the West, I have seen installations of electrical hoists with motor-generator sets, where the quantity of ore hoisted per day is quite small. While the motor-generator set is efficient and effective under certain circumstances, it is not right to use it everywhere for all kinds of hoisting. It has its own province, to which it should be limited. So, in the zinc district of southern Wisconsin, it is undoubtedly proper, under the conditions, to follow the methods there in use, and he would be a rash man who would attempt to criticize or suggest something different.

GEORGE S. RICE, Washington, D. C.—I am not familiar with the later developments in the Wisconsin District but I was under the impression, from my experience in the earlier development of the zinc sulfide mining, that there was not sufficient vertical height in the ore-bodies connected with the "flats" and "pitches" to permit the possibility of using the "shrinkage stope" system.

As concerns the mechanical loading machines referred to, I would like to ask what success has been attained in using these. At the present time the Bureau of Mines is making an inquiry in the various districts to determine how extensively and how successfully mechanical loaders are being used, the conditions surrounding their use, their efficiency, and cost per unit of ore handled in comparison with hand shoveling.

The statement was made in the paper, in reference to the use of gasoline locomotives underground, that it required time for the men to become accustomed to the exhaust fumes from the locomotives. That raises the physiological question whether men can become accustomed to breathing either large or small percentages of carbon monoxide, such as are given off by internal-combustion engines. It has generally been found dangerous to use gasoline locomotives or hoists under ground unless there is a positive ventilating current, the amount of current being proportioned to the quantity that may be given off by the engines under the worst conditions of carburation.

It has been contended that men may become accustomed to minute quantities of carbon monoxide in the atmosphere they are breathing. This is the subject of inquiry at the present time by the Bureau of Mines in investigations being conducted for certain state and municipal commissions, in the planning of vehicular tunnels. Certainly large quantities of carbon monoxide, over 0.2 per cent., could not be breathed with impunity under any circumstances.

W. F. BOERICKE.—That statement was not meant to infer that the men could become used to the carbon monoxide; it meant that, under present labor conditions, the men often use any excuse to ask for a raise in pay or to make a little trouble; the author's plan was to dilute the air. To dilute the fumes so that they will not be objectionable, he puts down a 6-in. hole and puts on a heavy pressure blow, not a suction. That dilutes the air so that the current is carried underground up the shaft and the odor is not noticeable. His experience has been that when he puts a locomotive underground, during the first week or two, the men raise a lot of trouble, and in some cases, get sick.

J. A. EDE, La Salle, Ill.—I would like to say that the Whitcock & Manchur Co. used this plan with most excellent results. I am surprised that more companies do not use it. As far as the gasoline fumes are concerned, it depends very much on whether you are on the intake or on the

outlet of the air. If you are on the outlet, the fumes will not give any trouble.

H. C. GEORGE, Platteville, Wis.—It is generally coming to be acknowledged that, in the Wisconsin zinc district, the results secured from prospect drilling are not always conclusive as to whether or not an orebody has been developed. The company with which I have been associated for a number of years has followed the plan of thorough prospect drilling, by which method an approximation of the amount and grade of ore has been supposed to be arrived at. This policy has not always developed a minable orebody, when the results of prospect drilling would indicate the presence of one. Basing our estimates upon 70 per cent. recovery of the zinc and 50 per cent. recovery of the iron secured from assays of drill-hole sludgings, which is a fair estimate of what milling recoveries show from the ores mined, we have met conditions where actual recoveries from some types of orebodies have been as low as 50 per cent. of the zinc content and 25 per cent. of the iron content from the ore milled.

This condition of low recovery in milling seems to be associated with orebodies that show uniform mineralization from top to bottom, individual sludgings from drill holes assaying, say, 4 to 7 per cent. zinc. Such orebodies are usually those that contain the mineral in a disseminated or sprangled form. Personally, I like to see a prospect drill hole show some sludgings that will run from 20 to 30 per cent. zinc, as then the orebody is usually of the typical flat and pitch formation.

There has been considerable criticism of the policy of some mining companies which, after securing several good prospect drill holes, sink a shaft without making any attempt to determine the extent of the orebody. I am coming more and more to believe that such a policy may give better information regarding the nature of the orebody than more extensive initial prospect drilling.

I recently made some tests on the sludgings secured from individual drill holes, in the horizon above the formation usually showing ore. I placed 15 lb. of zinc concentrates assaying 48 per cent. zinc and 9 per cent. iron in each of three drill holes, and then drilled 2 ft. and recovered the sludging. In the first hole, which was dry, the resulting sludging assayed 10.6 per cent. zinc and 2.9 per cent. iron; in the second hole, which had a little water, the resulting sludging assayed 8.9 per cent. zinc and 2.1 per cent. iron; and in the third hole, which had strong water, the resulting sludging assayed 6.9 per cent. zinc and 2.0 per cent. iron. These variations in recovery of known qualities of zinc, taken in connection with the numerous variations in the occurrence of the mineral in the orebody, would indicate that a shaft with development drifts should test the orebody before a costly mining and milling equipment is installed, no matter how thoroughly the orebody may have been drilled.

# Mineral Resources of the La Salle District

BY J. A EDE, LA SALLE, ILL.

(Chicago Meeting, September, 1919)

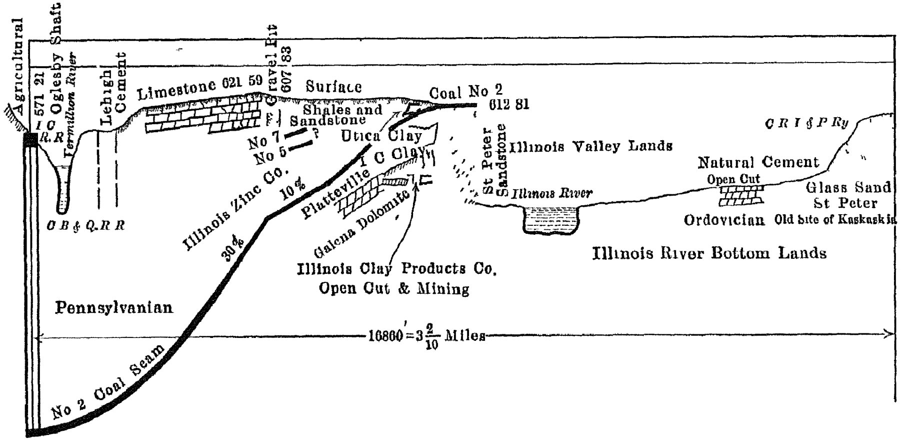
The object of the writer is to call attention to a rather unique aggregation of economic products distributed over a line of succeeding formations about 3 mi. long, to be seen within a few miles of La Salle, Ill.

Fig. 1 is an ideal section of the stratification exposed and exploited along a distance of 3⅕ mi. in a southwesterly direction from the quarry of the Utica Hydraulic Cement Co. located near the historic site of Kaskaskia village to the Oglesby shaft at Oglesby. No attempt has been made to cover the whole field, special mention only being made (with the exception of two places) of the operations found along the course or parallel to the direction assumed in the ideal section. I have used Fig. 2, after J. R. Bent and De Witt Kelly, to locate the relation of the La Salle anticline to the coal mines of this district and to show the operation of No. 1 and 3 mines of the Illinois Zinc Co.

## Anticline and the Rock Formations

Owing to the structural deformation between La Salle and Utica, the older rocks of the state are exposed to the surface along the northern embankment and the bottoms of the Illinois River valley near the latter place. At the Illinois Zinc Co. works, Sec. 16, Tp. 33, R 1, a well for water was sunk 1828 ft. and samples taken every 10 ft. of the ground penetrated; duplicates were sent to Mr. Cady of the Illinois Geological Survey. A general compilation or synopsis study of this section is here presented. As the well is over 1 mi. west of the Western limb of the anticline, the section affords some reliable data. These samples were studied and interpreted by H. M. DuBois as Lower Magnesian, and by G. H. Cady as the Lower Magnesian. The results are given in Fig. 3.

The most interesting geological feature of this district is the La Salle anticline. In 1865, Leo les Quereux, looking at Split Rock, asked how it happened that the upper coal measures overlie strata of the Lower Silurian and what has become of the formations which are generally found intermediate—the Upper Silurian, the Devonian and sub-Carboniferous, and the lower member of the coal measures?




| Agricultural | Water Power |
|---|---|
| Estimated $43,857,038 | H P 6974<br>V $1,791,000 |

| Coal La Salle County. | Portland Cement | Gravel | Clays, Brick, Tile. | Fisheries. | Quartz | Natural Cement Zinc Oxide & Pottery | Sand |
|---|---|---|---|---|---|---|---|
| P 1,151,156 ton<br>V $2,824,317. | P 3,375,786 bbl.<br>V. $4,670,745 | P 161,686 T.<br>V. $74,990<br>other than included with sand | P. clay 122,994 tons<br>V. aggr. $1,721,675 | P 23,896,000 lb<br>V $721,000 | P 260,292 ton<br>V $393,006 | P<br>V $76,427 | P 2,965,336 T<br>V $2,668,082 |
| | | | Total La Salle Co $12,911,274 | | La Salle, Bureau, and Putnam | $16,878,876 | |

FIG. 1.—LA SALLE ANTICLINE. IDEAL SECTION FOR 3.2 MILES ALONG LA SALLE MINING FIELD.

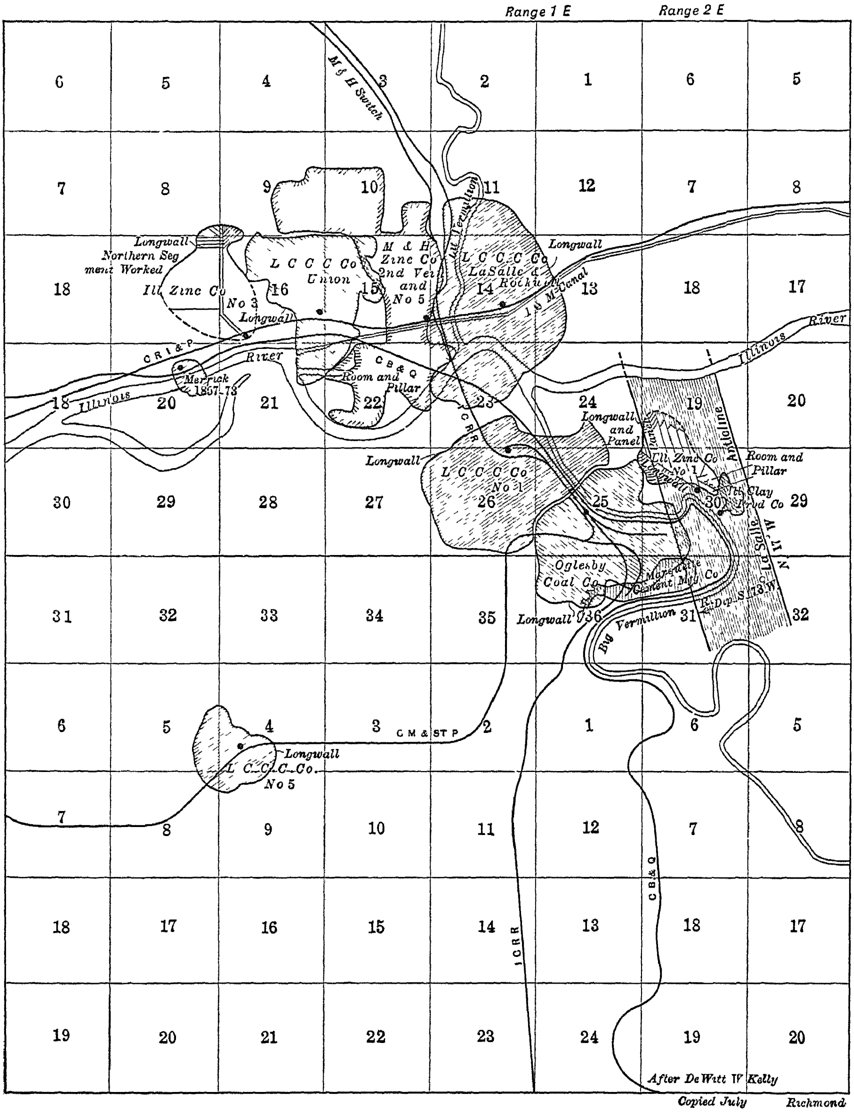



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FIG. 2.—LA SALLE REGION.

| System | Series | Formation | | | | |
|---|---|---|---|---|---|---|
| Quaternary | | | Alluvium, silt and sand | 72 | 72 | |
| Carboniferous. | Pennsylvanian | McLeansboro and Carbondale Formations | Shale gray to drab with some limestone | 86 | | Coal Portland Cement Clays |
| | | | Shale black fissile | 12 | | |
| | | | Shale gray to drab (sandy) | 78 | | |
| | | | Shale black fissile | 12 | | |
| | | | Limestone, grav, argillaceous | 3 | | |
| | | | Shale gray with coal | 70 | | |
| | | | Coal No 2 with some shale | 8 | 341 | |
| | | Pottsville formation unconformity | Fireclay. | 6 | | Coal Clays |
| | | | Shale black to grayish black | 17 | | |
| | | | Sandstone gray medium sized grains | 6 | | |
| | | | Shale greenish gray with coal | 50 | 420 | |
| Devonian? or Silurian | | ? | Shale, gray, calcareous, with some brown shale Pre-Pennsylvanian | 151 | 571 | |
| Silurian | | Niagara dolomite unconformity | Limestone, dolomitic or white dolomite | 241 | 812 | |
| Ordovician. | Upper Ordovician | Maquoketa shale unconformity. | Shale, gray to drab dolomitic | 164 | 976 | |
| | Middle Ordovician | Galena dolomite and Platteville limestone unconformity | Dolomite, straw color to tan, finely crystalline Deer Park portal to canyon. | 387 | 1363 | Limestone |
| | Lower Ordovician | St Peters sandstone unconformity | Sandstone, white rounded grains, fine to coarse Starved rock | 125 | 1488 | Glass Foundry Molding Sand |
| | | "Lower Magnesian" — Shakopee dolomite | Shale green quartz sand with gray, dolomitic limestone | 12 | | Natural Cement. |
| | | | White, calcareous sandstone | 11 | | |
| | | | Dolomite white to brown arenaceous | 163 | 1674 | |
| | | "Lower Magnesian" — Shakopee dolomite — New Richmond Sandstone member. | Sand, colorless, quartz, rounded dolomite, white, with a little sand 134–3–3 | 140 | 1814 | |
| | | "Lower Magnesian" — Oneota dolomite | Chert, white, with gray and white dolomite. | 14 | 1828 | |

FIG. 3.

In 1916, Carl Ortwin Sauer,[1] referring to the same subject, writes:

The first great recorded growth of the La Salle anticline occurred after the deposition of the Platteville, and before the formation of the coal measures as shown by the contrasted dip of these formations Deformation may have begun in the middle Ordovician period even during the Platteville epoch. The bowing up of the anticline was very slow * * * the arching may have elevated the older rocks above sea level, and exposed them again to erosion, at the same time that sediments were accumulating around the deformed areas, this would account for the absence in the anticline of beds intermediate between the Platteville Galena formation and the coal measures. * * * After the close of the period deformation affected the central area and again along practically the same axis as before, the beds were again folded, but not so severely as in post Platteville time. Perhaps this folding was a minor expression of the great movements that were then taking place in many parts of the earth, as in the Appalachian region, and which brought the Paleozoic, with its ancient form of life, to a close. With this uplift the history of marine deposition in this region closes. The great interior sea withdrew permanently and the later history deals with processes that shape land surface and not with the beat of ocean waves.

The La Salle anticline was not all developed at one time. The folding consisted of several movements extending through long periods of time which did not cease finally until long after the first beds were deformed. The Prairie du Chien shows beds which have been deformed more than any later ones. These beds and other rather indefinite data suggest the possibility of a first deformation after the deposition of the Prairie du Chien limestone. There is positive evidence of deformation between the Platteville Galena limestone epoch and the beginning of the "coal measures" period. During this interval the first great bowing up of the strata occurred. At the close of the coal measures period the beds were again deformed These two great periods of deformation find confirmation in many places along the western flank of the fold. In many places the Platteville and older formations show dips of 30°, 32° and 40° and directly overlying them are the carboniferous beds having a dip of less than 20° and commonly less than 15°.[2]

## Effect of Anticline on Mining

In 1900, the Illinois Zinc Co. opened a coal mine operated through a slope on the western limits of this anticline. The coal has been mined from a point about 2570 ft. (783 m.) from the axis, to where the dip is disappearing to conform with the normal inclination of the third vein and No. 2 seam of coal, a distance of 3263 ft. These measurements were along the course of the slope N. 60° W. and together measure 5833 ft. Fig. 4 is a section along which the mining operations referred to were extended. Owing to the seam inclining in some places along the dips at an angle of 51½°, a special method had to be adopted to mine the coal from this mine. This will be the subject of another paper; this paper will be confined to some of the unusual conditions prevailing along that part of the anticline covered by the mining operations, which may be of interest to those interested in the geology of the La Salle anticline.

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[1] Illinois Geol. Survey *Bull.* 27, 50. [2] Illinois Geol. Survey *Bull.* 27, 48.

*Coal.*—The seam is that of No. 2, better known in this district as the third vein. It varies in thickness along the extension of the slope from 2 ft. 6 in. (0.7 m.) to 5 ft. 8 in. (1.6 m.), both these measurements are exceptional, the average being between 3 ft. 2 in. and 3 ft. 4 in. and in some places 3 ft. 6 in. The hardness and general structure varied considerably as the mining progressed along the limb of the anticline, the coal in places being very hard with a conchoidal structure. The miners named it "curly coal." It used to cause considerable annoyance to the factory hands who had some difficulty in breaking it, to accommodate the producer. In order to extract the coal, blasting was necessary. The soft bottom was, however, equally responsible for having to use powder for this purpose. Its toughness or tenacity appeared to conform with the change of dip of the stratification, there being no uniform graduation from hard curly coal to bituminous normal; there is, however, some evidence of the hardening of the formation as it approaches the axis of the anticline.

There was observed some difference in the analysis but it failed to indicate any regular graduation in the constituents of the coal. The following analysis was made by Messrs. McCormack & Co., Armour Institute, Chicago:

| No | Moisture, Per Cent | Fixed Carbon, Per Cent | Volatile Combustible Matter, Per Cent. | Calcium Oxide, Per Cent | Ash, Per Cent | Sulfur, Per Cent | B t u. | |
|---|---|---|---|---|---|---|---|---|
| 1 | 16 0 | 36 81 | 45 79 | 2 18 | 13 30 | 4 10 | 12,155 | Moisture as recorded. |
| 2 | 16 63 | 41 52 | 41 56 | 2 65 | 11 42 | 5 40 | 12,256 | Moisture as recorded<br>Analysis dry sample. |
| 3 | 13 15 | 30 79 | 43 97 | 2.12 | 11 38 | 3 85 | 12,472 | Analysis dry sample. |
| 4 | 14 0 | . | . | 2 40 | 10 89 | .... | . . | Moisture as recorded<br>Ash or dry coal. |
| 5 | 16 5 | .. | . | 0 85 | 8 45 | ... | | Ash or dry coal. |
| 6 | 11 8 | . | .. | 1 35 | 14 71 | ... | .. | Ash or dry coal |

More lime was found in the normal field west of La Salle than in any of the coal analyzed on the limb of the anticline.

*Concretions.*—Interbedded with coal, in some places displacing the coal entirely, immense concretions occur within a distance of 2000 ft. (609 m.) from the axis. In an area of an acre twenty-three were encountered averaging 3.5 by 12 by 20 ft. (1 by 3.6 by 6 m.). A modified room-and-pillar method was adopted to mine the coal where they occurred.

*Clay.*—The clay under No. 2 seam of this district is termed the "mining." It varies in thickness between 6 in. to 18 in. and 3 ft.; in exceptional places it is considerably more, but the average thickness is

from 6 to 8 in. On the west limb of the anticline, the thickness was no where less than 4 ft. In the sumps sunk in different parts of the mine, the bottom of the clay was not reached at a depth of 8 ft ; when the bottom of the steep part of the incline finished, however, the clay appeared in the form prevailing in the longwall field, that is, from 6 to 8 in., increasing in some places to a greater thickness. This clay is the same as the well-known clay of Deer Park and Ottawa. From this location, the clay to the east retains an abnormal thickness, in some places being 15 ft. and over.

To the north of the slope an attempt was made to go through the upper clay where it was not very thick. Underneath the clay, forming the floor of No. 2 seam, there was a bed of sand about 10 in. thick, and below it boulders of very heavy, highly ferruginous limestone followed by clay, probably the equivalent of the Cheltenham; the depth of this was not ascertained.

## Roof of Coal

The character of the soapstone (argillaceous shale) shows considerable stress lines with slips and the draw slate is much more irregular than that found in the field where normal conditions exist. The greatest dip observed was 51½°. It was along this maximum dip we introduced our panel system. The inclination along the dip is shown in section, Fig. 5, commencing near the axis of the anticline where a drill hole showed the coal at 65 ft. (19.8 m.) from surface. Another drill hole 570 ft. along the dip showed an inclination of 5.3 per cent.

## Description of Natural Products

The rock formations are shown in order in Fig. 3. Their economic products are described in the following pages in ascending order. Overlying the hydraulic cement rocks (Oneota and Shakopee dolomite) lies the Ottawa and Utica glass and molding sand in the St. Peter sandstone; it is exposed along the bluffs of the Illinois river and mined in numerous quarries between Utica and Ottawa.

Above the sandstone are the clays mined at Deer Park, Utica, and Ottawa and also those clays below the base of the Carbondale at Deer Park presumed to be the Cheltenham equivalent. Above the clays, No. 2 seam of coal crops to the surface and is mined at Black Hollow (Illinois Zinc Co. No. 1 Mine); it is succeeded in the bluff by other seams of coal.

Some distance above No. 7 seam, the Portland cement rocks of the Marquette Cement Mfg. Co., Lehigh Portland Cement, and La Salle Portland Cement Co. are mined and quarried. Above the limestone, profitable gravel pits are quarried, some of the product being used for local purposes; and others, fully equipped with modern appliances, supply an extensive railroad business.

These successive layers of economic products are capped by a fertile black prairie soil. During the war Mr. Stevens, food administrator for La Salle County, compiled the following values as the production of La Salle County, which are published with his permission. There was produced:

| | |
|---|---|
| Wheat, 391,079 bushels, at $2.00 per bushel | $782,158 00 |
| Oats, 6,535,142 bushels, at $0.70 per bushel . | 4,574,599.40 |
| Rye, 39,241 bushels, at $1.80 per bushel . . | 70,633.80 |
| Barley, 126,003 bushels, at $1.60 per bushel . | 201,604.80 |
| Cattle, 52,624 head, at $100 00 per head. | 5,262,400.00 |
| Hogs, 102,454 head, at $30.00 per head . | 3,073,620 00 |
| For horses, poultry, bees, garden and fruit (estimated) | 4,892,022 00 |
| For corn (estimated) ...... . ... ... . . | 25,000,000 00 |
| Total . . . . | $43,857,038 00 |

*Utica Hydraulic Cement Rock, Shakopee and Oneota Dolomite*

*Production* (1918), 60,000 Bbl

The Utica hydraulic cement has been in constant use since 1838. The dams on the canal between La Salle and Chicago were built with it. The annual output at one time exceeded 600,000 bbl. according to Mr. Norman Cary, formerly president of the company. The price varied from 25 to 31c. per barrel but some was sold for 18c; labor could be procured at the time for $1.25 and $1.50 per day. Two beds of the cement rock were selected for operating, the rock was mined on the room-and-pillar system. Mr. Preuss, who had charge of the operations, says that the rooms were 35 ft. (10.6 m.) wide with a 10-ft. pillar. The height of the bed was about 7 ft. Mining drilling machines were used. The cost per yard for excavating is given at from 53 to 60 c. per cu. yd. at that time.

The modus operandi is very simple from the mine to the producer, but it requires skilled attendants at the mill to secure the best results. The rock contains a mixture of lime, silica, alumina, and iron oxide. During the burning process, the carbon dioxide of the limestone is almost entirely driven off and the lime combines with the silica, alumina and iron oxide, forming a mass containing silicates, aluminates, and ferrites of lime. The powder produced by grinding the mass is natural cement. The process is something as follows:

When starting up, the kilns, which are oblong brick kilns enclosed within an iron casing with a cubical contents of 2100 cu. ft. (58.8 cu. m.), are charged heavily with kindling wood. They are then covered with a charge of rock and coal slack, in the proportion of 7 per cent. slack to 93 per cent. rock. This charge is lighted and allowed to burn from 10 to 14 days. On the fourteenth day, cement is drawn in small quantity and on the fifteenth day the normal discharge is acquired. From the

kilns, the burnt cement rock is conveyed to a Gates crusher where it is crushed to egg size. It is then passed through a Schmitt ball-mill, thence, through a tube-mill, after which it is ready for packing, which at the Utica plant is accomplished by an automatic apparatus. The natural cement rock presents the lowest geological formation of this district. It is quite possible that this class of cement will again be called to fulfill an industrial demand; when this time comes nature has provided here an economical source of supply.

At the present time the cement rock is obtained by quarrying. Owing to the declination of the eastern limb of the La Salle anticline the limestone observed on the north side of the Illinois bluff is found on the bottom lands, and the cement bed, mined by an adit level at the former location, is beneath the surface between the Black Ball mill and Utica bridge.

### *Glass, Molding Sand, and Ground Quartz St. Peter Sandstone*

*Production,* 1,823,803 *tons, valuation,* $1,572,537, *including small production of glacial sand and gravel*

Between Utica and Ottawa there are in operation over twenty sand quarries. The sand occupies both bluffs of the valley, the stratification gradually descending east of the level of the valley until at Ottawa it is quarried below the surface level. "The manufacture of glass and brick and tile depends on the local supply of glass sand of the St. Peter's formation. The glass sandstone is soft, of even texture and may be worked with great ease in many places with pick and shovel; (*b*) the sand is of the highest quality for the manufacture of glass, being almost pure silica free from loam (*c*) with one exception this is the only commercial outcrop of this sand in Illinois; (*d*) the sandstone occurs in bluffs along the railroad lines that follow the Illinois and Fox valleys and is loaded directly from the pits into the cars."[3] The standard Ottawa sand for testing purposes comes from this locality.

The sand varies in thickness from 140 to 200 ft. (42 to 61 m.) and is classified, by E. J. Reynolds Sand Co., in the following order, from the top down.

Surface, burden, gravel varying from 6 ft. to 18 ft.
60 ft., steel casting sand oxidized, sand.
12 ft., furnace sand, stands more heat than top sand.
5 ft., welding sand, clean bed of sand.
63 ft., mixed sand.

These measurements are an arbitrary selection to facilitate regular shipment for this particular bank.

The upper bed of the "Lower Magnesian" limestone occurs beneath

[3] Sauer: Illinois Geol. Survey *Bull.* 27, 191.

the base of the sand and is reported to be very hard, for 3 ft. containing potassium. I presume the bed is the same as that studied by Du Bois and Cady, as Sample 171 of the Illinois Zinc Co., 4 ft. of which is shown in "Little greenish shale; much colorless rounded grains of quartz; much gray vitreous, semi-crystalline dolomitic limestone; little white chalky colored limestone." The analysis shows the sand to be of a very fine quality when fully prepared for market, selected samples giving silica, 99.576 per cent.; alumina, 0.283 per cent.; iron oxide, 0.0903 per cent.; lime, 0.0197 per cent.; magnesia, 0.002 per cent. About a mile to the east, an attempt was made to mine the sand; and at Ottawa the operation is that of underhand stoping and pumping. The stripping costs from 52 to 85 c. per yd. The face of the Reynolds bank is blasted down with dynamite. The holes are placed about 30 ft. (9 m.) apart and 50 ft. deep; 1400 lb. of powder generally displace 200 cars of sand, or about 8000 tons under favorable conditions. Most of the banks send the sand to market without any preparation, but a few wash it.

In one quarry where water under a pressure of 150 to 220 lb. is used, the sand is carried with the water to a grizzly perforated with ½-in. (12.7-mm.) holes after which it is forced, by a Worthington pump, to an altitude of 30 ft. (9 m.) from which elevation it flows to the mill a distance of about 500 ft. It again passes a grizzly and is flushed before it is pumped by a Knowles sand pump to the settling tanks, from which it is passed through a rotary drying kiln and stocked or sent directly to market as prepared sand.

### *Clay, Brick and Tile, Pottsville and Carbondale Formation*

*Raw clay production,* 122,994 *tons, valuation,* $193,352; *clay products valuation,* $1,528,323

In the base of the Pottsville formation there are two prominent clay beds mined and worked open cast. The upper bed which has been worked very extensively between La Salle and Ottawa, indicates the top of the Pottsville. The lower occupies, at Deer Park, see Fig. 4, an unconformable position at the top of the Galena Dolomite. This bed has been recognized as the equivalent of the Cheltenham clay of St. Louis. The Utica clay property is located in Sec. 20 Tp. 33 R. 2. The bed varies in thickness from 4 ft. (1.2 m.) to 15 ft. The clay is dumped into a chute on the side of the escarpment, forming the south side of the Illinois Valley and conveyed to the works at Utica, a distance of 2 mi. by a small tramway. At Utica, the company owns and operates eight kilns of the downdraft type. At Ottawa the same clay bed is operated on a much larger scale by the Chicago Retort and Firebrick Co.

The bed of the Illinois Clay Products Co. is near Oglesby, on the C. B. & Q. R. R., within ¾ mi. of the entrance of Deer Park Canyon. It is located where the Pottsville has a vertical section of not over 20 ft.

directly above the top of the Galena formation. The upper seam is being mined in a modified room-and-pillar system. Mr. S. O. Andros, describing the operations, states that the mine is operated by driving two parallel main entries 6 ft. (1.8 m.) high and 8 ft. (2.4 m.) wide directly

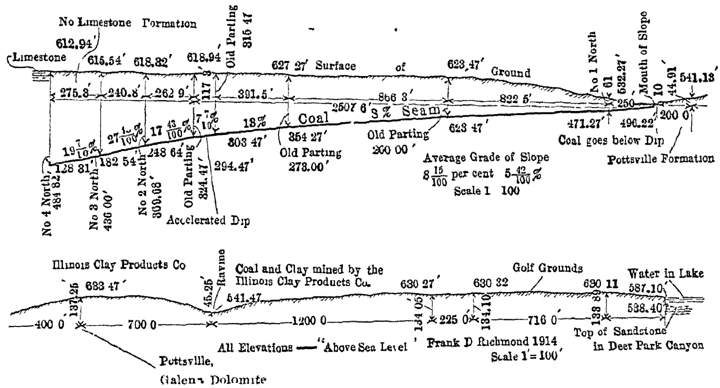


FIG. 4.

under the coal, leaving the coal as a roof. These entries are 60 ft. apart. At 60-ft. intervals along the main entries cross-entries are driven at such angles with the main entries that the grade for mule hauling is easy. This angle is usually 45° at intervals, 60-ft. panel entries are driven parallel to the main entry, the 60-ft. pillars are left for roof support

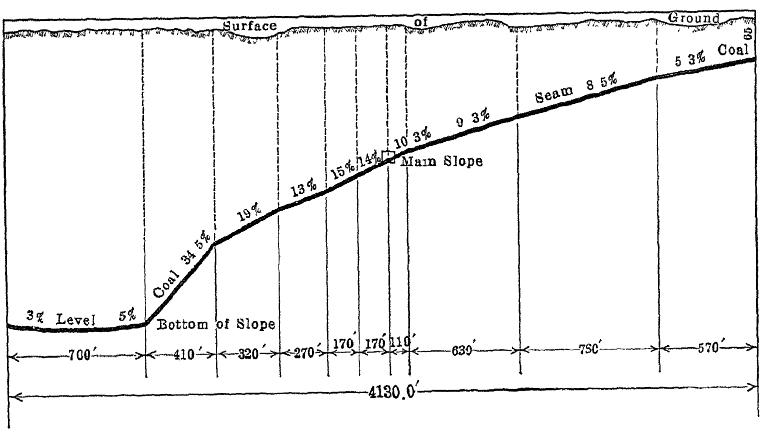



FIG. 5.—SECTION ALONG DIP, BLACK HOLLOW MINE.

and the clays taken only from the face of the entries as they advance. The clay is mined by blasting with 40 per cent. dynamite. The output of the mine is about 300 tons per day; during the last 6 years about 250,000 tons of clay has been mined at an approximate value of $500,000.

The clay is fed into an 8½-ft. Clearfield dry pan grinder, the weight of which is 53,000 lb. Two crusher rolls, weighing 8500 lb. each, are revolved by a pinion and crown overhead drive in a pan having slotted plates for the bottom; these slots are ⅜ in. wide. The material coming in is guided directly under the rolls and as it becomes sufficiently ground it escapes through the slots. The ground material is kept from accumulating by means of the scrapers attached to the pan bottom which, as they revolve

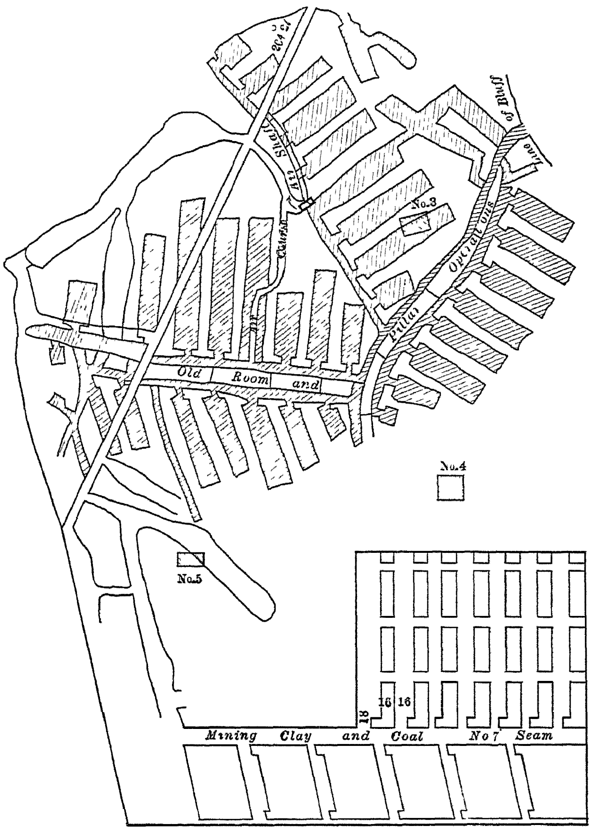



FIG 6.—MINE NO. 2, ILLINOIS ZINC CO.

with the bottom, sweep the dust round to an opening in the flooring through which it drops to a bucket elevator, which carries it to Newaygo screens that reduce the rough powder to 20 mesh. The finished product is carried over the river to the C. B. & Q. R. R. by a two-bucket, Leschen & Co. aerial tram; each bucket holds 1600 lb. The tram line requires only 10 hp. and handles 40 tons per hour. The power installation consists of a Kewanee boiler of 150 hp. and a 150-hp. Corliss Filer Stowell engine.

Under the No. 7 coal and above the La Salle quarry limestone, now

better known as the Portland cement rocks, some very useful clay and commercial shale has been mined and manufactured by the La Salle Pressed Brick Co. The clay under the No. 7 seam of coal was worked room and pillar, at the Illinois Zinc Co.'s Mine No. 2. The rooms were 16 ft. (4.8 m.) wide leaving a 16-ft. pillar. The coal was first mined in the room and the clay was taken out when the room had reached its prescribed length. The 16-ft. pillar and clay were then attacked and removed to within about 18 ft. of the entry. A pillar of 18 ft. was left each side of the entry until it had reached its limit after which it was also extracted together with the coal. The cost of mining and operative expense was 30 to 50 c. per ton for clay, and $1 per ton for coal. The current selling price at the time was 50 to 75 c. for clay, and $1.10 to $1.50 for coal. Fig. 6 is a horizontal map of the method described.

The upper clays, representing the upper series of the McLeansboro shales, are mined between La Salle and Peru by the La Salle Pressed Brick Co. The bed is about 30 ft. thick. Twenty-five men are employed with a daily output of about 50 tons when the pit is worked to its capacity.

The following are some of the principal clay concerns of this locality; Chicago Retort & Firebrick Co., La Salle Pressed Brick Co., Ottawa National Fireproofing Co., Illinois Clay Products Co., Herrick Clay & Sand Co., Barr Clay Co , Utica Firebrick Co., Streator Clay Co., Michael Gorman, Streator Brick Co., Lowell Stoneware Co., Streator Drain Tile Co. Special mention is made of these clays in Bulletin No. 30 of the Ill. Geo. Sur., page 63, with analyses and their heat behavior.

### *Carbondale and McLeansboro Formations*

*Production,* 1,151,156 *tons; valuation* (1917), $2,824,317

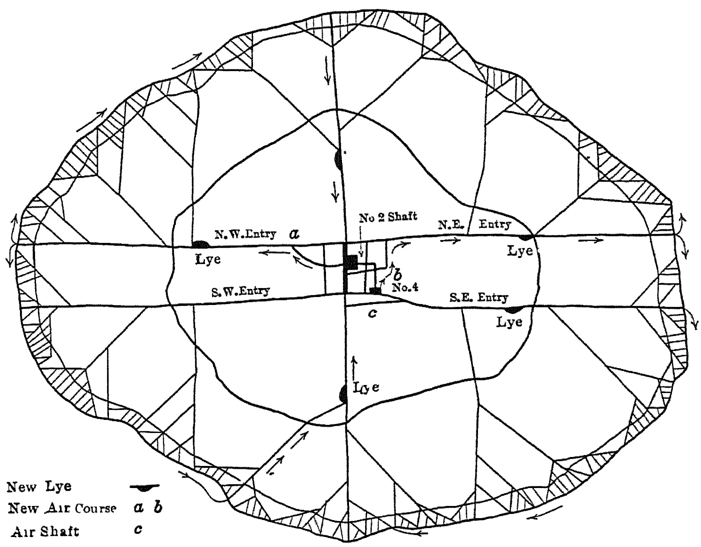
The coal mined in the vicinity of La Salle is principally from No. 2 seam, known by the trade as the third vein coal. No. 5 is worked at the Mathiessen and Hegeler and for some years No. 7 was worked in conjunction with the clay underneath it at the Illinois Zinc Co. No. 2 mine. No. 2 seam is worked on the longwall system, Nos. 5 and 7 on the room-and-pillar system.

*Longwall Mining.*—The system is so well known and has been so aptly described in repeated bulletins of the Coöperative Mining Investigations of Illinois that there is nothing that can be said that has not been covered in one or another of these bulletins.

Fig. 7 is a typical map of a longwall mine as worked in this district. The coal is found at a depth varying between 360 and 564 ft. (109 to 171 m.). The seam of coal averages 3 ft. (0.9 m.) to 3 ft. 3 in. The strata above the coal, or roof, consists of 18 ft. of an argillaceous shale overlaid by 3 ft. of a dark carbonaceous slate, called the "black slate."

During a fire at Spring Valley, this slate caught fire and was very difficult to extinguish. Immediately above the coal, there appears a small bed of slate that the miners name the draw slate. This varies in thickness from a few inches to a foot or more, and is a bone of contention between the manager and the miner. At times it falls down and mixes with the coal, and the miner is docked, that is, fined for sending up dirty coal.

Underneath the coal there is a small bed of clay from 6 in. to 18 in. thick, in occasional spots 4 ft. deep, varying in density or texture from a medium soft material to a hard substance that the miner cannot bring out with his pick. In the latter case it is named "forced mining." The floor of the coal is not even; the clay exhibits an irregular contact with the bottom




Plan of No 2 Workings, Spring Valley Mines

FIG. 7.—LONGWALL WORKING IN LA SALLE DISTRICT.

of the coal. In one room, the mining may be 6 in., in the adjoining room 18 in.; when it is 18 in. we have again "forced mining."

The coal seam, in exceptional places, becomes very thin. When the seam is reduced under 3 ft. with rolls, the term "deficient" mining is applicable. Occasionally the roof becomes full of slips or a boulder interferes with the capacity of the room; excessive bands of sulfur are commonly met with; all these conditions call for an adjustment and no day passes but some miner has a case for adjustment.

Longwall mining presumes the extraction of almost all the coal from a given area, excluding the pillar left at the bottom of the shaft, which in this locality is from 300 to 600 sq. ft. There are places where no coal is left at the bottom of the shaft. The form of longwall mining adopted

in this district is to extend the face of the coal in the form of a circle until the limit of the area to be extracted has been reached. The face of the coal is approached and maintained by a series of entries and rooms diverging from one another at an angle of 45° in the form shown in Fig. 8. The main entries are the passages for the aggregate haulage of the coal and traffic. All the coal from the side entries is brought into the main entries.

Side entries are driven from the main entries at a distance of 225 ft. apart diverging at an angle of 45° to the right or left. Along the side entries, every 59.4 ft., rooms are driven that diverge from the course of

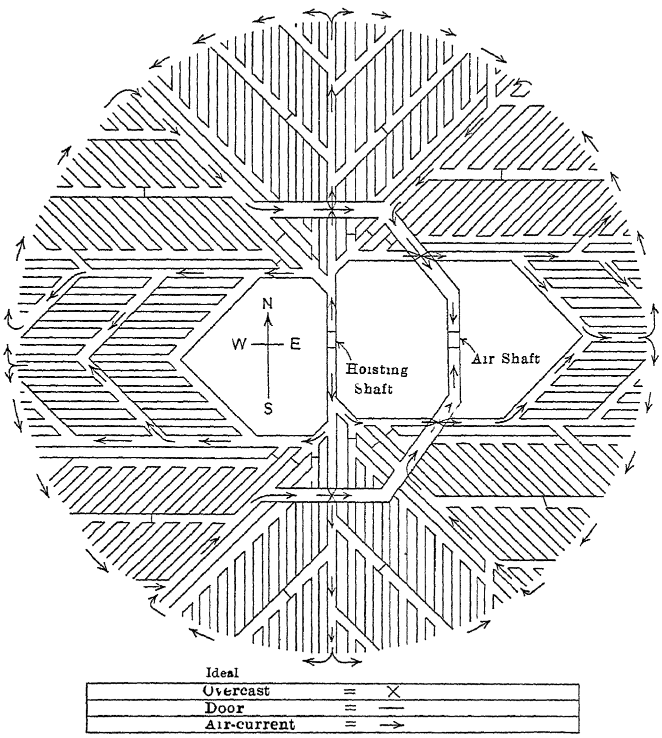



FIG. 8.—ARRANGEMENT OF ENTRIES AND ROOFS.

the side entry, to the right or left. If the side entry is to the right of the main entry, the room will be turned to the left, if the side entry is driven to the left of the main entry, the room will be driven to the right. This arrangement of main entries, side entries, and rooms presents a fan form appearance, and in their extension east and west and north and south from the bottom of the shaft the face of the coal is supposed to form a circle, but Fig. 7 shows how far from circular the face may actually be. The working faces, which may be main entries, side entries, or rooms, are placed 42 ft. apart. This is the working place of the miner; each place is supposed to be occupied by two men.

It has been found that taking down 2 ft. of the roof and removing the clay from under the seam affords sufficient material to pack a substantial gob from march to march, which is about 42 ft. The term "march" denotes the division line between the working places of the miner, that is between the rooms The roadways are not packed.

The miner is paid $1 13 per ton run of mine for all coal taken out. In the working of his room, he is expected to take down 2 ft. of the roof above the coal and build on each side of his road head, or working face, a wall extending 12 ft. With his room in this form, having a substantial building 12 ft. on each side and the rest of his room filled in with clay and refuse reënforced with props of timber arranged to support every evidence of weakness in the roof at any part of his allotted working face, he is prepared to undermine the coal ahead of him and to accommodate the weight of the strata above, on which he relies for the breaking down of his coal. In this operation he brings into play a skill that can only be acquired after a long experience. The coal has to be undermined from 2 to 3 ft., and his use of the pick for this purpose and judgment in securing his room shows his proficiency.

The output per man is from 2½ to 5 tons, 2½ tons to 3 tons per man being about the average; where that tonnage is exceeded, the conditions are unusually favorable. The mining prices paid about the mine for 1918 are as follow:

| Inside Prices | | Outside Prices | |
|---|---|---|---|
| Timbermen. | $5 00 per day | Cager. | $5 00 per day |
| Tracklayer. | 5.00 per day | Cager helper | 4 75 per day |
| Drivers. | 5 00 per day | Mule feeder | 4.75 per day |
| Timbermen helper | 4 75 per day | Coal dumpers | 4.36 per day |
| Trapper | 2 75 per day | Teamster. | 4.36 per day |
| Mine examiner | 5 54 per day | Firemen | 4.75 per day |
| Motor man | 5.15 per day | Blacksmith | 5.00 per day |
| Yardage. | 1.51 per yard for closed places | Engineer class A | 165 00 per month |
| Cogs | 10 52 including brushing branch | Engineer class B | 149.00 per month |

The air problem is solved by forcing or exhausting the air of the mine along selected entries in quantities prescribed by the state, which is 100 cu. ft. per min. for every man and 500 cu. ft. per min. for mules. The air, after being forced to the face, is split and returned along the face until it accommodates 100 men, after which it is conducted into the return air-courses and thence to the surface. A split is presumed to be made for places occupied by 100 men. The problem of ventilating some of the old mines of this field taxes the ingenuity of the mine manager.

The timbering of a coal mine is essentially the same as the timbering of a metalliferous mine, only not so complicated. Cogs and "roof props" are more conspicuous in a coal mine. In turning a room, cogs are used

for supporting the entrance and props are used wherever the roof shows any weakness. It is the business of the manager and the miner to observe closely his roof.

Very little change has taken place in the treatment of the coal at the surface since the writer described the method used at Spring Valley.[4] Since the introduction of the mechanical stoker in the industrial plants, there is a greater demand for crushed coal and a crusher has been added to some of the plants. In some of the mines of the adjoining district, the coal is washed; and during the war, for the purpose of conserving the sulfur, preliminary provision was under way in this district to save that now going to waste. Mr. Holbrook of the Bureau of Mines brought this matter forward; and there is no doubt that the sulfur bands mixed with the coal seam will in time be done away with.

It is, I believe, through the investigation of Messrs. Hazen, Swift, Bent, Conners, Smith and other men prominent in the coal industry of this district, that coal-cutting machines were introduced here. Until recently they were installed in all the prominent mines and are still being operated in restricted numbers in most of the mines, with alternating or direct current. The machines at La Salle are the Sullivan C. H. Regardless of the fact that experience in this district has shown that, with the present price paid the miner for machine work, the cost of mining coal is considerably above that of pick mining, negotiations are pending between the operators and miners for the purpose of formulating a more equitable adjustment of the differential between pick mining and coal-cutting machines. In considering coal-cutting machines, it should be borne in mind that the price paid for undermining of the coal is only one factor in the equation. The following should be considered: The differential between pick mining and machine mining, replacement of man power, rock distribution, regulation of face of coal, and increased output.

*Differential Between Pick Mining and Machine Mining.*—The price of pick mining in this district is $1.13 per ton, as agreed between the Coal Operators' Association and the United Mine Workers of America. The price for machine mining is 72 c. per ton for loading; 14 c. per ton for cutting, the company to pay for the brushing and other cost.

*Replacement of Man Power.*—There was striking evidence of the need of a substitute to replace men during the war; at one time the men were leaving almost every day for more remunerative fields in the south. During the last 20 years, a radical change has taken place in the personnel of the coal miner of this field. The following is a classification of the nationality of the miners who worked in Black Hollow mine, June 1, 1901 and Mar. 1, 1919.

[4] *Trans.* (1900) **29**, 187.

1901

| Nationalities | Number | Per Cent |
|---|---|---|
| American | 16 | 31 |
| Welsh | 4 | 8 |
| English | 22 | 43 |
| Scotch | 6 | 12 |
| Irish | 1 | 2 |
| Swedish | 2 | 4 |

96 per cent. Americans or natives of British Isles.

1919

| Nationalities | Number | Per Cent. |
|---|---|---|
| American | 18 | 19 |
| English | 1 | 1 |
| Swedish | 4 | 4 |
| Scotch | 2 | 2 |
| Polish | 4 | 4 |
| Lithuanian | 12 | 13 |
| Belgian | 22 | 24 |
| Austrian | 21 | 23 |
| Italian | 9 | 10 |

22 per cent. Americans or natives of the British Isles, 78 per cent. other nationalities.

There are not in this district the number of expert longwall miners there were in 1901; these miners have moved to districts where they can earn more money. A few, however, remain and their daily performance emphasizes very strongly the loss to the community of those who have migrated to other fields. A longwall miner cannot be educated to his work in a day. His skill is enhanced by his heredity. A really expert longwall miner, that is, a man who knows how to cut his coal, build his road, secure his top, and fill his gob to the best advantage must have muscles specially adapted and a sensitive intuition of his environments. To the operator it is a matter of profit or loss whether his mine is worked by such miners or simply by novices.

These qualifications of a miner for longwall work are given because it is in the absence of such men that we find compensation for the coal-cutting machine. The value between the expert longwall miner and one just used to dig coal is minimized considerably by the introduction of the machine. The machine cuts the coal and much of the intuitive knowledge of the miner in this particular performance is necessarily not so essential. There appears, however, in this change of method a demand for a more technical and mechanical mind, an intelligence that presents a useful field for the employment of vocational training. It is necessary to have strong men to load the coal and competent brushers to break down the rock and keep the rooms in working order. As stated, coal miners are migratory and often leave when the need for them is most urgent; the coal-cutting machine costs comparatively nothing while idle but when needed replaces a large number of men. The em-

ployment of machinery in the excavation of coal corresponds with the trend of events in other departments of industry where machinery replaces manual labor and these machines have come to stay in the longwall mines.

*Rock Distribution.*—A stranger visiting this field and not familiar with the "modus operandi" of a longwall coal mine is curious to know what are the pyramidical mounds that dot the surface of this district. The mounds are rock taken from the mine for which no value, up to the present, has been found and represent a dead cost to the operator. The amount of rock taken from different mines varies, depending on the character of the roof and the seams of the coal. An ideal longwall method assumes that sufficient space is left after mining the coal to compensate for all the rock taken out as brushing together with the clay (called mining). Owing, however, to quite a number of reasons such a condition is seldom attained. Owing to a deficiency in the seam, such as an abnormal thickness of clay at the bottom of the seam, forced or deep mining conditions in the roof cause more rock to come down while brushing than is desirable. Falls on the entries, development work in the solid in the way of improvement or maintenance, and caved rooms furnish more rock than can be accommodated under ground.

From data furnished by some of our mines the amount of waste taken to the surface is reduced one-half where machines are employed. The amount of rock taken out varies in different mines. In some it is as high as 30 per cent. of the coal output, in others 15 per cent., and in others 10 per cent. Where the roof is exceptionally good, it is 5 per cent.

*Regulation of Face of Coal.*—The method adopted in this district for mining the coal is called "longwall advancing." It is modified at No. 1 and No. 3 mines of the Illinois Zinc Co., to overcome structural conditions and to accommodate the demand of the zinc plant. It is the ambition of the mine manager to keep the circle of the face of his coal with its radius and coördinates as uniform and symmetrical as possible; in practice this is not accomplished.

When the coal face is irregular with humps, elbows, and protrusions, it can be presumed the coal is not breaking well and it is a fair inference that the miner in that locality is demanding extra pay. This condition comes from no fault of the management, it can occur in many ways: It may be caused by the men working in the same section differing in efficiency; one man mines more coal than the other and pushes his place along until he has formed a pocket. It may be caused by some men working their places more regularly than their neighbors. It may be the result of structural and physical conditions of the roof or coal, for which we have only theoretical reasons to explain the cause. Or it may be due to a deficiency in one part of the face with exceptionally favorable conditions in the other; in the first case, the coal may have clay slips and

forced mining or excess of draw slate while the other place will have high coal and ideal conditions. If coal-cutting machines were used only for the purpose of straightening up the face of the coal and keeping the circle in a symmetrical shape, they would compensate very fully for their installation.

*Increased Output.*—With machines a larger output can be furnished. The cost per ton is regulated pro rata by the amount of the output. During the last few years, the hauling system in these mines has been much improved. The electric trolley and storage locomotive have been substituted for the use of mules, although the shaft in many of the mines is over 1 mi. from the working face. The conveying of the coal from the face is a much more simple problem than it used to be. At the La Salle County Carbon Coal Co., Union mine, an electric hoisting engine has been installed and the mine very fully provided with electrical equipment.

### *Portland Cement, McLeansboro Formation*

*Production, shipments,* 3,375,786 *tons; valuation* (1917), $4,610,745

At La Salle and along the bluffs of the Vermillion River, a conspicuous feature is a massive limestone underlaid by a bed of shale and a small seam of coal. This limestone is No. 10, 11, 12 of Freemans section, named by him the La Salle quarry limestone, now better known as the Portland Cement rock. At the present time it is operated at three places by the La Salle American Portland Cement Co., at La Salle, and the Lehigh Portland and Marquette Cement Mfg. Co. at Oglesby. At the Marquette plant the rock is mined and at the other two places quarried. Messrs. Bleininger, Limes, and Layman, in Bulletin 17, Ill. Geol. Sur., have described the Portland cement resources of this district very fully. Mr. T. Shultz, works manager of the Marquette Cement Mfg. Co., and Mr. F. W. Armstrong of the Lehigh Portland, submit the following description.

*Operation of Marquette Cement Mfg. Co.*—The plant of the Marquette Cement Mfg. Co. is located at Oglesby, Ill., about 6 mi. southeast of La Salle. This company is making Portland cement, using limestone from the La Salle limestone and shale from a deposit directly below this limestone for its raw materials. The operations of the company are unusual in that the raw materials are secured through an underground mining operation rather than from the usual open quarry. Mining is necessitated by the heavy overburden, which varies from 60 to 120 ft. The La Salle limestone is a nearly horizontal deposit varying in thickness from 28 to 35 ft. (8.5 to 10.6 m.) on the property of the Marquette Cement Mfg. Co. It is not a pure limestone but contains varying quantities of shale; it is sufficiently pure, however, to make it usually necessary to mix shale with it to secure the proper raw mixture for the manufacture of Portland cement.

The mine of the company is opened on the room-and-pillar system. Development is effected by main entries from 22 to 25 ft. (6.7 to 7.6 m.) wide by 14 ft. (4.22) high, with cross-entries of the same size opened at intervals of about 900 ft. (274 m.). The working rooms are opened perpendicular to the cross-entries and are 40 ft. wide by 21 ft. high, with a continuous rib 20 ft. thick between the rooms.

Drilling is done with hammer drills actuated by compressed air with a pressure of 90 to 95 lb. per sq. in. at the drill. The mining operation is carried on in two benches. The top bench is about 12 ft. high and is broken by a center V-cut side holes breaking to this relief. The bottom bench is stoped out from the same rigging of the drills; the top bench is always kept one cut in advance. The average cut is from 7 to 8 ft. All loading is done with steam shovels operated by compressed air, No. 20 and special No. 25 Marion shovels being used, with air receivers replacing the boilers. The larger shovels normally used in the rooms are equipped with 1½-yd. dippers. The rock is loaded into 8-ton steel mine cars and hauled by electric locomotives to the crushing plant.

The rest of the process is that usually employed in the manufacture of Portland cement. The rock from the mine is reduced in jaw and gyratory crushers, Ball mills and Fuller mills to a fineness of 80 per cent. or more through a 200-mesh sieve. This powder is then calcined and clinkered in rotary kilns using powdered coal as fuel. The resulting clinker is cooled, the proper proportion of gypsum added, and the finished cement produced by grinding the mixture in Kent mills and tube-mills to a fineness of 81 to 82 per cent. through a 200-mesh sieve. This finished cement is then stored in bins until required for shipment. All sacking of the cement is done automatically by Bates packers and the sacked cement is loaded in cars for shipment. The normal manufacturing capacity of the plant is about 6300 bbl. of cement per day, the maximum shipping capacity is about 16,000 bbl. in 8 hours.

The entire mill is electrically driven, with individual motor drivers on all units. The company makes its own power in a well-appointed power house. The boiler equipment consists of 12,400-hp. Stirling water-tube boilers with chain grate stokers and operated under induced draft. The electrical equipment consists of five direct-current generators driven by cross-compound condensing engines. The rated capacity of the generators is 5600 kw. at 250 volts. Three air compressors, having a combined capacity of 6300 cu. ft. free air per minute furnish compressed air for the mine and mill.

*Operations of Lehigh Portland Cement Co.*—The quarry of the Lehigh Portland Cement Co. at Oglesby, Ill., lies between the junction of the Illinois and Big Vermillion Rivers; at the present rate of consumption there is sufficient rock in sight to last, approximately, 95 years. The first operation consists in stripping off the overburden, which has an aver-

age depth of about 16 ft. Underneath this overburden is a 30-ft. stratum of limestone, which is worked by 70- and 95-ton shovels. Underlying this rock is a 16-ft. ledge of clay or shale and midway in this ledge is a coal measure of from 1 to 3 in. This coal measure is used as a floor upon which the work of the 45-ton shovel used for excavating clay is performed. Only the top 8 ft. of this clay is worked to facilitate loading, as it is possible, with this scheme, to have tracks for cars on top of the clay face. Both rock and clay are loaded into 4-yd. dump cars and carried to a McCully Mammoth 42-in. gyratory crusher. At this place, the mixing of the rock and clay is done; that is, so many tons of clay to a given weight of rock. After passing through the crusher, the mixture is screened through a 3-in. circular screen, the fines going into a pan conveyor leading into rock bins and the oversize being elevated and crushed through a Jeffrey hammer mill, after which it is also conveyed to the rock bins.

The rock is blasted with dynamite in well-drilled holes based on 6-ft. (1.8 m.) centers. It has been found that about 4½ tons of rock is produced per pound of powder consumed. The average yearly tonnage of the quarry, when working at full capacity, is about 400,000 tons; the mix consists of about 16 per cent. shale and 84 per cent. rock, which gives an average content of $CaCO_3$ in the mix as fed to the kilns of 76½ per cent. The face of rock worked at present is about 1 mi. in length.

*La Salle Portland Cement Co.*—This mill, located in La Salle, has a daily capacity of 4000 bbl. Its open quarry has modern equipment and is in the same geological zone as the Marquette and Lehigh. There is unusual stripping with back filling in the quarry; a steam-shovel loader of both limestone and shale. A Fairmount roller crusher, 36 by 60 in., is used for the preliminary reduction and a Williams mill and tube-mill for the final reduction of the raw material. There are three 10 ft. by 164 ft. rotary kilns, Sturtevant mills, Kominuters and tube-mill for finishing. The electrical power is purchased.

*Sand and Gravel.*—A typical gravel pit is that of the Western Sand & Gravel Co. of Spring Valley. This plant consists of a drag line arrangement by means of which material is drawn from the pit and dumped into a car on an incline that runs to the top of a tipple, where the car is dumped into a revolving screen, and all material larger than 1½ in. is rejected at the end of the screen and then passed to the crusher, to be reduced to the required size. Across the pit is stretched a cable to which is attached a movable sheave through which the rehaul line runs by moving this sheave either way. The bucket can be started wherever desired. The bucket holds from 1½ to 2 yd. of material and its contents is dumped into a trap opening which leads into the car beneath. From the face to the tipple is 300 ft. The capacity of the pit, based on an 8-hr. day when pulling from a distance of from 100 to 150 ft., is about 900 tons.

# Titaniferous Iron Sands of New Zealand

BY V. W. AUBEL,* CLAIRTON, PA.

(Chicago Meeting, September, 1919)

AMONG the iron-bearing ores of the world, the titaniferous iron sands of New Zealand are probably the least known to American engineers. This is not surprising in view of the fact that American ironmasters look with disfavor on any, even domestic, ores containing over 1 per cent. titanium. The possibilities of smelting titanium-bearing ores in the blast furnace are still undetermined, but the problem of their efficient reduction is becoming more important each year. There seems to be little inclination on the part of American furnace operators to tackle this problem, and they will no doubt continue to utilize the non-titanium ores at their command, leaving research and experiment to those countries not so fortunately endowed. New Zealand has, however, been intensely interested in her deposits of titanium-bearing iron sand, and during the last 50 years considerable English and New Zealand capital has been expended on this problem.

The extent of the New Zealand iron-sand deposits is almost beyond measure. From Patea north to Manuka Heads, along almost the entire west coast of the North Island, these deposits exist. Wherever there is beach sand along this 300 mi. of coast line, the sand is iron bearing. It varies considerably in grade; but wherever the contour of the coast affords small bays and inlets, large dunes are found that are remarkably uniform in quality. Several of these large deposits, especially those at New Plymouth and Patea, are noted for their high iron content. The dune at New Plymouth runs between 55 and 60 per cent. of metallic iron; the panning action of the prevailing westerly winds and northwesterly currents has no doubt resulted in this concentration. Whatever the origin of this material may be, there is no doubt that deposition is going on continuously. At New Plymouth, a small artificial harbor, protected from the drifting sand by a large breakwall, has been dredged. Here sailing vessels have for years been loaded with iron sand for ballast by means of a crane and grab bucket operated from the wall of the breakwater. Thousands of tons of sand have been removed from this spot yet each succeeding storm renews the supply. The sand dune shown in Fig. 1 has been tested to a depth of 50 ft. (15 m.) without losing its

*Assistant Blast-furnace Superintendent, Clairton Works, Carnegie Steel Co.

high quality. The beach itself is of high iron content at a depth of 30 feet.

The origin of this ore is generally accepted as being subterranean and volcanic. Whether the volcanic action is still proceeding or whether this sand is the accumulation of an extinct volcano is in dispute. The New Plymouth district is, however, the western extremity of a volcanic belt, that extends across the North Island, the eastern end is the well-known active thermal region of Rotorua, see Fig. 2. New Plymouth is well connected by rail with both Wellington and Auckland and its deposit of sand is easily accessible by both rail and water.

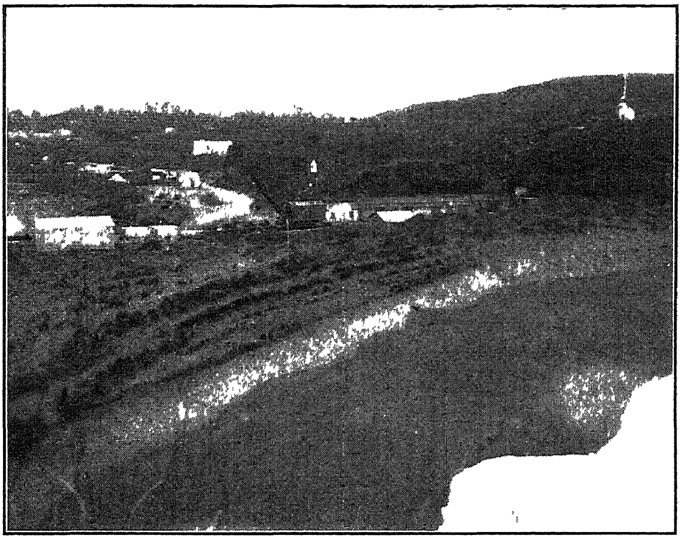


FIG 1.—SAND DUNE AVERAGING 55 PER CENT. METALLIC IRON.

New Zealand is comparatively only a young country, settlement having commenced about 1840. A great many of the first settlers, being from the iron-producing centers of Great Britain, were at once attracted by the nature of the sand on the beaches. When it was discovered that this sand contained a high percentage of iron, considerably higher even than the ores obtainable in the famous Cleveland district of England, companies were formed to work this material. Early attempts were made with the Catalan forge, using charcoal as fuel; small quantities of wrought iron were made in this way and some of the specimens are at present in the Auckland Museum. The first attempt to smelt the sand in a blast furnace is recorded as having been made in 1865 in England. No details are available except an analysis of the sand used, and also of the product made. This iron is reported to have been puddled, and from

the resulting wrought iron, hoops, bars, chains, etc. were made, and pronounced excellent in quality.

Analysis of Iron Sand Used in English Test

| | Per Cent |
|---|---|
| Iron | 59 2 |
| CaO | 1 75 |
| MgO. | 1 00 |
| $TiO_2$ | 15 2 |

Analysis of Pig Made

| | Per Cent |
|---|---|
| Sulfur .. | . 0 274 |
| Phosphorus | . Trace |
| Silicon | 1 40 |
| Carbon . | 2 40 |
| Titanium. | 0 18 |

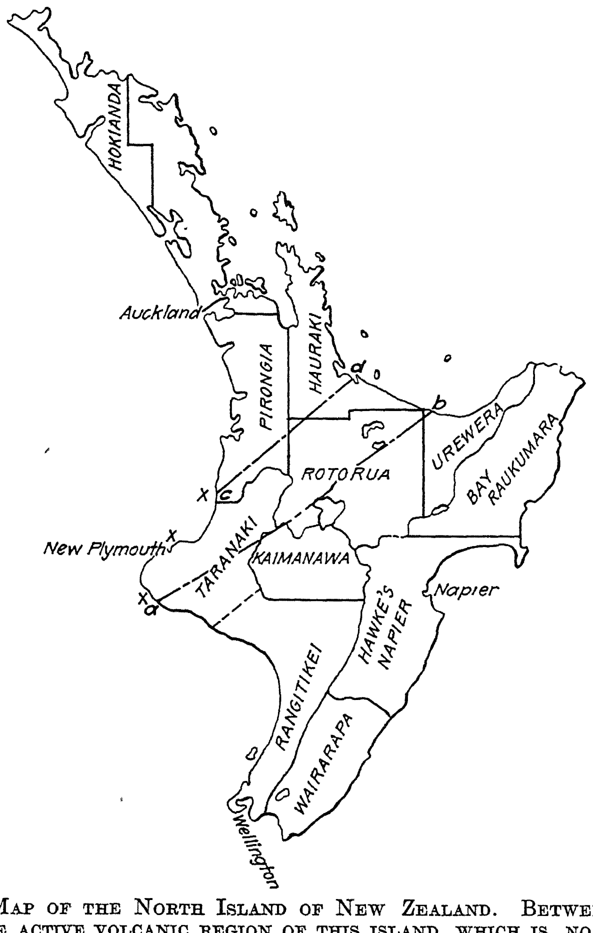



Fig. 2.—Map of the North Island of New Zealand. Between lines *ab* and *cd* is the active volcanic region of this island, which is, no doubt, the source of the iron-sand deposits. Places marked *x* are the principal deposits. The currents on this coast have scattered this sand in large quantities practically all the way to Auckland.

About 1868, a small blast furnace was built at New Plymouth and the iron sand, mixed with clay and baked into bricks, was charged into the furnace with charcoal as the fuel. This test was evidently not very

successful. The next attempt of any magnitude, and the one perhaps that deserves the most mention, was made at Onehunga about 1884. Considerable interest was being aroused at this time in England over the possibilities of iron sand. As Canada and New Zealand were both rich in this mineral, the matter was brought to the attention of the foremost steel and iron men of that day. Sir William Sieman, just before he died

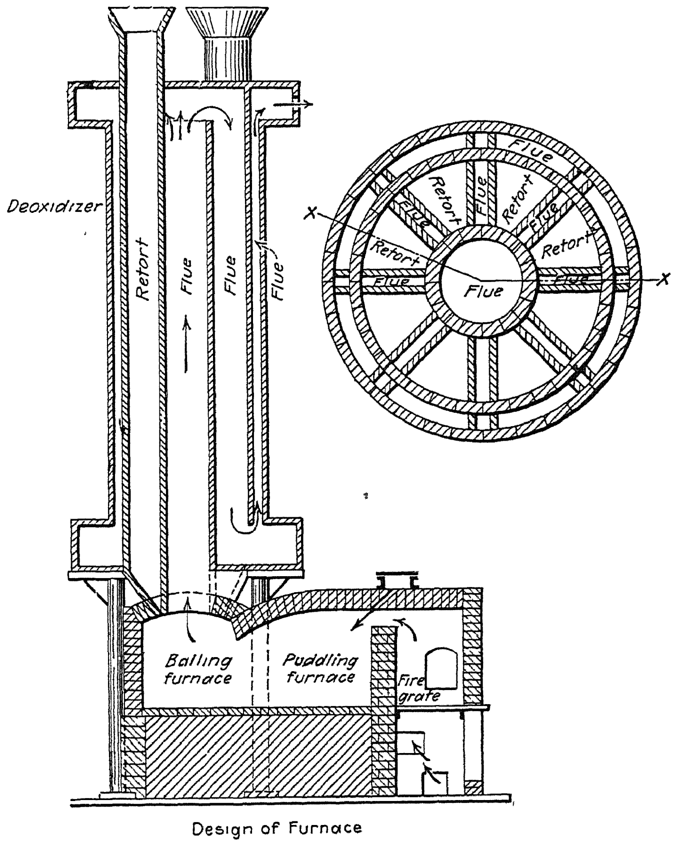



FIG. 3.—FURNACE FOR THE PRODUCTION OF WROUGHT IRON DIRECT FROM IRON SAND, USED IN EXPERIMENTS WITH IRON SAND AT ONEHUNGA, NEW ZEALAND, IN 1886–7.

in 1883, developed a combination balling furnace and open-hearth furnace which was to be used to treat the Canadian iron sand. Sir Henry Bessemer also became interested and carried out some laboratory experiments. He reported that the successful treating of iron sand would necessitate a great deal of research work, and that he was too old to undertake it. About this time, however, the furnace shown in Fig. 3 was patented in America and a company formed in New Zealand received contracts for

the erection of several of them. The iron sand was to be mixed with coal and charged into the shaft of this furnace. After sufficient time had elapsed for the charge to become deoxidized by the heat of the gases burning in the surrounding flues, it was dropped on to the hearth and puddled or balled in the usual manner. One unit of this plant, with the necessary bar mill and finishing mills, was completed in 1884. As the attempt made by this company to produce wrought iron direct from the iron sand was not successful, the company dissolved after nearly 3 years of experimenting.

During the next 25 years a great many companies were formed for the exploiting of these sands. Some of these made honest efforts to utilize the iron sand, while others were of a distinctly wildcat nature. The operations of some of these wildcat companies have caused considerable prejudice against this material both in England and New Zealand.

In 1914, a company was formed for treating the iron sand for the production of pig iron by a process patented in New Zealand. The sand from the beach was dried, put through a magnetic separator, mixed with a large proportion of fine coal, and coked in fireclay retorts. The product from the retorts, termed ferrocoke, was smelted in a large cupola with a little limestone for flux but without the addition of any extra fuel. The amount of fixed carbon in the ferrocoke was to be kept great enough to supply sufficient heat and carbon for the reduction of the already partly deoxidized iron sand This process was tried out in a variety of ways for nearly 3 years but the iron produced was very inferior. Such a process has little to recommend it from a metallurgical point of view and it is not surprising that it was abandoned. The scarcity of pig iron in New Zealand, and the resulting high price due to the war, encouraged another attempt to use these sands; late in 1917, it was decided to build a small hand-filled blast furnace, equip it with hot blast, and experiment still further.

The utilization of such an ore of iron as this titaniferous iron sand for the production of pig iron in the blast furnace presents two very difficult problems. The first is the physical condition of the ore; being in the form of fine beach sand, there is obviously the necessity of briqueting, sintering, or otherwise preparing the sand for charging into the furnace. The second is the necessity of either eliminating the $TiO_2$ content of the sand wholly or in part by magnetic-separation milling methods, or attempting to slag it off without interfering with the continuous operation of the furnace or the production of a marketable iron. In this respect attention was naturally directed to the possibility of reducing the $TiO_2$ content by magnetic separation. The measure of success, of course, of such a method depends on the existence of free ilmenite in the sand.

Six samples of sand of varying iron content were taken:

| Sample Number | Iron Per Cent. | $TiO_2$, Per Cent | $P_2O_5$, Per Cent. |
|---|---|---|---|
| 1 | 49.56 | 9 2 | 0.64 |
| 2 | 59 92 | 10 6 | 0 65 |
| 3 | 23 97 | 3 6 | 0 53 |
| 4 | 20 22 | 3 0 | 0 50 |
| 5 | 12 43 | 2 2 | 0 39 |
| 6 | 36 4 | 6 2 | 0 28 |

These samples were tested with both a strong and a weak hand magnet; the results of these tests on samples 1 and 6 are given. Samples were also ground to fine powder, and tested under water with the magnet, but in no case was the $TiO_2$ reduced appreciably under 9 per cent.

| | 1 | | | 6 | | |
|---|---|---|---|---|---|---|
| | Ordinary | Strongly Magnetic | Feebly Magnetic | Ordinary | Strongly Magnetic | Feebly Magnetic |
| Iron ...... | 49 56 | 56 2 | 4 7 | 36.4 | 49 3 | 4.1 |
| $TiO_2$ ....  ....... | 9 2 | 10 3 | 6.6 | 6 2 | 7.6 | 5.7 |
| Phosphorus. ....... | 0 28 | 0.32 | .... | 0.28 | 0.32 | ... |

The results from this series of experiments were scarcely encouraging. Sample No. 2 was taken from that part of the beach which constituted the ore supply for the furnace and was used also when testing the sand with the magnetic separator for the elimination of the $TiO_2$.

Average Analysis of Sand from New Plymouth Deposits

| | Per Cent | | Per Cent. |
|---|---|---|---|
| Metallic iron ... | 56 22 | MgO. .......... | 2 80 |
| $SiO_2$ ..... . . | 6 20 | $TiO_2$ ....... ..... | 10 30 |
| $Al_2O_3$ ..... .. | 2 00 | $P_2O_5$. . ....... .. | 0.74 |
| CaO .. . . . | 0 62 | $V_2O_5$ ... ... ...... | 0 39 |

Experiments were conducted on a large scale with the magnetic separator, which was part of the original equipment. This machine consisted of two pairs of rolls operating between two pairs of fixed magnets. The sand was fed and distributed to the rolls in a uniforn stream by a revolving feeder. The magnets were excited by a small generator, driven by the motor that operated the rest of the machine. They could be adjusted both as to strength of current and as to length of magnetic field. The dry sand dropping from the rolls in a uniform sheet was deflected on either side by the magnets; the tailings, or non-magnetic material, dropped on to a conveyor belt and discharged outside of the building. The magnets were excited and adjusted from a very weak magnetic field to a very strong magnetic field, with no appreciable lowering of the titanic-acid content of the concentrates. Not even with a very weak field and the consequent loss of iron in the tailings did the titanic acid in the concentrates drop under 9 per cent. In this respect the New Zealand sands differ materially from the St. Lawrence river iron

sands of Canada. These latter, although they are of considerably lower grade, lend themselves easily to wet magnetic concentration with a product containing from 65 to 70 per cent. of metallic iron and only 2 to 3 per cent. of titanic acid.[1]

Because of the small amount of money available it was necessary to use as much of the old equipment as possible. It was impossible to secure any standard type of sintering or briquetting machine even if the product could be guaranteed. A roll type of egg briquetting machine and a roaster of the company's own design were, therefore, used to sinter and prepare sufficient of the sand for the contemplated test, see Figs. 4 and 5.

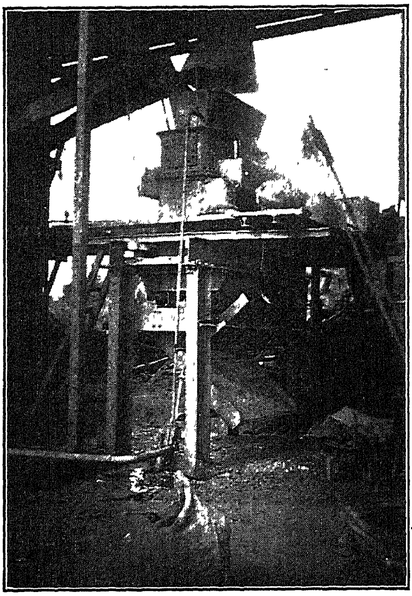


FIG. 4.—VIEW OF ROASTING MACHINE.

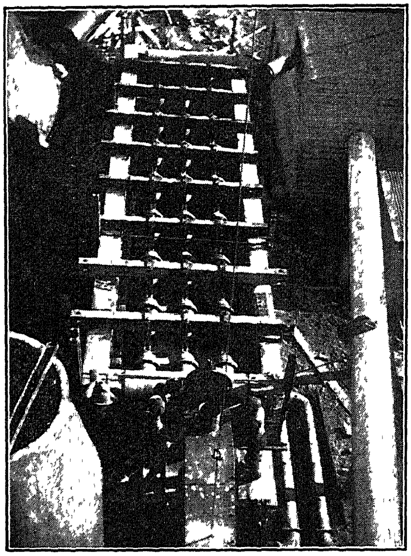


FIG. 5.—PARTLY COMPLETED HOT-BLAST STOVE.

In the first experiments with the eggette machine, the attempt was made to use the raw sand from the beach mixed with 12 per cent. of ground coal. The pressure exerted by the rolls, however, was not sufficient to bind this material and the proportion of good eggettes was probably not over 50 per cent. An effort was then made to grind the coal and the sand together; this gave a much better yield of good eggs and led to the purchase of a 33-in. Fuller grinding mill.

The treatment of the sand finally adopted was as follows: The sand was loaded from the beach into small side-tipping narrow-gage trucks; fed by bucket elevator into a rotary coal-fired kiln and discharged by a

[1] Magnetic Iron Sands of Natashkwan, Canada Dept. of Mines (1912) 145.

screw conveyor; elevated by bucket chain and fed to the distributor of the magnetic separator; the dried concentrated sand was mixed with 12 per cent., by weight, of low-sulfur coal, elevated and fed to a Carr disintegrator, and discharged into a storage bin; fed from bin by a screw conveyor feed to the Fuller mill, all the product from the mill passed through a 100-mesh sieve; discharged from Fuller mill, slightly dampened with water, into an enclosed mixing conveyor and elevated by bucket belt to storage bin. From the bin the material was fed by a Hendy roller feeder to a vertical pug mill acting as a mixer and feeder to the eggette machine; eggettes were discharged over shaking screen on to a conveyor belt running to feeding platform of roasting furnace then hand-fed into blast roasting furnace, automatically discharged over screen, and stored for smelting in the blast furnace.

FLOW SHEET

Raw wet sand
Rotary dryer
Magnetic separator
Waste tailings — Concentrates (60 per cent. Fe 10 per cent $TiO_2$) 12 per cent. coal added
Carr disintegrator
Storage bin
33-in. Fuller mill
Conveyor mixer (dampened with water)
Storage bin
Hendy feeder and vertical pug mill (water added)
Eggette machine
Broken eggettes and fines back to bin by elevator — Eggettes by conveyor belt to feeding platform of roaster
Blast roaster (coke breeze added)
Fines back to storage bin } Roasted eggettes to furnace bins

Sufficient electrical power was not available to operate all of these units continuously so the process was divided into three stages and operated as follows: Stage one, including all operation as far as the storage bin of the Fuller mill, and stage three, including eggette machine and roaster, were operated on a day shift of 8 hr.; stage two, or the Fuller mill alone, was operated on a night shift of 8 hr. In this way about 15 tons of material was put through each 24 hr. Without including an engineer and a couple of millwrights, only three men were necessary to operate stage one, two men for stage two, and four men for stage three. Of these units, the dryer and separator were on the yard level of the blast furnace. The disintegrator, Fuller mill, eggette machine, bins, etc. were housed in a large single building on a level about 50 ft. (15 m.) above the yard level, or approximately on a level with the blast-furnace charging platform. The roaster was just outside of this building and adjacent to the blast-furnace storage bins. This machine was made of

steel plate, flanged and riveted so as to be water-jacketed throughout. It was in two sections, separated by a 2-in. (5 cm.) slit, which acted as tuyeres, and was connected to a 12-in. Roots blower. The grate consisted of two rolls, turned in opposite directions by means of a tooth-and-pawl arrangement. The desired speed was obtained by a lathe-head cone pulley and a connecting rod fastened to the face plate of a lathe-head stock. To start this furnace, it was first filled to the tuyere level. A wood and coke fire was then built and the blast turned on very lightly. In 10 min. the furnace was ready for charging with raw eggettes; little coke breeze was needed as the material already contained 12 per cent. of coal. The success of this roaster depended on the ability of the operator to keep the temperature down to a red heat, otherwise the eggettes would fuse and run together and large lumps would form and interrupt the continuous operation of the discharging rolls. With the amount of blast constant and the feeders instructed to keep the roaster full, it was possible for the operator to regulate the temperature by the speed of the discharge rolls. The resulting eggettes were hard and also slightly porous, due no doubt to the carbonization of the coal.

The blast-furnace plant, as completed, consisted of a small furnace of about 15 tons daily capacity, a 24-pipe hot-blast stove, and a 24-in. Connersville smelter blower. The blower was either direct driven by a 70-hp. vertical steam engine or belt driven by a 75-hp. motor. The furnace was 46 ft. (14 m.) high, 4 ft. (1.2 m.) in diameter at the hearth, and 9 ft. (2.7 m.) in diameter at the bosh. The bosh was spray cooled and pierced for six 3-in. tuyeres; the tuyere coolers were of cast iron with coiled cooling pipes. Several cast-iron cooling boxes were inserted in the tuyere area. The furnace shell was of ½-in. plate stiffened by rails bolted on the inside. The top was closed by the usual cone and bell, with a 24-in. bleeder leading off the downcomer. Two burners led the gas to the combustion chamber of the hot-blast stove. This stove was 40 ft. long, 10 ft. wide, and 14 ft. high. Twenty-four 10-in. diameter U-pipes were suspended in three rows of eight each. The walls were lined with firebrick and insulated with a 3-in. layer of granular pumice. All the units of the plant were electrically driven except the blower.

The analyses of the ore used, the coke, which we obtained locally, and the limestone are as follows:

| Eggettes | Per Cent. | Limestone | Per Cent. | Westport Coke | Per Cent |
|---|---|---|---|---|---|
| Iron | 50.0 | CaO | 54 1 | Fixed carbon | 93 55 |
| CaO | 2 8 | $SiO_2$ | 4.0 | Volatile hydr | 1.85 |
| $Al_2O_3$ | 1.8 | $Al_2O_3$ | 0 9 | Water | 0 25 |
| $TiO_2$ | 9.2 | MgO | 3 17 | Ash | 4 35 |
| $SiO_2$ | 5.7 | | | Ash includes sulfur | 0 85 |
| MgO | 2.0 | | | and phosphorus | 0 035 |
| Carbon | 10.0 | | | | |

It is at once seen that, excluding the titanic acid, the amount of slag-forming ingredients is very small, and it would be extremely hazardous to attempt a run without diluting this titanic-acid content to some extent by the addition of some foreign slag-forming materials. This material was found in a local volcanic rock, which it was thought would furnish the necessary ingredients to make those complex silico-titanates of lime and alumina considered necessary to flux away any great quantities of $TiO_2$. This rock also contained a considerable amount of alkalies, which are well known to form easily fusible compounds with titanic acid. An analysis of the rock as used is:

| | Per Cent | | Per Cent |
|---|---|---|---|
| $SiO_2$ | 53.3 | Mn | 0 86 |
| $Al_2O_3$ | 20 4 | Alkalies | 6 6 |
| CaO | 8 1 | $Fe_2O_3$ | 8 1 |
| MgO | 0 36 | | |

This test run was not so much a question of production or cost as an attempt to secure some practical data. Naturally, under such conditions, the furnace operator would keep a high slag volume per ton of pig, and also a low burden of ore per ton of coke. Starting off in this way with $TiO_2$ very low in the slag and working up to a normal burden, and then working down on the slag volume, with a consequent rise of $TiO_2$ in the slag, until the point of economic balance had been reached, would, of course, mean running over a considerable length of time to obtain satisfactory results. The furnace was filled to start as follows:

Four charges coke, 1500 lb. each.

Five charges, each: coke, 1500 lb.; limestone, 636 lb.; silica rock, 600 lb.

Three charges, each: coke, 1500 lb.; limestone, 730 lb.; silica rock, 700 lb.

Two charges, each: coke, 1500 lb.; ore, 500 lb.; limestone, 810 lb.; silica rock, 700 lb.

Three charges, each: coke, 1500 lb.; ore, 800 lb.; limestone, 760 lb.; silica rock, 600 lb.

Starting with these charges on a coke unit of 1500 lb. (680 kg.), the ore was gradually increased to 2000 lb. and the amount of the silica rock and limestone was gradually decreased in order to bring up the percentage of titanic acid in the slag. The first day the speed of the blower was

brought up to a maximum of 60 revolutions, which gave an average blast pressure of 20 oz. at the blower and about 14 oz. at the furnace. The average blast temperature was controlled at approximately 1000° by regulating the supply of gas to the stove. The furnace was cast and flushed regularly the first 5 days. The first slag was essentially a silica-lime slag, as only what $TiO_2$ was contained in the silica rock was present. All the slags obtained through the cinder notch from the beginning to the end of the run were extremely fluid, more so than the average blast-furnace slag. On the fifth day, one of the tuyeres was burned and before the trouble was located, due to the high-water pressure and relatively low-blast pressure, the furnace had secured considerable water. On the cast succeeding the changing of this tuyere difficulty was experienced in getting the tapping hole open; the remaining 5 days of the run the furnace seemed to be getting steadily worse in this respect. It was often necessary to burn the tap hole open with oxygen, and the hearth bottom seemed to be getting higher each day. During this time the cinder obtained through the cinder notch was hot, gray, and very fluid, while the slag obtained through the tap hole at the end of the cast showed none of these very desirable characteristics. The tap-hole cinder seemed to be darker, heavier, and to have a tough sticky nature. Analysis of this material is given with that of the other data obtained from this run. On the last 2 days of the run, considerable iron was tapped through the cinder notch. Outside of the hard tap hole, iron at the cinder notch, and building up of the hearth bottom, there were none of the indications of a bunged up furnace. The iron was hot and gray and the cinder was hot and very fluid. On the seventh or eighth day of the run, the brickwork around the tuyere zone on one side of the furnace began to steam and moisture oozed out around the joints in the brickwork. The coil-cooled cast-iron cooling blocks were suspected of leaking so the water was shut off of these. On account of the bunged up condition of the furnace and suspicion of water leaking, it was decided to "blow out" on the tenth day.

The furnace had made about 90 tons of iron. Some of the analyses are given. Test bars were made from some of this pig and tested in Auckland, with the results given. On cleaning out the furnace, it was found that the hearth had built up about 2 ft. from its former level; the analysis of the salamander is given. One of the tuyere coolers was found to be leaking and the furnace no doubt obtained considerable water from this source and also from the blocks. From this test we had wished to observe the quality of the iron obtained, the nature of the slag in relation to its $TiO_2$ content, and to obtain evidence to either prove or disprove the contention that $TiO_2$ segregated in the furnace hearth. Due to the water trouble encountered, the data obtained from this run could not be considered conclusive, so it was decided to have another test of at least a week's duration.

The following analyses were made by the New Zealand Government Dominion Laboratory.

REPRESENTATIVE ANALYSES OF IRON OBTAINED ON THE FIRST RUN

| | Silicon, Per Cent. | Titanium, Per Cent | Phosphorus, Per Cent | Vanadium, Per Cent. | Sulfur, Per Cent | Manganese, Per Cent | Combined Carbon, Per Cent | Graphitic Carbon, Per Cent. |
|---|---|---|---|---|---|---|---|---|
| Iron from first cast | 8 18 | 0 98 | 0 73 | 0 11 | 0 007 | 0.28 | 1 53 | 2 04 |
| Iron from fourth day. | 1 63 | 1 69 | 0.45 | 0 03 | 0 030 | 0 31 | 1 34 | 2 07 |
| Iron from seventh day | 1 49 | 0 39 | 0.56 | ... | 0 080 | 0 44 | 1 15 | 2 80 |
| Iron from ninth day . | 1 60 | 0 45 | 0 57 | . | 0 090 | 0 38 | 1 14 | 2 79 |

REPRESENTATIVE ANALYSES OF SLAGS OBTAINED ON THE FIRST RUN

| | $SiO_2$, Per Cent | $Al_2O_3$, Per Cent | Iron, Per Cent. | Ferrous Oxide, Per Cent | CaO, Per Cent | MgO, Per Cent. | $TiO_2$, Per Cent | Alkalies, Per Cent. |
|---|---|---|---|---|---|---|---|---|
| First slag obtained | 43 | 20 | | | 32 60 | | | |
| Slag on third day | 32 40 | 15 17 | | 1 15 | 35 80 | 6 36 | 7 09 | 2 03 |
| Slag on sixth day .. | 31 29 | 13 90 | .. | 2 73 | 37 97 | 6 28 | 7 72 | |

Analysis of sticky, tough slag secured from tap hole toward the end of cast was

| | PER CENT | | PER CENT | | PER CENT. |
|---|---|---|---|---|---|
| $SiO_2$ | 15.36 | Ferrous oxide | 1.53 | $TiO_2$ | 9 06 |
| $Al_2O_3$. . | 15 48 | CaO . . | 18 63 | $V_2O_5$. . . | 0 58 |
| Iron . | 35 17 | MgO.. | 2 49 | | |

This material was of such a pasty nature that small pellets of metallic iron was carried with it mechanically mixed and no doubt accounts for the high iron content in the analysis. There was about 2 ft. of the following material in the bottom of the hearth; it was very hard and flinty, and showed the presence of considerable $TiO_2$ by its color:

| | PER CENT | | PER CENT | | PER CENT |
|---|---|---|---|---|---|
| $SiO_2$.. | 12 70 | Ferrous oxide. . | 5 20 | $TiO_2$ | 19 06 |
| Iron . | 43.70 | CaO.. | 9 04 | $P_2O_5$ . | 0 44 |
| $Al_2O_3$. . | 6.74 | MgO .... | 2 44 | | |

There was probably only about 3 bu. of material carried over and found in the dust catcher; its analysis is as follows:

| | PER CENT | | PER CENT. |
|---|---|---|---|
| Iron.. . ..... | 38.86 | Soda. .. ... .. ...... | 0.78 |
| $TiO_2$.. . . | 5.78 | Carbon.... . . .. .. | 7.14 |
| Potash . .. ... ... . | 1.74 | | |

Without considering the effect of any water that, no doubt, had leaked into the furnace both through the leaking tuyere cooler and through

several cracked plate coolers, it is at once seen, both from the operation of the furnace and from the analyses of the slags and the material found built up in the hearth bottom, that all of the $TiO_2$ in the charges was not finding its way out of the furnace.

In preparing the furnace for the second run, every possible care was taken to avoid any repetition of the previous water trouble. The bronze tuyeres were made heavier in construction and given a larger water feed at a lower pressure. The cast-iron coil-cooled blocks were dispensed with entirely, under the theory that no cooling would be necessary in the tuyere area for such a short run. New and better tuyere coolers were provided. The second run lasted 11 days, but there was no trouble from water leaking into the furnace. At the completion of the run, all the cooling bronze was as sound as at the start.

The furnace was filled to start in the same way as for the first run. With a coke unit of 1000 lb., the ore charge was gradually increased to 1800 lb., limestone to 448 lb., and silica rock to 270 lb. The furnace was flushed and cast regularly, no trouble being experienced until the fifth day. During the first 5 days, the iron at no time ran under 2.50 per cent. silicon. On the fifth day some trouble was experienced in getting the tap hole open; on this cast it was impossible to get any slag to follow the iron. The next cast was a very small one, and from then on all slag and iron were tapped together out of the cinder notch. About the eighth day, a hole was cut above the tap hole at the same level as the slag notch and from this hole all the slag and iron were tapped together at regular intervals until the end of the run. No further building up of the hearth occurred; if anything the hole was lowered to some extent by this method of tapping. On the fifth day, when the trouble was first experienced with the tap hole, the iron obtained was white; no change was made in the furnace burden from that indicated above. White iron was obtained on the sixth, seventh, and eighth days but after the tap hole was cut in the front of the furnace, the iron turned gray and the silicon content jumped at once to over 2.0 per cent. Good, gray iron was the rule from this time until the end of the run.

From our experience with these two runs we have drawn a few conclusions that may be of interest. The difference between the slag obtained from the cinder notch and that obtained from the tap hole at the end of the cast was emphasized before. This difference was very marked on the second run. After the cinder notch was abandoned, on the second test, and all the slag and iron were run out together from the improvised tap hole, a large sample of the slag was taken in a mold and allowed to cool. On breaking this sample a very distinct line of demarkation was shown between two formations. The bottom layer had a specific gravity of 3.8 and the top layer a specific gravity of 3.2. The bottom layer was of a very much deeper blue color than the top, and showed plainly the pres-

ence of more $TiO_2$. From this it would appear that when flushing slag out of the furnace through the cinder notch, we are obtaining slag with the low specific gravity and segregating the $TiO_2$ in the hearth. This would account for the vast difference in the appearance of the slag from the cast and that from the flush. It would also appear that when all the slag and iron are taken at regular periods from the same tap hole a greater percentage of the $TiO_2$ is eliminated.

On the ninth day of the second run the supply of prepared eggettes was exhausted. It was, therefore, decided to try a few days' operation with the raw beach sand in place of the prepared material. The raw sand was wetted down with a 5-per cent. solution of clay wash, just sufficient to prevent filtration and also to prevent excessive loss in the dust catcher. The coke, sand, limestone, and silica rock were thoroughly mixed before charging into the furnace. Practically the same weights were used as when using the prepared material. This practice was followed for the two remaining days of the run, or until the stock of coke was exhausted. No difference was noticed in the workings of the furnace. The iron obtained was high in silicon, averaging about 2.40 per cent. It is unfortunate that we were unable to obtain sufficient coke to keep on with this second run. It was the consensus of opinion, however, of those in charge that it would have been no trouble whatever to have run the furnace indefinitely. Further efforts along these lines are to be made to utilize this material, and preparations are under way for a more extended trial of the furnace. When further trials are completed more information may be available.

These test bars were approximately 1 in. square in section, rough as cast and rough cleaned. Unfortunately I am unable to find, in my notes, the length of these bars and the distance between the bearing points

Results of Tests Made with Cast-iron Bars on New Zealand Government Machine

| Test Specimen | Deflection Inch | Pounds to Break | Remarks |
|---|---|---|---|
| 1 PSS[a] | | 2900 | Flaw in casting at breaking point. |
| 2 PSS | 0.5 | 3250 | Small flaw in casting. |
| 3 PSS | 0 5 | 3530 | Bar not broken. |
| 1 P | 0.5 | 3400 | Broke at small flaw. |
| 2 P | 0 5 | 3500 | No break. |
| 3 P | | 3550 | No break. |
| 1 M | | 3550 | No break. |
| 2 M | | 3550 | No break. |
| 3 M | | 3550 | No break. |

[a] PSS indicates proportion of two parts of good machinery scrap to one part of pig iron; P indicates all pig iron, M indicates three parts of pig and one part of scrap.

when testing, but I think the bars were 24 in. long and the bearing points 18 in. At all events, the men who represented the New Zealand railways department, and who tested these bars, were unanimous in declaring that they had stood up better in these tests than any bars they had tested in routine work.

## DISCUSSION

F. E. Bachman, Port Henry, N. Y. (written discussion*).—Experiments with titaniferous ores found in Essex County, New York, made by the MacIntyre Iron Co. in 1914, showed[2] that titaniferous concentrates are reduced in the furnace with no greater, and probably with less, expenditure of heat, and consequently of fuel, than non-titaniferous magnetites; and as a greater proportion of their oxygen is removed by carbon monoxide than is the case with non-titaniferous magnetites, a lower fuel consumption for the reduction of iron may be expected.

After the furnace was blown out at the end of the experiment, the formation in the hearth was found to be abnormal. The brickwork was replaced, entirely around the furnace, by a layer which at the tuyere circle was made up of cinder, fine carbon, and coke; the coke, however, gradually disappeared toward the bottom, being replaced by cinder and graphite.

The usual salamander of graphite and iron was missing. The material left in the hearth seemed to be cinder that had not drained from the last cast; its composition, though, would indicate that it was in the furnace previous to the taking off of the titaniferous ore. At first, this material was thought to be blast-furnace cinder mixed with less iron and graphite than is normal; but an examination showed it to be made up of cinder with small masses of iron and graphite. There were also patches of a steel-gray material having a metallic luster; a microscope showed these patches to be made up of black metallic crystals interlacing white crystals.

As there had been no indication of the furnace hearth having been filled with unmelted material, and as iron and steel bars had been driven several times into the hearth of the furnace to a depth of at least 18 in. (45 cm.) below the top of this layer, this material must have been present in the furnace in a melted state. Determinations of its melting point and fluidity tests indicated that it had a low melting point but did not become fluid enough to flow at furnace-hearth temperatures. As its specific gravity was apparently very much below that of cast iron, I concluded it was present in a layer resting on the iron. All analyses of this material indicate excessive amounts of sulfur and titanium.

*Analyses of Salamander.*—Suspecting that this material caused the "dirty hearth" condition, which was the only abnormal condition not

* Received Sept. 28, 1919.

[2] Amer. Iron and Steel Inst. *Year Book* (1914), 371–419.

met and overcome during our run on titaniferous ores, I sent some to Mr. Porter W. Shimer, Easton, Pa., for analysis. His report is as follows:

The piece was broken down coarsely and sampled. A portion weighing 103½ gm. was ground down, with much difficulty, so that 96 gm. passed through a 100-mesh sieve; 7½ gm. of small iron particles remained on the sieve. The part that passed through the 100-mesh sieve was used, in its air-dried state, for the full analysis, which follows:

| | Per Cent | | Per Cent |
|---|---|---|---|
| Metallic iron | 23.16 | Magnesia | 2.49 |
| Titanium | 29 34 | Calcium sulfide | 13.61 |
| Vanadium | 0.15 | Potassa | 0 20 |
| Chromium | 0 00 | Soda | 0.27 |
| Nickel | 0 00 | Combined carbon | 3.39 |
| Manganese | 0 13 | Graphitic carbon | 0.94 |
| Silica | 10 06 | Nitrogen | 3.86 |
| Alumina | 4 54 | Moisture | 0.32 |
| Lime | 7 02 | | 99.48 |

Sulfur (by fusion), 6.05 per cent, sulfur (by evolution), 6 02 per cent., confirmed by duplicate.

The water-soluble was 2 15 per cent. The sulfur in this was 0 949 per cent., equivalent to 2 14 per cent. calcium sulfide. This seems to show that only a small part of the calcium sulfide, and nothing else, was dissolved out by boiling with water for some time. By dissolving in hydrochloric acid (1.12) I found approximately 58 per cent. to be soluble and 42 per cent. insoluble. The soluble part consists of about 24 parts of iron in minute particles and 34 parts of slag. The insoluble part consists of about 40 parts of titanium carbide, nitride, etc, and 2 parts of an almost insoluble silicate showing transparent colorless sand-like grains under the microscope.

Partly by direct determination and partly by calculation from the analysis, I have arrived at the following figures. The iron in the salamander was analyzed on the sample of 7½ gm. which remained on the 100-mesh sieve It contains silicon, 1.39 per cent.; sulfur, 0.071 per cent., total carbon, 1.57 per cent., confirmed by duplicate.

The slag, figured out as best it could be done, is as follows:

| | Per Cent. | | Per Cent. |
|---|---|---|---|
| Silica | 26.77 | Potassa | 0.52 |
| Alumina | 11.79 | Soda | 0.70 |
| Lime | 18.22 | Titanic acid | 0 21 |
| Magnesia | 6 46 | Sulfur | 15.70 |
| Calcium sulfide | 35 33 | | |

The titanic acid soluble in hydrochloric acid (1 12) is figured in the slag. It is surprisingly low. The insoluble carbide and nitride residue has the following composition:

| | Per Cent. | | Per Cent. |
|---|---|---|---|
| Titanium | 67.88 | Nitrogen | 8.66 |
| Iron | 0 55 | Silica and silicate | 11.60 |
| Carbon (combined and graphitic) | 11.24 | | |

The insoluble silicate, which in one case amounted to as much as 2.67 per cent. of the total, was analyzed with the following result:

| | Per Cent |
|---|---|
| Silica | 44 91 |
| Alumina | 30.71 |
| Lime | 14 61 |
| Magnesia | 9 00 |

In reply to a letter making further inquiry, Mr. Shimer says: "The sulfur in the salamander slag is far higher than anything that I have ever found in furnace slag, though I understand that such slags are usually much higher in sulfur than the regular blast-furnace slag. The thought occurred to me that some of the sulfur might exist as titanium sulfide, but a determination showed that this is not the case. That it occurs in the slag only is also shown by the fact that the fusion sulfur and evolution sulfur agree closely.

"As to the $TiO_2$ in the slag, it is possible, as you suggest, that it remained insoluble in the hydrochloric acid and appears with the titanium in the insoluble residue. Remembering the very unusually high sulfur in the slag and the presence of carbon and nitrogen in the titanic residue, it seems to me quite possible that the slag may really be quite low in $TiO_2$."

The material, therefore, is undoubtedly a mixture of blast-furnace slag, exceptionally high in sulfur, iron, free graphite, and a mixture of titanium carbide and nitride in about equal quantities; which of the nitrides has not been determined. It seems probable that the nitride consists of a mixture of about equal parts of $Ti_2N_2$ and $Ti_3N_2$. The excess of sulfur leads one to expect the presence of titanium sulfide, but Mr. Shimer states that none was present and that the sulfur undoubtedly exists as calcium sulfide. Experience has demonstrated that slags containing 9 per cent. of calcium sulfide are anything but fluid, and that no furnace can be operated with a slag of such a sulfur content as is shown by the above analysis. Not having determined the sulfur contents of slags found in furnace salamanders, nor having seen an analysis published, I wrote to several furnace managers asking for information as to their sulfur contents. All replied that they had never analyzed nor seen analyses of such slags. Mr. Geo. Collard, general manager of the Shenango Furnace Co., subsequently analyzed slag from a salamander from a furnace operated on Lake ores and Connellsville coke but found no excess of sulfur.

From the information at hand, I was unable to decide whether, if the material caused the dirty hearth, the trouble arose from the presence of the calcium sulfide or from the titanium compounds. The melting point of calcium sulfide has been determined to be above 1700° C. (3200° F.) while the melting point of the titanium compounds, obtained by dissolving the slag and iron from them by hydrochloric acid, and separating the graphite as well as possible by decanting the lighter substances, could not be determined.

During casting, pieces of salamander were placed in the iron runner between the dam and skimmer. They floated on the iron and became mushy, but did not become fluid enough to spread over the surface of the pool of iron. A piece of titaniferous cinder and a piece of salamander were clayed to the bottom of the cinder trough. The cinder melted almost completely while one flush of cinder ran over it; the salamander was melted to the same extent after two and one-half flushes ran over it.

*Location in Furnace of Salamander-forming Material.*—The following investigation was made to determine the probable location of material forming the salamander in the furnace hearth, while the furnace was producing iron, and to determine the result arising from the introduction into the furnace hearth of substances that should liquidate the calcium sulfide or oxidize the titanium compounds, or both.

The material to be treated was placed in crucibles made from carbon electrodes, which were then placed in a furnace similar to a crucible-steel pot furnace, using blast. Coke fuel was used. A quantity of titaniferous cinder, iron borings, and salamander was crushed, carefully mixed, and sampled, and the sulfur content of each determined, as were the $TiO_2$ in the cinder and the titanium in the iron and salamander. The crucibles were kept covered during the melt to prevent oxidation. The mixtures were melted as rapidly as possible and, after melting, were held in the furnace 2 hr., at approximately hearth temperature, then withdrawn, cooled in the crucible, and examined. The sulfur and titanium content of the iron and the sulfur and titanic oxide of the slag were determined. Melts Nos. 1 and 3 were lost, and reported under other numbers. The analyses were made by Mr. Wm. V. Knowles of the Titanium Alloy Mfg. Co. They are as follows:

| | Sample No 9 2A Slag | Sample No 10 4A Iron | Sample No 11 4A Slag | Sample No 12 No. 5 Iron | Sample No. 13 No. 5 Slag | Sample No. 14 No. 6 Iron | Sample No. 15 No. 6 Slag |
|---|---|---|---|---|---|---|---|
| Analysis number | 2016–A | 2017 | 2017–A | 2018 | 2018–A | 2019 | 2019–A |
| Titanium, per cent. | | 0 095 | | 0 074 | | 0 092 | |
| Titanium oxide, per cent. | 6.48 | | 5.06 | | 5.16 | | 5.46 |
| Sulfur, per cent. | 2.29 | 0 122 | 2.18 | 0 085 | 1.74 | 0.195 | 2.53 |

| | Sample No. 1 Cast Iron | Sample No 2 Slag FEB | Sample No 3 Salamander | Sample No 4 Iron Borings from No. 2 | Sample No. 5 Slag from No 2 | Sample No. 6 Iron Borings from No. 4 | Sample No. 7 Slag from No. 4 | Sample No. 8 2A Iron |
|---|---|---|---|---|---|---|---|---|
| Titanium, per cent. | 0.302 | | 25.29 | 0.170 | | 0.323 | | 0.236 |
| Titanium oxide, per cent. | | 5.77 | | | 6.58 | | 7.69 | |
| Sulfur, per cent | 0 200 | 1.39 | 6 53 | 0 058 | 2.13 | 0 138 | 2.43 | 0.076 |

*Experiment* 2.—In the second experiment, 40 gm. of salamander were placed on the bottom of the crucible and covered with 10 gm. of sodium carbonate and then with 250 gm. of cinder, mixed with 500 gm. of iron borings. The iron was in a solid button on the bottom of the crucible; there were no iron shot through the slag, and a small portion of the salamander was found attached to the bottom and sides of the iron button. The slag was crystalline and had a high $TiO_2$ appearance. Crystals of titanium carbide and nitride could not be determined in the slag next above the iron, or through it. The titanium in the iron was reduced to 0.17 per cent., and the sulfur to 0.058 per cent.; the $TiO_2$ in the slag increased to 6.58 per cent. and the sulfur to 2.13 per cent. If the titanium removed from the iron was oxidized and none of the titanium carbide or nitride of the salamander was oxidized, the $TiO_2$ in the slag should have been 5.57 per cent., indicating an oxidation of 16.76 per cent. of the titanium contained in the salamander. The sulfur of the slag, if it contained all the sulfur originally in the salamander and that removed from the iron, would have been 2.43 per cent. The greater part, but not all of it, was, therefore, liquidated and absorbed.

*Experiment* 4.—In this experiment, 40 gm. of salamander were placed on the bottom of the crucible and covered with 20 gm. of magnetic-iron ore containing 90 per cent. of magnetic oxide, and over this was placed 500 gm. of iron borings mixed with 250 gm. of slag. The salamander all left the bottom of the crucible. Two small particles, the size of a large pinhead, were noted on the side of the iron button. There were some iron shot through the slag immediately over the button and the slag was crystalline, darker than No. 2, but the color did not indicate the presence of iron oxide. An examination with a microscope showed what were thought to be crystals of titanium carbide and nitride evenly distributed through it. The titanium in the button was 0.323 per cent., the sulfur 0.138 per cent.; in the slag, the $TiO_2$ increased to 7.69 per cent. and the sulfur to 2.43 per cent. The expected $TiO_2$ in the slag was 5.24 per cent. if there was none oxidized from the salamander; the sulfur, if it contained all that in the salamander, 2.31 per cent., indicated an oxidation of 28.40 per cent. of the titanium content of the salamander, and a complete absorption of the sulfur.

*Experiment* 2-*A*.—The 40 gm. of salamander placed on the bottom of the crucible were covered with 20 gm. of pyrolusite of unknown composition, then with 500 gm. of iron borings mixed with 250 gm. of cinder. The result was that the salamander floated from the bottom of the crucible; slightly more of it than in experiment 4 was found attached to the side of the iron button and there were no iron shot. The slag had a decided manganese color. Examination with a microscope showed what were probably titanium carbide and nitride crystals in the slag next above the iron, and possibly some diffused throughout it. In the iron there was 0.076 per

cent. of sulfur; in the slag, 2.29 per cent. There was 0.236 per cent. of titanium in the iron and 6.48 per cent. of $TiO_2$ in the slag. If all the sulfur entered the slag, 2.29 per cent. was expected; if no titanium was oxidized from the salamander, 5.18 per cent. $TiO_2$ was expected, indicating complete absorption of the sulfur by the slag, and the oxidation of 22.03 per cent. of the titanium content of the salamander.

*Experiment 4-A.*—The 40 gm. of salamander were placed on the bottom of the crucible and covered with 20 gm. of fluorspar, and then with 500 gm. of iron borings mixed with 250 gm. of cinder. As a result, the salamander remained on the bottom of the crucible; there were no iron shot, and the cinder was dense and crystalline. Examination with a microscope did not indicate the presence of titanium carbide and nitride crystals in the cinder, on top of the iron, or through it. The iron button was so hard on the bottom next to the layer of salamander that it was difficult to drill, but the upper portion was soft. There was 0.095 per cent. titanium in the iron and 0.122 per cent. sulfur. The $TiO_2$ in the slag showed 5.06 per cent., the sulfur 2.18 per cent. The expected sulfur, if all was dissolved in the slag, was 2.98 per cent.; the expected $TiO_2$, 5 23 per cent., thus indicating practically complete absorption of the sulfur, but no oxidation of the titanium content of the salamander.

*Experiment* 5.—The 40 gm. of salamander were extracted with hydrochloric acid 1.12 specific gravity. Hydrofluoric acid was added to dissolve the $SiO_2$, which had gelatinized, filtered, and washed. This removed all of the sulfur and iron and almost all of the slag, leaving the titanium carbide and nitride, as shown by Mr. Shimer's analysis. The 50 gm. of iron borings placed on the bottom of the crucible were covered by the extracted salamander, then with 450 gm. of iron mixed with 250 gm. of slag. The result showed no salamander material on the bottom or sides of the iron button and no iron shot. Examination of the cinder with a microscope indicated the probable presence of titanium carbide and nitride in the lower portion of the slag. The cinder contained 5.16 per cent. $TiO_2$ and 1.74 per cent. sulfur. The expected $TiO_2$, if the titanium removed from the iron was oxidized, was 6.45 per cent.; the expected sulfur was 1.60 per cent. The $TiO_2$ content of the slag was less than that originally contained in the cinder used, indicating either a reduction of titanium and its solution in the cinder or an error in determination. The sulfur reported in the slag was slightly in excess of the total sulfur charged.

*Experiment* 6.—Over 250 gm. of borings placed on the bottom of the crucible there were poured 200 gm. of salamander, which were covered with 250 gm. of iron borings, mixed with 250 gm. of slag. This time the iron settled to the bottom, except for rather more iron shot than were found in any of the other melts. There was no salamander on the bottom or sides of the iron button, the salamander material being found next

above the iron. This looked to be slightly diluted with slag. The titanium carbide and nitride crystals gradually decreased toward the top, when they seemed to be entirely absent. The titanium in the iron was reduced to 0.092 per cent.; the sulfur was 0.195 per cent. The slag contained 4.46 per cent. $TiO_2$ and 2.53 per cent. sulfur. The expected $TiO_2$ content of the slag, if that removed from the iron was oxidized, was 4.74 per cent. The sulfur content, if it contained all the sulfur of the salamander, was 4.84 per cent., indicating an oxidation of 2 82 per cent. of the titanium content of the salamander, and a removal of about one-third of its sulfur content.

The estimates of titanium oxidized and sulfur fluxed or liquidated were based on the assumption that all the iron charged, plus 20 per cent. of the weight of the salamander, the iron in the magnetic ore used, and half the manganese charged were found in the iron buttons. The slag was assumed to contain the cinder charged, 41 per cent. of the weight of the salamander, one-half of the manganese as MnO, and all the slag-making material of the fluxes. The slag samples were in all cases taken from the top of the layer of slag, when they should have been free from titanium carbide and nitride if these materials are not dissolved in or absorbed by blast-furnace slag. The iron content of the salamander was determined by analyses; the slag content partly by analyses and partly estimated; to the weight found as above, was added the increased $TiO_2$ and sulfur contents.

Although the investigation reported above was rather crude, I think I am safe in assuming that the following conclusions are reasonably safe deductions.

1. The salamander-forming material floated on the iron bath in the furnace hearth, forming a layer between it and the cinder, and was not a salamander until the furnace was blown out. This is demonstrated by its floating on the iron in the runner and by its rising to the top of the iron in five out of six of the experiments.

2. The freeing of the salamander from sulfur does not cause the absorption of titanium by the iron. (I had rather expected the freeing of the material from sulfur would cause the titanium compounds to settle to the iron layer and be absorbed by it.) The titanium of the iron was reduced in all, except one, of the experiments and there was clearly no absorption of it by the iron.

3. The removal of the sulfur from the salamander causes the titanium compounds to be absorbed by or diffused through the cinder lying above it. This conclusion is by no means proved, it is based on the disappearance of the greater part of the titanium compounds and the presence of titanium carbide and nitride crystals through the slag.

To my knowledge, the presence of titanium carbide or nitride in blast-furnace slag has never been reported, but there are several facts which

make its presence probable. First, the analyses of slags containing $TiO_2$, when the titanium content is figured to be present as $TiO_2$, generally foot in excess of 100 per cent. Second, these slags are never entirely soluble in hydrochloric acid or mixtures of hydrochloric and sulfuric acids, but are readily soluble in a mixture of hydrochloric, sulfuric, and nitric acids. Third, treatment of the residue from solution with hydrochloric and sulfuric acids with nitric acid, freed from nitrous oxides, results in the production of brown fumes of nitrous oxides. The direct determination of titanium in the presence of $TiO_2$ is difficult, if not impossible. A determination, however, of combined carbon and nitrogen in the slag portion insoluble in hydrochloric acid would demonstrate the presence or absence of the carbide and nitride, and, without doubt, the correct estimation of the carbide. The titanium combined as nitride could only be estimated after determining which of the numerous nitrides were present. The analyses on which these conclusions are based were made on the assumption that all the $TiO_2$ present in the slags was soluble in mixtures of hydrochloric and sulfuric acids, or if not all soluble, the amounts soluble were relatively the same in all of the analyses reported.

4. The titanium compounds, when present in the hearth of a furnace, can be easily removed by introducing materials that will liquidate the calcium sulfide and oxidize the titanium compounds; this is demonstrated by the results of experiments 2, 4, and 2-A. The use of iron oxide, either fine ore or scale, fed into the tuyeres while casting or immediately after casting, offers a convenient means to the end desired. At first thought, the introduction of iron oxide through the tuyeres might seem objectionable, owing to its reducing hearth temperature, but where it is reduced by the oxidation of titanium carbide, there is a gain instead of a loss of heat. In experiment 4, the heat produced by the titanium carbide oxidized was one and three-fourths times the heat necessary for the reduction of all the iron in the oxide used and the fusion of the reduced iron. It is probable that this reaction explains the observed excessive hearth temperature, while we were using titaniferous ores, owing to unreduced iron oxide which was carried into the hearth being reduced by titanium instead of by solid carbon. The excess hearth temperature also explains the lower sulfur content of iron made while using titaniferous ore. The cause of the presence of the excess of calcium sulfide and the titanium carbide and nitride remains unsolved. It has occurred to me that excess calcium sulfide may be as much a cause of all dirty-hearth phenomena as is fine fuel, to which I have heretofore attributed it, and that, therefore, it may not be as uncommon as it at first appears. All experienced furnace managers have gone through periods of persistent dirty hearths. The presence of the titanium compounds was due either to separation of them from the cinder in which they were dissolved and their settling through it, owing

to their being of higher specific gravity, or more probably to their exuding from iron which was supersaturated with them.

Since the above notes were made, Mr. Porter W. Shimer has separated both titanium carbide and titanium nitride from a sample of charcoal blast-furnace cinder made about 1854 in a furnace operated on an ore mixture containing approximately 18 per cent. of $TiO_2$. Analyses of this cinder show it to contain from 23 to 25 per cent. of $TiO_2$, all of the titanium contents being figured as $TiO_2$. In a sample of coke-furnace cinder containing 9 per cent. of $TiO_2$, when all of the titanium contents was figured to $TiO_2$, he has separated titanium carbide but was unable to find any titanium nitride.

# Ore Deposits of the Mogollon District

BY DAVID B. SCOTT,* MOGOLLON, N. MEX.

(New York Meeting, February, 1920)

THE Mogollon mining district, New Mexico, has received little public attention, although for 15 years it has been the leading silver producer of the state; it is situated in a region remote from the principal lines of travel, and its activities have usually been limited to the operations of two or three companies. The value of the total production of the district in the 44 years of its history has been estimated at $15,000,000. For the years 1904 to 1917, inclusive, the output was: gold, $4,370,000; silver, 10,042,000 oz.; copper, 874,862 lb.; total value, approximately $10,500,000. In 1913 the district contributed 80 per cent. of the total silver production of New Mexico, but in 1917 only 41 per cent., due to serious interruption in the operations of the leading producer. Since 1914 the silver output has diminished one-half, and the present high value of silver is nearly offset by the increase in cost of production. The output of gold has rarely exceeded 35 per cent. of the total for New Mexico. The ratio of silver to gold, by weight, has averaged 46:1 based on the whole output of the district.

## Location and Topography

The town of Mogollon is 75 mi. (121 km.) by a poorly located highway from the nearest railroad terminals, at Silver City and Tyrone. All supplies are transported by motor trucks and teams.

The area, sometimes called the Cooney Mining District, lies on the western flank of the Mogollon Mountains, which occupy the western part of Socorro County. The elevations drop steeply on the west to the valley of the San Francisco River, adjacent to the Arizona boundary. The topography is very rugged, the whole area being sharply dissected; deep canyons with steep gradients and rugged walls are characteristic of the whole western slope of the Mogollon Range. The main canyons run east and west, and usually maintain a flow of water during the entire year. As would be expected in a region of such deep dissection, the mines are generally dry. The mountain ridges between the canyons have altitudes varying from 7000 to 10,000 ft. (2130 to 3050 m.) and drop sharply, on their western face, to the floor of the San Francisco Valley at an elevation of 5000 ft. (1525 m.). Erosion has been rapid, and the

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western face of the range is distinguished by broad alluvial deposits sloping to the San Francisco River.

The mountains are profusely timbered with Douglas fir, yellow pine, and spruce, the timber line being at approximately 7500 ft. (2300 m.)

The water resources in the neighborhood of the mineral area have been developed in a sporadic way; Mineral and Whitewater Creeks

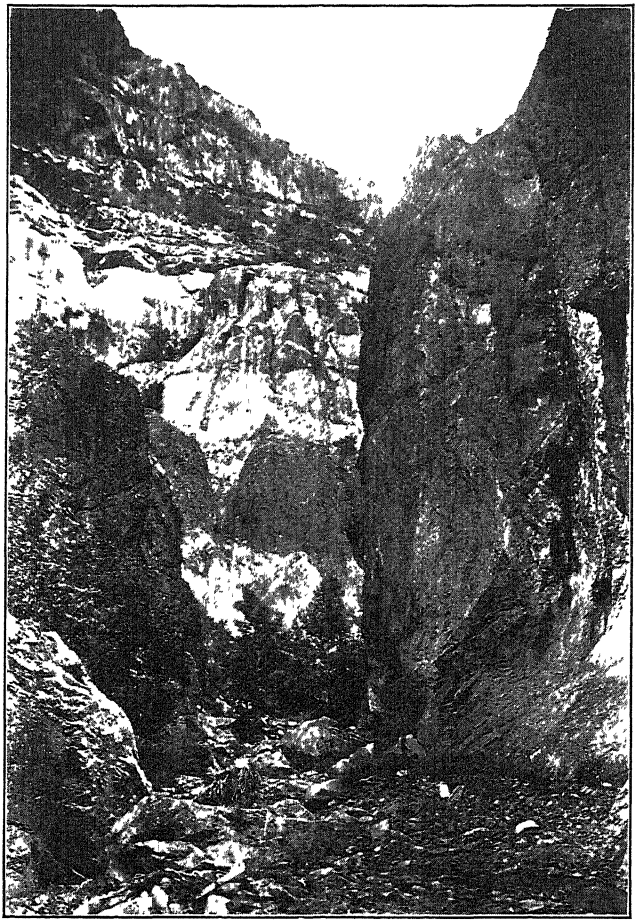

FIG. 1—RHYOLITE EXPOSURES ON MINERAL CREEK, SHOWING HORIZONTAL BEDDING IN A 1000-FT. SECTION.

have both furnished water power at various times, but the extreme annual variation and the relatively small volume of the streams both necessitate large capital outlays for power development. The more distant Gila and San Francisco Rivers offer better possibilities, but their development is restricted by local factors.

Beginning at Copper Creek, 5 mi. north of Mogollon, the district

has been progressively developed to the southward, the section centering on Mineral Creek having been operated before the discovery of the present active area around Silver Creek The mineralized zone extends from Copper Creek to Whitewater Creek, being 3½ mi. (5.6 km.) long and 5½ mi. (8.9 km.) wide. The productive area is limited on the east

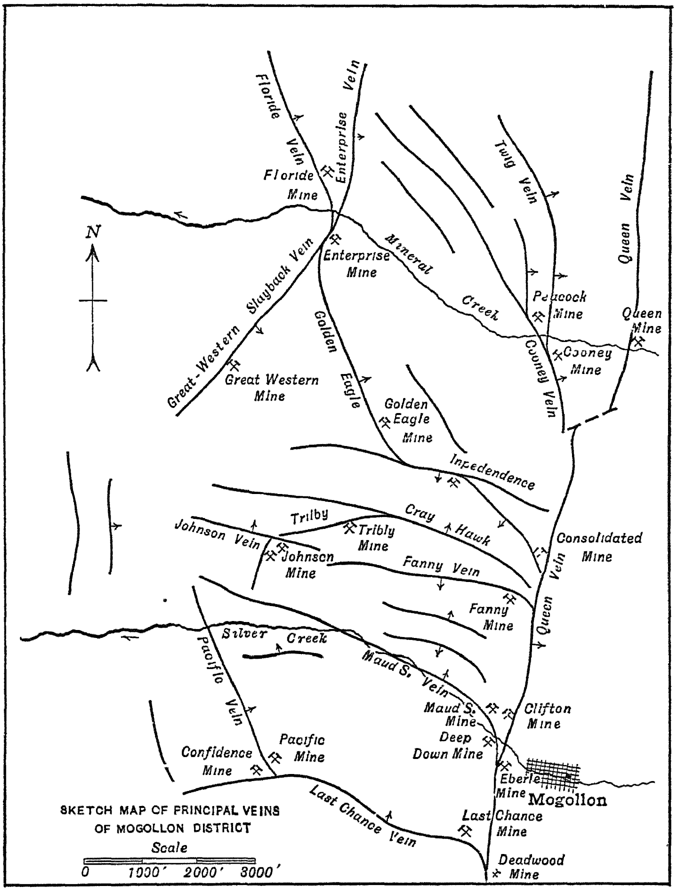


Fig. 2.

by a prominent north-south vein known as the Queen; on the west the limitation occurs through diffusion of the veins, although several prominent, but not extensive, veins lie along the western boundary.

## Historical Outline

In 1875, Sergt. J. Cooney noted the presence of copper-silver ore in the Mogollon mountains, and in the following year he and six others located and operated claims on Mineral Creek. In April, 1880, Victorio's

band of Apaches raided the settlement and Cooney was killed. The village of Claremont was established on Copper Creek soon after Sergt. Cooney's discovery, and the town of Cooney on Mineral Creek soon became the center of mining activity. The early production was confined to the shipment of rich copper ore which was found near the surface on Mineral Creek. Prospecting resulted in the discovery of the silver veins to the south, and in the establishment of the town of Mogollon on Silver Creek. These later discoveries led to the development of the most productive area of the district, and practically no attention has been paid to the older sections for 10 years. Improvement of metallurgical processes stimulated the development of the district, and after the introduction of the cyanide process at the Silver Creek mills, mining activity was at its height from 1905 to 1912. During the last few years, operations have been limited practically to the Socorro Mining and Milling Co., and the Mogollon Mines Co., although some work has been conducted by development companies, of which the Oaks Company and the Alberta Development Co. have been the most important.

## Geology of the District

*General Geology.*[1]—Western New Mexico was undergoing marine sedimentation until the close of the Cretaceous period, when a great uplift took place; that there was long era of erosion is proved by the broad Eocene sediments in the northwestern part of the state. In the late Tertiary, apparently great adjustments occurred in the earth's surface, accompanied by violent volcanic activity; this produced great extrusions of andesite, and then discharges of more acidic rhyolites and ashes. In the Mogollon Mountains, the adjustments following this period of volcanic activity resulted in this fissuring being generally east-west and north-south. The faulting was apparently eastward, whereby the older rocks are now exposed on the present surface toward the west; the throw of the faults was not very great, however, and the older rocks are at no great depth in the eastern part of the district.

The volcanic flows which constitute the backbone of the Mogollon mountains are probably underlain with Paleozoic granites, gneisses and schists, judging from evidence in the Black Range, to the southeast. The thickness of the extrusions cannot be approximated; a vertical range of 5000 ft. is evident in the area, and it is reasonable to suppose that the series will be found at least 1000 ft. below the present deepest mine workings. The center of eruption is difficult to determine, but the facts that

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[1] The geology and ore deposits, and the mining developments of the Mogollon district up to 1905 are described in "The Ore Deposits of New Mexico," U S. Geol. Survey *Prof. Paper* 68 (1910) 18, 42–47, 68–71, 191–204. See also U S. Geol. Survey *Bull.* 285 (1906), 118.

the later rhyolitic rocks are more abundant, and that the only known mineralized area is in the western section of the Mogollon range, make it probable that the center was not far from the present mining district. No other workable ore deposits are known within 30 mi. to the west, north, or east of the area under discussion.

*Stratigraphy.*—The rocks of the district are exclusively igneous, and all are varieties of andesitic and rhyolitic flows, lying in nearly horizontal beds. The andesite grades into a basaltic rock, and there are various exposures of basalt dikes. A formation resembling sandstone is found in a few localities, very limited in extent, but microscopic examination reveals that it is composed of andesite grains with a siliceous cement; this rock is directly related to the andesite near which it is found, and represents merely a recent erosion phase of the andesites.

*Rock Types.*—For purposes of identification, the various flows may be classified under 13 distinct types, most of the varieties representing different phases in the history of both effusive and intrusive rocks. They are listed as follows, in the order of their apparent age

1. Diabasic or ophitic andesite (oldest flow).
2. Lower andesite.
3. Rhyolite; spherulitic type.
4. Andesite; extrusive.
5. Fragmental rhyolite.
6. Andesite-"upper," olivine.
7. Volcanic ash; consolidated as rhyolite
8. Eastern andesite.
9. Andesite agglomerates.
10. Tuff; andesitic sandstone.
11. Tuff; rhyolitic.
12. Basalt dikes.
13. Glasses.

*Andesites.*—The andesitic rocks, of which there are three distinct and easily recognizable varieties, have been variously identified as latites, trachytes, olivine basalts, and andesites. All are andesitic in appearance, but they can be distinguished by chemical analysis and microscopic examination. The high potash contents, over 3 per cent. $K_2O$, as determined by W. T. Schaller,[2] has led some to classify these rocks broadly as latites. Certain varieties, notably No. 4 and 6, when studied under the microscope, give every evidence of being an olivine basalt.

The andesite rocks are either chocolate brown or dark olive green in color; they may be fine-grained, or so diabasic in structure as to resemble a diabase; or in certain localities so amygdaloidal as to look like a miniature pudding-stone. Alteration of the wall rocks of certain veins has con-

[2] U. S. Geol. Survey *Prof. Paper* 68, 193.

verted the andesite in some places into seams of pale-green gouge. At the lowest horizon reached in mining, a distinct variety has been recognized, No. 1, or diabasic andesite; this has well developed feldspar phenocrysts, soda feldspars, probably in excess, and a rich augite or hornblende groundmass. It is diabasic in appearance, except that the feldspars are more feathery and less regular in shape than in typical diabase. It is possibly the interior phase of a great extrusive sill, the outer parts of which are the olivine basalt and amygdaloidal andesites; there is reason to believe that this rock is common to the whole underlying basement of the district. The change from the greenish-brown andesite to this diabasic andesite is very marked in the lowest part of the Fanny mine, the transition being especially noticeable in the main shaft at a depth of 1145 ft. (349 m.). Although exposures of its contact with the overlying andesite are infrequent at present, it does not appear that the upper is intrusive into the lower andesite.

*Rhyolites.*—By comparison with rock types in eastern Socorro County, and by examination of the deeper rocks of this district, it seems highly probable that the rhyolites are younger than the andesites in this district. Although the flows of andesite and rhyolite alternate in the upper exposures, the more basic andesite generally precedes the acid rhyolites. The rhyolites have three distinct characters, resulting from their different origins—spherulitic, fragmental, and hard siliceous; all give evidence of a period of violent volcanic ejectment, fragments, breccia, and ash being their common constituents. Only one variety of rhyolite exhibits the marked flow structure that is usual in this rock; this is a very acid surface flow, sometimes called the "Fanny rhyolite," which is spherulitic and rather vesicular. It has every appearance of a surface flow.

Following the andesite called No. 4, the rhyolites show marked fragmental and ashy structure, this suggesting that prior to this period the extrusions of both rhyolite and andesite occurred as gentle surface flows. Some exposures show a bedded rhyolitic ash, which is useful locally as a building stone. Included in this later series, but of small thickness, are several beds of an andesitic tuff, greenish-gray in color, with abundant fragmental material. In the southern part of the area, and also in some of the deeper mine workings, a rhyolite occurs which is difficult to classify. It resembles a cemented ash, highly siliceous and very hard; it is said to reveal a spherulitic structure under the microscope, and this fact, together with its location in one of mines, suggests that it is an intrusive rock. It is the hardest rock encountered in the district, and is locally designated as "Pacific" rhyolite.

*Basalt Dikes.*—The relatively unimportant exposures of basalt indicate that the rock is of the intrusive-sill type, with occasional variation to the dike form. The principal exposures are on the northern wall of

Cooney canyon; there are a few dike exposures in Silver Creek and Whitewater canyons. The basalt is generally gray or black, and very fine-grained. It evidently belongs to a later period of activity, and, where exposed as sills, is well above the "upper" andesite.

*Glasses.*—A glassy rock, probably a phase of the gentler rhyolitic extrusions, is found in a limited area in the extreme southern part of the area. It is black, and some varieties approach true obsidian. The best exposures are found in Whitewater canyon, above its junction with the south fork of the creek.

## Ore Deposits

*General Features.*—The ore deposits may be classified as Tertiary mineralizations in rhyolite and andesite. The veins are all of the fault-fissure type, and the predominant filling is quartz and calcite.

Practically all of the profitable mining has been limited to those veins which have a general east-west strike. A few of the minor north-south veins have been exploited to some extent, and on one of them a small mine, the Pacific, was developed. The two most interesting features of these veins are the mineralization of wall rocks, and the indefinite distribution of ore deposits. The fault fissures originally contained considerable brecciated material, and this has been largely replaced by the mineral-bearing vein filling. The period of fissuring was evidently closely followed by the appearance of mineralizing solutions. The veins are often of the open type, containing cavities and druses, and the valuable minerals occur in a finely divided form. The dip of the veins is rarely less than 65°, the principal veins being all close to 70°; those having a dip to the north predominate over those inclining to the south.

To the east and to the north of the Mogollon district the country rocks exhibit little variation in character over a large extent of territory, but there is a complete absence of mineralization and of vein systems. The Queen vein apparently limits the mineralized area on the east.

The vertical range of exploration thus far attained is approximately 1700 ft. (518 m.), but there is reason to believe that mineralization extends to depths below the present workings. The extensive faulting and the persistence and strength of many of the veins are promising indications of a downward extension of the deposits. The great depth to which oxidation has penetrated, 1200 ft. (366 m.) in the Fanny mine, indicates the open character of the deposits.

Vein outcrops are no indications of ore deposits in this locality. Many of the richest orebodies, notably in the Last Chance mine of the Mogollon Mines Co., and in the Fanny mine of the Socorro Mining and Milling Co., did not reach the outcrop. Conversely, some promising surface deposits on the strong Enterprise or King vein, in the western

part, gave evidence of failing in depth. The highly siliceous character of some of the veins, notably in the Mineral Creek area, has preserved their outline, and they project above the enclosing wall rocks on the surface; this is particularly true of the Queen vein. Many of the profitable veins, however, are oxidized to such an extent that the erosion of their outcrops easily keeps pace with the disintegration of the wall rocks. Intersections of veins are common, but their influence on ore deposition depends entirely on the strength of the mineralizing solutions; some of the intersections show notable mineralization, while others are peculiarly barren.

The veins, almost without exception, contain copper, but usually only a trace; at several localities, notably in the Cooney mine, copper becomes more important. Traces of copper are found at every horizon, and in every variety of vein filling. The occurrence of the commercial copper deposits is not essentially different from that of the silver-gold deposits, but copper deposits are not of large horizontal or vertical extent anywhere in the district.

*Vein Characters.*—The vein filling is fairly uniform through the entire district, being composed of quartz, calcite, and fluorite, in order of relative abundance. The quartz varies through crystalline, massive, and hackly varieties, and in some localities the vein filling is entirely of chalcedony. Some veins are so dense that oxidation has been unable to penetrate. This is a feature of the ore-bearing zones in the Queen vein; here the sulfide mineral, probably entirely argentite, is exposed at the outcrop and where disclosed underground is found in irregular banding. The best ore in the Queen vein is invariably associated, in the upper horizons, with this hard cherty texture. This has been turned to profitable account by using the rock for tube-mill pebbles in the Socorro company's mill.

The inclusion of wall rock in the vein filling, and the replacement of these inclusions by the mineral solutions are pronounced in the Fanny and Last Chance veins. Where these inclusions are numerous, the ore is often of good grade, as the sulfide particles appear to surround the fragments in a selective manner; this is particularly true of andesite fragments. In other sections of the same vein, the filling may be composed entirely of white quartz and calcite of moderate hardness.

The fissures which contain the veins are usually distinct and clean, and of fairly uniform width, tending to indicate that they originated from deep forces and are therefore not shallow. Zonal shattering, which is more characteristic of shallow fissuring forces, is seldom observed close to the veins of this district. The Fanny, Last Chance, and Queen veins, especially, show a marked regularity in dip and persistence to a depth of at least 1200 ft. (366 meters).

Several of the veins, among them the Fanny, show a branching and

dissipation at their western extremities. The layers of andesite and rhyolite, in this particular zone, are composed largely of tuffs and breccias; the predominant mine rock in the western end of the Fanny mine is rhyolite, mostly of the fragmental variety. These tuffs and breccias, unless greatly altered, would not be favorable for the forming of strong definite fissures which would remain open during the process of vein filling. While a strong fissure might be indicated at the surface in the silicified rocks, exploration at lower horizons in the series of tuffs and breccias might discover a series of feathered and branching veins; this is the case in at least one prominent instance.

*Alteration and Oxidation of Wall Rocks.*—Both andesite and rhyolite close to the veins show many evidences of alteration and secondary mineralization. The texture of these rocks was evidently favorable to penetration by thermal waters, and the effects noted are those due to alteration, silicification and pyritization. Andesites, notably the dark brown varieties, show propylitic alteration. The rhyolites give abundant evidence of silicifying processes, and these evidences are very noticeable in the ashy varieties at considerable distances from the veins. Many small crevices are filled with quartz, and in a long hanging-wall crosscut in the Fanny mine, stringers in the silicified rhyolite carry silver values more than 300 ft. (91 m.) from the vein. The lower andesite, as revealed on lower levels of the principal mines, shows evidence of great chemical action at distances of 100 ft. and more from the veins. Mineralization of the wall rocks to a profitable extent is found in several instances in both the silver and the copper deposits.

Oxidation of the vein materials themselves is also very noticeable at depth. In certain localities, mentioned later, leaching has occurred many hundred feet below the outcrops, with possible reconcentration. The surface oxidation in these veins does not reveal much of interest. There is no great variation in the ratio between gold and silver near the surface and deeper down. Throughout the mines, water courses and accompanying mud seams are found, sometimes at depths as great as 1400 ft. (427 m.) below outcrop. In some districts, silicification of wall rocks has been found favorable to ore deposition; while this is more or less true in the Mogollon deposits, it cannot be considered as a sure ore indicator.

*Silver-gold Veins.*—Both silver and gold are found in all the veins in the district. The principal silver-bearing veins are the Last Chance, Maud S., Little Fanny, Independence, Johnson, Cooney, and Floride. Between these veins, occasionally having a strike nearly at right angles to them, are others of less importance, notably the Pacific and the Comet. At present, mining is confined to the east-west veins which, with one exception, dip to the north. Most of the east-west veins show a marked curvature to the south near their junction with the Queen vein on the

east. The Last Chance parallels the Queen vein, near the junction, and the Deadwood mine is located upon this southerly extension.

*Copper Veins.*—The Cooney vein is the only prominent example of copper mineralization yet developed in the district. This copper deposit is localized near the junction with the Twig vein, and is a peculiar deposit of almost certainly primary chalcocite, bornite, and chalcopyrite, between andesite walls. The other occurrences of copper occur chiefly between andesite walls, with extensive copper mineralization of the rocks. Copper deposits in andesite may usually be viewed with suspicion, as they are notably pockety and uncertain; the Cooney deposit yielded a large tonnage although it had small lateral extent, and it was rich in silver and gold, a feature which is noticed in the other copper deposits of the district.

*Mineralogy of the Veins.*—The principal silver mineral is argentite, which is usually finely divided. Native silver, cerargyrite, bromyrite, and stromeyerite have also been noted. The stromeyerite may possibly be classed as an argentiferous chalcocite, excepting that its copper content, by actual analysis, does not much exceed 30 per cent.; this mineral has been found rather abundantly in the Johnson and Cooney mines. Native gold has been found, the most striking occurrence having been in small andesite deposits; the gold in the principal veins is usually invisible and commonly associated with the iron minerals. Fluorite, calcite, and quartz are found in crystals; the fluorite, which is present in all of the east-west veins, is green and purple in color. Hackly quartz in very large and showy specimens was found in the old Cooney mine.

The copper minerals have been determined as chalcocite, bornite, chalcopyrite, malachite, azurite, and occasional cuprite. Some occurrences of a mineral resembling tetrahedrite have been seen. Pyrite, both cupriferous and normal, is present in considerable quantities, especially in the andesite wall rocks; it is also found in the veins.

*Secondary Enrichment.*—Evidences of secondary enrichment are few, and it is probable that there has been more migration than enrichment. The only indications of secondary enrichment are the prevalence of iron minerals in the veins and the apparent migration of values. It is known that silver can be precipitated from solution by pyrite and other sulfides. In the occurrences of redeposited values so far observed in these mines, however, there is little to support the idea that the primary minerals have been enriched or that the minerals usually accessory to enrichment are present. The gold may be associated with the silver minerals, or it may be entirely free; it has doubtless been subjected to some secondary action. Ferric iron is capable of dissolving gold, but large areas of the veins in the developed mines have quite evidently not been subjected to any leaching or secondary action.

*Values and Ratios.*—Various orebodies have produced many thousand

tons of ore running well above 30 oz. (933 gm.) of silver per ton. The values do not show any regular gradation from low-grade to high-grade ore. The vein material is either very low-grade, having a silver content of 2 to 3 oz., or it runs above 7 or 8 oz. (235 gm.) per ton; there are no large deposits of ore which average just under the limit of profitable mining. The upper limits are variable; considerable high-grade ore has recently been mined in the Last Chance and Fanny mines, running from $20 to $40 per ton. In the earlier days, the extensive deposits of ore found in the upper 900 ft. (274 m.) of the mines, carried from 30 to 40 oz. (avg. 950 gm.) of silver per ton.

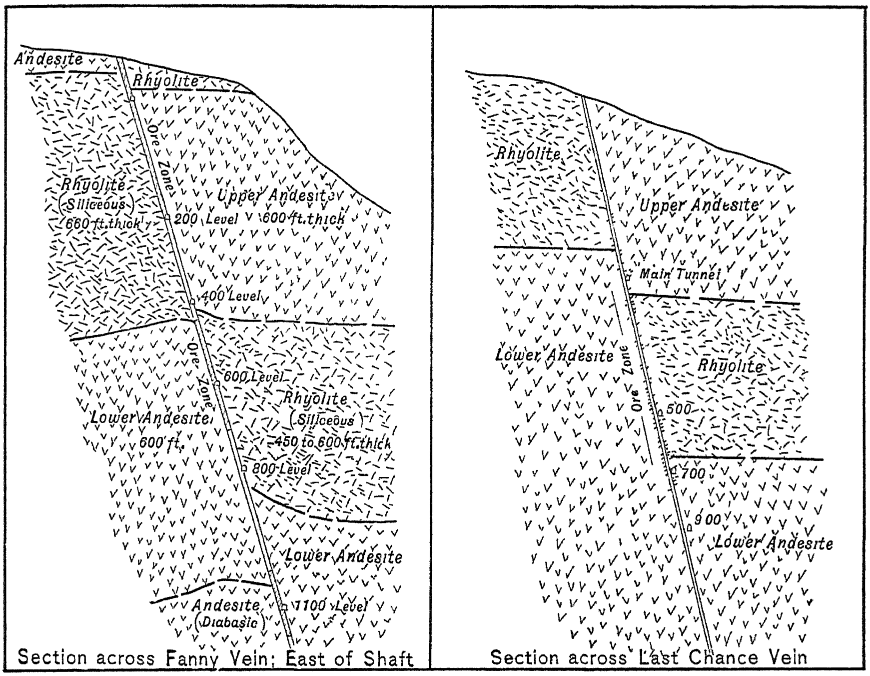


FIG. 3.

The veins do not afford positive proof of much impoverishment in depth. The ores in the lower levels are often very low-grade, but in the deepest level of the Fanny mine is found what appears to be the top of an orebody carrying values which compare favorably with any found 900 ft. above this horizon. This district has not been developed to sufficient depth to show positive evidence of general impoverishment.

The ratio of silver to gold, by *weight*, ranges from 33:1 to 84:1; the average may be regarded as 50:1. When the silver content is above 10 oz. (310 gm.) the ratio usually becomes larger, and in the highest-grade ores it may reach 150:1. The doubling in the market price of

silver within recent years has therefore meant an increase of 60 per cent. in the average value of the ores of this district.

*Extent and Size of Orebodies.*—The zones of profitable mineralization are rather well defined; they are often as long as they are deep; they are irregularly distributed throughout the veins; and they bear no definite relation to one another. In fact, the ore shoots are extremely puzzling in outline, and their extensions vertically or laterally can never be predicted with certainty. Large ones have been found, one in the Last Chance mine having produced 145,000 tons of ore. Many of them have been from 300 to 400 ft. (91 to 122 m.) in length, and some of these long shoots have bottomed nearly horizontally along their entire length.

The orebodies in both of the principal mines are notable in one particular; they are situated at a horizon which contains a change and a reversal of the wall rocks. That is, they are found in greatest development where the rhyolite is succeeded by andesite on the foot wall, and where the hanging-wall andesite is underlain with rhyolite. Above and below this change in wall rocks, and at no great distance from it, have been found the best orebodies of the district. This change in opposing wall rocks is due, of course, to faulting, the throw of the principal faults being from 450 ft. (137 m.) to nearly 700 ft. (213 m.). The principal deposits have been found under two conditions: with the upper andesite as hanging wall and rhyolite as foot wall; or with the lower andesite for foot wall and a rhyolite hanging wall. The latter combination has produced almost 60 per cent. of the stoping area of the Fanny vein. Taking the Fanny mine as a further example, we find that the zone having a rhyolite foot wall and andesite hanging wall, which is the upper 500 ft., yielded ore of profitable grade from 28 per cent. of its total area; the next zone, having andesite foot wall and rhyolite hanging wall, about 450 ft. in vertical extent, contained minable ore in 47 per cent. of its area. The bottom zone, generally with andesite on both walls, contained profitable ore in 13 per cent. of its area. This example is believed to be fairly characteristic of the district.

*Ore Indicators.*—The variations in the character and the mineralization of the veins are so great that persistent development along the whole length of a vein is essential. It has been noted frequently that where numerous stringers from the vein make off into the walls, there is likely to be an orebody in close proximity. A widening of the vein is more likely than not to be accompanied by good values. The presence or absence of either fluorite or calcite has no significance; the latter mineral itself seldom contains values greater than $1 per ton. The appearance of the vein, the presence of banding or ribbon structure, such as is noted in some of the valuable deposits, and the crystallization of the gangue minerals, have little relation to the values. In exploration work, therefore, no isolated occurrence of values in the vein can afford to be over-

looked; the driving of a short raise on a promising streak in an otherwise unpromising vein has often resulted in the development of a valuable orebody.

## Mining Developments

The developed areas may be divided into two groups: the Silver Creek and the Cooney. The first group contains all of the properties now active, and includes probably 95 per cent. of all the development that has been done in the district. It is also the section from which 85 per cent. of the total gross production of the Mogollon district has been derived. In all, 13 mining companies have been in active operation in the district at one time or another. There have been 10 reduction plants, and six of these are now operating. The extraction processes have passed through pan-amalgamation, coarse sand leaching, and stamp-mill crushing, to the ball-mill grinding and slimes treatment as practised in the most recent mill.

### *Silver Creek Group*

*Mogollon Mines Co.*—The properties of this company are situated at the southerly end of the vein system. The principal mine, the Last Chance, holds the record in the district for continuity and productiveness. Production has come almost entirely from the Last Chance vein, and the developed orebodies have been found in the Last Chance and Top claims. This company is now operating the extension of its vein in the ground of the Treasure Mining and Reduction Co., and in that of the Deadwood Mining Co. The recorded production of the Mogollon Mines Co., for the last 25 years, has been about $7,500,000.

The company owns a 40-stamp cyanide mill, of 200 tons per day capacity. Some of the values are recovered by concentration, but the principal extraction is effected by leaching, and by agitation in small Puchuca tanks. Precipitation on zinc shavings is followed by reduction in the refinery to bullion averaging above 900 fine, which is shipped as the finished product. The company operates its own electric generating plant, which is equipped with De La Vergne oil engines of 500-hp. rated capacity, and has been in successful operation for eight years.

The Last Chance mine is opened to a depth of 900 ft. (274 m.) on the incline below its main tunnel level, and the vein has been developed through a total vertical range of 1700 ft. (518 m.). The main entry is by a tunnel above the upper end of the mill. A modern hoisting plant was installed three years ago. The hoist and the storage pockets are underground, and the ore is trammed to the mill ore-bins. Development below the tunnel level is principally by levels 200 ft. (61 m.) apart on the dip of the vein. Below the 900-ft. level, diamond drilling has failed to indicate any promising ore shoots.

All of the development has been confined to the Last Chance vein, with a few hundred feet of exploration on the Queen vein. Longitudinally, the development covers about 2000 ft. (610 m.) on the vein, which is nearly east-west. The east end of the mine will eventually be connected with the workings of the Deadwood mine, and present plans contemplate connection of the 500 level with the Confidence workings on the west end. These plans will make continuous all of the mine workings on the longest productive vein in the district.

The wall rocks of the Last Chance-Confidence vein are clearly defined. The foot wall is a siliceous rhyolite for 600 ft. (183 m.) on the dip

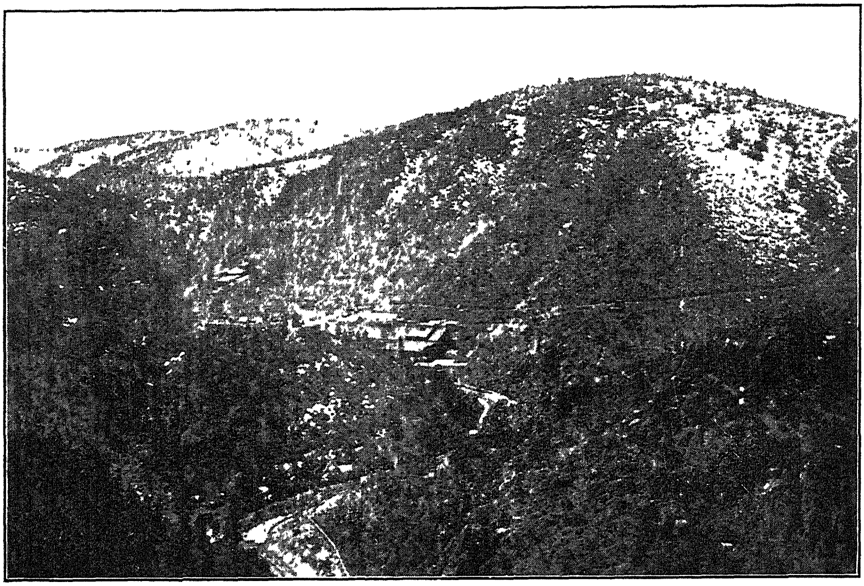

FIG. 4—LOOKING SOUTH IN THE MOGOLLON MOUNTAINS PLANT OF THE MOGOLLON MINES CO. IN THE CENTER.

below the outcrop. This is underlain by an olivine andesite, which continues to the lowest workings of the mine. The hanging wall in the upper 700 ft. of the vein is andesite, followed by a rhyolite flow 550 ft. (168 m.) thick, which is underlain by the lower andesite; andesite therefore composes both walls below the 600 level. The contact between rhyolite and andesite flows is said to dip 12° to the west. The vein is one of the strongest, and in the early days, was the richest in the district. Its dip is fairly regular, at 70° to the north. The vein width ranges from 4 to 6 ft. (avg. 1.5 m.), but some stopes have been carried to a width of 20 ft. (6 m.). It is generally regular in strike, and contains notably few splits and horses; only one split, recently encountered, has been observed. Upon its approach to the Queen vein on the east end, the vein curves

sharply to the south and parallels the Queen, the separating wall rock not exceeding 20 ft. in thickness. Along the bend of the Last Chance vein are found relief cracks in the hanging wall, radiating normally to the curvature.

Hard quartz, calcite, and andesite breccia predominate in the filling; there is also some fluorite. The fragments of wall-rock breccia are often completely replaced by the vein matter, and the andesite fragments frequently are surrounded by concentric banding of silver sulfides. This brecciated character is found in most of the best ore shoots, a feature also noted in other mines of the district. Small pockets of bornite and chalcocite are encountered occasionally, which invariably run high in gold and silver. Oxidation of the vein is marked at depths of 800 and 900 ft. (274 m.) below outcrop. In a recently opened section on the west end of the 500 level, there was strong evidence of extensive oxidation and possible migration.

The ore zone, in general, has been found to extend from 100 ft. (30 m.) above the change in foot wall rocks to about the same distance above the change in hanging-wall rocks; *i.e.*, above the change from rhyolite to andesite in each instance. The bottom limit of ore so far developed lies from 900 ft. to 1100 ft. (335 m.) below the outcrop. The lower levels are entirely in andesite, and the orebody on the east end of the 700 level has andesite walls. Development on the 900 level revealed a strong vein 6 ft. wide, within andesite walls, but without extensive mineralization. The ore shoots are generally vertical, and there have been five distinct and highly productive ones; the largest, No. 2, produced 145,000 tons of ore. All of them have occurred above the 700 level, or less than 1400 ft. (427 m.) below the outcrop. The mine is notably dry throughout its entire extent.

*Socorro Mining and Milling Co.*—The property of this company includes 66 mining claims, the principal mine workings being the Fanny, the Johnson, and the Cooney. The company has operated since 1909, and previously the various units of the property were under development as far back as 1893. The Fanny is the largest mine in the district, and has furnished nearly the entire production of this company; its total production of gold and silver, to date, has been $4,869,000.

The main plant, situated high on the north side of Silver Creek canyon, has a capacity of 250 tons per day. Crushing is done in Hardinge ball-mills and in tube-mills, followed by agitation in 45-ft. Pachuca tanks, and finally filtration and zinc precipitation. The bullion has frequently exceeded 930 fine. Power is developed by four De La Vergne semi-Diesel oil engines, having a total sea-level rating of 960 hp. The hoisting equipment includes a 5-ton electric hoist, 75-ft. steel headframe, and 1½-ton skips, which deliver to the crusher ore-bins. The crushed ore is delivered by belt conveyor to the mill ore-bins.

The Fanny vein has been the principal source of production; this belongs to the east-west series, and differs from all of the other productive veins by dipping to the south. The vein has been mined continuously for 2700 ft. (823 m.) along its strike, and the workings extend to a depth of 1400 ft. (427 m.) below the highest outcrop, but only 1200 ft. (366 m.) below the average surface. The wall rocks are essentially like those in the Last Chance mine, except that a greater variety of andesite is represented. One type of andesite, diabasic in structure, which is found in the foot wall of the 1100 level, is essentially different from any type encountered elsewhere, except in the bottom of the Cooney mine. The

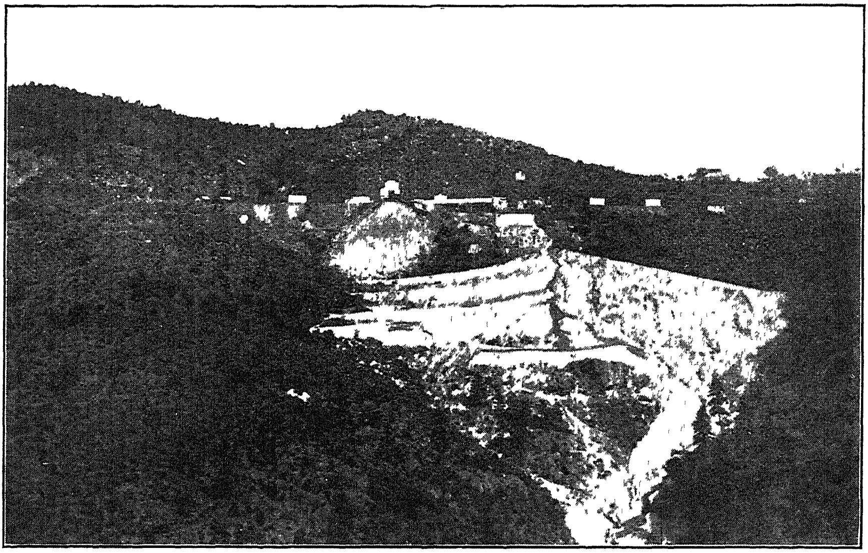

FIG. 5.—MAIN PLANT OF THE SOCORRO MINING AND MILLING CO., MOGOLLON, NEW MEXICO.

vein has been mined in places to a width of over 20 ft. (6 m.); the width decreases somewhat with depth, although on the 1100 level a 14-ft. orebody was found. There are several interesting splits which have furnished large ore deposits.

The Fanny mine is developed by a 1200-ft. vertical shaft, with levels at 100-ft. intervals, its upper 800 ft. being in the hanging wall. The lowest level, the 1200, is about 200 ft. ( 61 m.) lower than any other accessible workings in the Silver Creek group. Developments to the west have indicated a feathering of the vein, while the east end has not been developed to the Queen vein; the productive length is therefore limited to 2400 ft. (732 m.). Development includes 46,200 ft. (14,282 m.) of drifts, raises and shafts, and the total area of vein blocked out by this work is 2,800,000 sq. ft. (260.130 sq. m.) of which 33 per cent. is

considered minable. On the basis of tonnage produced, the average mined width for the whole mine is 6.5 ft. (2 m.).

The orebodies are notably irregular and suggest no origin from a common center. Nearly the whole production has come from above the 900 level. Below the 300 level, the vein shows a division into two distinct productive areas, separated by a barren zone 400 ft. in horizontal length. The eastern zone is the larger, the ore having been fairly continuous for 900 ft. (274 m.), while the western zone is less regular and only about 500 ft. (152 m.) long; the depth of both zones is about 550 ft. (168 m.) on the dip of the vein. Above the 300 level, a third productive zone showed ore for a distance of 1400 ft. (427 m.) and depth of 250 ft. (76 m.). The lower areas are separated from the upper by a horizontal barren zone nearly 200 ft. high. Indications of undiscovered ore deposits at the east end and below the 1100 level are not lacking.

The Fanny vein occupies a faulted fissure, which has a displacement of 400 ft. (122 m.) on its east and 550 ft. (168 m.) on its west end. The contacts between rhyolite and andesite are nearly horizontal; a series of small cross-faults, however, causes an apparent step series on the west end. A section across the vein at the eastern end of the mine (Fig. 3) shows the foot wall to have a thin strip of andesite at the top, followed by a rhyolite flow 660 ft. (201 m.) thick. This is underlain by the lower andesite, which, after a thickness of 600 ft., grades into an andesite having a structure resembling diabase. The hanging wall is capped by a rhyolite, underlain by 600 ft. (183 m.) of the upper andesite; this is followed by a siliceous rhyolite corresponding to the foot-wall rhyolite, and below this is the lower andesite. The zones of mineralization are found immediately above and below the changes in the wall rocks. The levels below the 900 are in andesite entirely, and good ore has been mined in this horizon.

At the Johnson mine, operated by the Socorro company, development consists of two inclined shafts, 400 and 450 ft. (137 m.) deep respectively, and 7700 ft. (2347 m.) of drifts, exploring the vein for a length of 1800 ft. (550 m.). The vein is a narrow east-west lode in andesite. A junction of veins in the west end of the mine has afforded a narrow ore shoot of high-grade silver ore, and a north-south vein, near that locality, has produced some high-grade copper-silver ore; the latter occurs in irregular deposits between andesite walls, and is not believed to be persistent.

The Socorro company also has workings on the Queen vein, although the low grade and extreme hardness of the ore make its treatment expensive. This vein, as developed in the Consolidated workings, shows a width of at least 90 ft. (27 m.), but the valuable portion is limited to 10 ft. (3 m.). The hanging-wall side of the vein is composed of barren calcite, and the foot-wall side largely of chalcedonic quartz. The values are irregularly distributed, but the vein is estimated to yield minable

ore valued at $3 to $6 per ton. Reserves of such ore are large, but unprofitable for the present.

*The Deadwood Mining Co.*—This property adjoins that of the Mogollon Mines Co., and is now under option to the latter. Operations of the Deadwood company began in 1905, and produced 44,000 tons valued at $325,000 before suspension of its operations. The mine occupies that section of the Last Chance vein which is adjacent and parallel to the Queen. It has a 50-ton cyanide plant, and a De La Vergne oil-engine power installation. Mine development amounts to 4500 ft. (1372 m.), reaching a depth of 500 ft. and length of 1550 ft. (472 m.) by a vertical shaft from which crosscuts are driven on five levels. Although the vein was unusually wide, 30 ft. (9 m.) in some places, the mining width was only 7 ft. (2 m.), and the foot-wall side always carried the best values. Only one occurrence of copper ore was found, and this was in the foot-wall andesite, away from the vein.

Three orebodies were developed, of which the largest was 350 ft. long and 300 ft. high (107 by 91 m.). The vein is on the foot-wall side of the Queen, and at a distance of 10 to 15 ft. from it. The relations of the wall rocks suggest that it is really a part of the Queen vein, but there has not been enough development to determine whether the hanging-wall component of the fault has been displaced along the Deadwood-Chance or along the Queen vein. The principal ore zone in the Deadwood mine was found below the 215 level, entirely within that section which had a rhyolite hanging wall and an andesite foot wall.

*The Confidence Mine.*—This property, now operated by the Mogollon Mines Co., has produced ore worth $1,200,000 from a limited area. The mine occupies the westward extension of the Last Chance vein, and is developed to a depth of 950 ft. (290 m.) below outcrop, and for a length of 2000 ft. (610 m.), mostly above the 450-ft. level. The principal orebody was mined to a width of 10 to 15 ft., and from its bottom to the surface.

*The Maud S. Mine.*—The gross production of this property is given as $800,000, ranking fourth in the district. The vein is a strong east-west lode north of the Last Chance property, and near the bottom of Silver Creek canyon. It dips to the north, and flattens to nearly 60° in places. Development extends 1500 ft. laterally and to a vertical depth of 625 ft. (190 m.). The main orebody, originally developed from a tunnel, was nearly 1000 ft. long and 500 ft. (152 m.) high. The highly siliceous rhyolite foot wall forms a prominent topographical feature on the north side of Silver Creek.

*The Alberta Development Co.*—This property, still only a prospect, is located on the Ida May vein, which strikes a little south of east, and lies to the north of the Fanny vein. Development consists of a 400-ft. tunnel to the vein, and about 500 ft. of drifting, which shows a rhyolite foot wall and an andesite hanging wall. The vein is hard and siliceous,

with prominent ribbon structure, and in close proximity to its junction with the Independence vein there is evidence of considerable mineralization.

*The Pacific Mine.*—The Pacific workings occupy a north-south vein which intersects the Confidence vein in the vicinity of the Confidence mine, and are credited with a production of $200,000. The vein is narrow, and has been developed, on five levels, 1200 ft. laterally and 505 ft. vertically. Entry is by a tunnel, with an inclined shaft which descends 385 ft. (117 m.) on the vein below the tunnel level. At surface both walls are andesite, but below the tunnel level the foot wall changes to rhyolite. The orebodies have been erratic, but the ore, where found, has been of sufficient value to pay for the excessive development charge which accompanies a mine of this character. Very little ore has been found below 250 ft. from the surface.

*The Oaks Co.*—This company holds several small properties for development, among them the Clifton, Eberle, and Deep Down mines; it also holds, under working agreement, the Maud S. and Pacific mines, previously described. The Clifton includes a little work on the Queen vein, which has indicated good values. The Eberle is on the Queen and also on the southward curve of the Maud S. vein. The Deep Down mine, also on the southward curving Maud S. vein, has been developed a little at depth. The group is operated through a shaft on the Deep Down claim, which is now connected with the old Maud S. workings. An interesting feature in this group is the occurrence of profitable ore in the Queen vein, as exposed in the Clifton tunnel, and by the southern workings on the Eberle claim.

*Other Properties.*—Among other groups which have been under development at different periods may be mentioned the Gold Dust, located on the most southerly vein in the district, and developed by two tunnels; the vein is reported to have shown exceptionally high gold values. The Trilby, located on the eastward extension of the Johnson vein, has been prospected with a 250-ft. (76-m.) inclined shaft and a small amount of lateral development. The west end of this vein is notable for the quantity of fluorite it contains.

### *The Cooney Group*

Developments in this area are not extensive, and only one large mine has operated successfully. The floor of Mineral Creek canyon is more than 600 ft. (183 m.) lower than the bed of Silver Creek. Mineralized veins are traceable for several miles to the north of Cooney, and a few prospects have been found on Copper Creek, which parallels Mineral Creek on the North. The veins in the vicinity of Mineral Creek are notable for their copper values, especially in the eastern portion of the group. No active mining has been carried on here since 1913.

*The Cooney Mine.*—The Cooney vein was the first source of ore in the Mogollon district, its discovery in 1875 being the first mineral location in this part of New Mexico. The vein has a northwesterly strike, and curves somewhat to the south as it approaches the Queen vein. All of the development is on the eastern end of the vein, at the crossing of Mineral Creek. The production credited to this vein is valued at $1,700,000, most of the ore being copper-silver.

The Cooney mine, with its related Peacock mine, is located at the junction of the Cooney and Twig veins. Mining was confined to the Cooney vein, extending 900 ft. laterally and to a depth of 750 ft. (229 m.) below outcrop, the bottom workings being nearly 500 ft. lower in elevation than any of the Silver Creek mines. The orebody had an extreme length of 250 ft. (76 m). but much of it was restricted to narrower limits. The copper values in the upper workings evidently did not persist at great depth; it was reported that the horizon 400 ft. (122 m.) below the tunnel level showed a preponderance of silver ore. The vein varies in width from 2 to 50 ft., the extreme widths including a considerable mineralization of the hanging wall. The junction of the Twig vein with the Cooney vein is easily seen on the surface, but it is not conspicuous in the mine workings. The main ore shoot of copper sulfides was in close proximity to this junction, and most of it was found to the north of the intersection; only unimportant amounts of copper were found elsewhere. The ore consisted chiefly of chalcocite, bornite, chalcopyrite and various oxides of copper, associated with considerable silver and gold. It is stated that some of the ore, as mined, ran as high as 45 per cent. copper, and some of the specimens preserved from this deposit show that parts of the vein were nearly pure chalcocite. There is no evidence of leaching, and little to suggest secondary enrichment.

The wall rocks in the upper workings are both andesite, but differing in texture, one of them being tuffaceous and the other fine-grained and probably approaching latite in relation of the feldspars. Specimens from the dump, derived probably from the lower levels now inaccessible, indicate that an andesite having the diabase features noted in the Fanny mine was found in this mine. The vein matter is mostly hard and siliceous, and much hackly quartz was found in the early days. Prospecting on the vein to the south revealed a probable faulting of the Cooney by the Twig vein; the Cooney vein is apparently cut off on the south, and exploration was never carried far enough to develop its extension.

Connected with the Cooney mine, on the 400 level, is the Peacock mine, lying to the northward. A little development has been done here on three levels, and irregular deposits of silver ore have been mined. The absence of copper is noteworthy, considering its close proximity to the copper mineralization in the Cooney mine. The vein is about 4 ft. in width, between andesite walls. The vein filling is quartz-calcite-fluorite,

and some very high-grade pockets of argentite have been found. The Cooney vein at this point has not yet indicated continuous ore deposits.

*Enterprise Mine.*—A vein running generally north-south cuts Mineral Creek canyon near its western end; this is one of a series of undefined north-south veins which traverse the western side of the district, and is variously known as the King, Great Western, Slayback, Floride and Enterprise. The Enterprise vein is continuous with the Slayback, which forms its southern extension. It dips to the east at about 70°, and lies chiefly within rhyolitic walls. To the south of Mineral Creek, the vein was prospected along a total length of 1100 ft. (335 m.), and over a vertical range of 450 ft. (137 m.), without disclosing any considerable values. The vein is quite uniformly mineralized with low-grade material; the values seem to exist chiefly in highly oxidized pyrite, and are said to consist principally of gold. Predominance of gold is also noted in the southward extension of this vein, on the Slayback claim, belonging to the Socorro Co.; this part of the vein was profitably worked many years ago, but developments were never more than surficial.

*Other Properties of the Cooney Group.*—Several small operations have been conducted, principally on the silver-gold veins, among which may be mentioned the Golden Eagle, the Floride, and the Queen. The Golden Eagle, situated on a tributary gulch, south of Mineral Creek, is interesting chiefly on account of the predominance of gold, constituting 70 per cent. of the total value, a ratio of silver to gold of only 9:1 by weight. The Floride workings are high on the north wall of Cooney canyon, and indicate a strong vein, included mostly within walls of rhyolite ash. The vein strikes to the southeast, and occupies a very obvious fault plane. The Queen mine is on the Queen vein, at the eastern side of the Cooney group. The vein, as prospected in this horizon, is hard, dense, and highly siliceous, but its chalcedonic character is not so pronounced as, for example, in the Consolidated workings on the same vein. It is stated that considerable ore of profitable grade was disclosed in the Queen tunnel, but the workings, which have been intact for over 20 years, show no evidence of extensive stoping. The early operators attempted to treat this ore by pan-amalgamation, resulting in quick failure.

## Conclusion

Successful operation in the Mogollon district demands continuous and persistent development. Too often development has been retarded until it failed to anticipate the requirements of production. An interesting approximation as to development standards may be gained by comparing the tonnage produced per foot of development work in several of the mines: the Fanny averages 10.5 tons per ft. (3.2 tons per m.) of development; the Deadwood, 11.8 tons per ft. (3.6 tons per m.); and the Pacific, on a narrow and unreliable vein, only 1 ton per ft. (0.3 ton per m.).

A considerable area of the district awaits exploration, and there is no reason for doubting that further work will disclose valuable deposits. The district suffers from its remote situation—it is probably the most detached important district in the Southwest today—but it has shown great enterprise in solving its own transportation problems. It is a potential source of a noteworthy silver and gold production, and deserves public interest in the development of its resources.

## DISCUSSION

H. G. Ferguson, Washington, D. C.—On page 300, Mr. Scott discusses the relation of the orebodies to the wall rocks. I think that that is less a function of the influence of wall rocks on the solution than of the original distance below the surface at the time of ore deposit. It is probable that the surface at the time of ore deposition was about 500 ft. above the base of Number 11 of Mr. Scott's tabulation on page 293. East of the Queen vein, not shown on Mr. Scott's map, there is a series of veins, similar in position to those on the west, but consisting principally of barren calcite. In the extreme western part of the district, the veins, as far as developed, have shown rich bodies of oxidized ore at the surface, but have been disappointing in depth. In the central part, in which are the mines described by Mr. Scott, we have practically the full zone of original deposition favorably situated for development.

# Rapid Formation of Lead Ore

BY H. A. WHEELER, E. M., ST. LOUIS, MO.

(New York Meeting, February, 1920)

THAT lead and zinc deposits are the result of prolonged, slow deposition is the idea of most students of ore deposits, and in many cases, where the ore-bearing solutions have been very weak or the precipitating conditions have been very feeble, the ore accumulations have undoubtedly taken a long period of time. As these two vital conditions have been subject to great variations in the different mining camps, the following example is highly interesting in showing that deposition can be very rapid under favorable conditions, geologically speaking, and also extremely recent. In fact, the writer is inclined to regard most, if not all, of the Mississippi Valley deposits of lead and zinc as probably being very young, at least not older than Tertiary, if not Quarternary, in age.

The extensive Joplin lead and zinc district, in southwestern Missouri, which is about 60 years old and has numerous mines that were abandoned from 1 to 50 years ago, shows that the lead and zinc today are in an active condition of solution and redeposition, although these two metals occur in the predominant form of sulfides. The mine waters from the older and more or less abandoned portions of the district are frequently so acid, mainly from the oxidation of associated pyrite, as to necessitate the use of wooden pipes and pumps.

The first boom in the Oklahoma zinc fields, which is the southwestern extension of the Joplin district, occurred when ore was accidentally struck in drilling a water well. Among the mines then opened up was the Mission mine, at Lincolnville, Ottawa County, Okla. This mine was opened in 1903 in lean "sheet ground," or a flat blanket formation of chert carrying disseminated zinc blende, or "jack," and galena. The orebody was very free from iron pyrites and the concentrates were very high grade, the "mineral," or lead concentrates, assayed 80 to 85 per cent. lead and the jack, or sphalerite, assayed 60 to 65 per cent. zinc. The orebody was very shallow, occurring at a depth of 75 to 95 ft. (22 to 28 m.) and very wet, requiring three or four pumps to handle the water. The mine was operated until 1914, when it was closed by litigation and immediately filled with water.

The second Oklahoma boom, five years ago, opened up the large, active camps of Picher, Tar River, and Century, which are located 2 to 5 mi. west of Lincolnville. They are considerably deeper, being farther down on the westwardly dipping monocline that emanates from the Ozark

uplift in central Missouri and they all had to contend with very heavy reservoir water. The orebodies in these camps occur in a highly fractured, more or less parallel series of zones that are very open; and while the pumping is exceedingly heavy for one or two years in opening a mine, amounting to 1000 to 3000 gal. per min., the inflow is very moderate after the reservoir water is exhausted. The Church and Mabon properties at Century, 3½ mi. northwest of the Mission mine, had a very heavy pumping proposition for over a year, but by 1916 the ground water was exhausted, since which the Admiralty mine, one of the largest and deepest producers of this group, has had barely enough water for its mills.

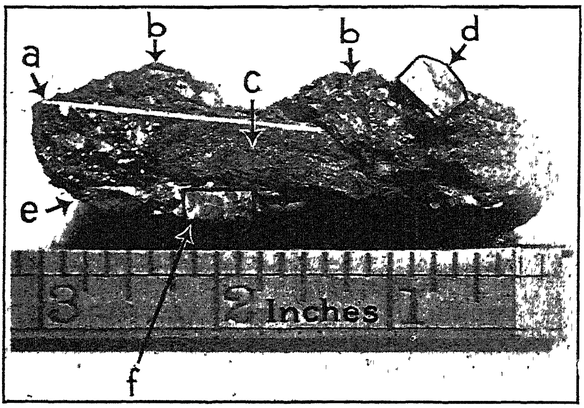


FIG. 1.

On examining the Mission mine in 1916, it was found to have been completely drained by the Admiralty mine, 3½ mi. distant, and the old iron tools left when it was abandoned two years earlier were found to be more or less coated with crystals of galena or lead sulfide. These crystals ranged from 1/50 to ½ in. (0.5 to 12.7 mm.) in size and were intimately associated with limonite (hydrated iron-sesquioxide) from the oxidation of the metallic iron. Some of the crystals were attached directly to the metallic iron, while others were enclosed in an amorphous limonitic mass that adhered tightly to the underlying iron and contained fragments of chert.

The accompanying full-size photograph of an old railroad spike illustrates the mode or occurrence of this recently deposited galena and the spike probably rested on the floor of the drift in the position shown. On the upper right-hand portion of the spike is an irregular mass of intermixed galena, limonite, and chert sand—the latter in angular fragments 1/20 to ¼ in. (1.27 to 6.35 mm.) in size. On this limonitic mass is a well-crystallized cube of galena about ½ in. in size. On the lower left-hand portion of the spike is another galena crystal ½ in. in size and ad-

jacent to it is a piece of chert at least ½ in. long, which probably rested on the floor of the drift and is firmly cemented to the spike by limonite. On the left face of the spike, small galena crystals $\frac{1}{20}$ to ¼ in. in size are firmly attached to the spike by a limonite cement and above them, resting on top of the spike, is another mass of intermixed galena crystals, limonite with small fragments of chert.

It is not unlikely that the entire spike was covered with galena and limonite before it came into the possession of the writer, as the specimen had been roughly handled during the months it was an object of curiosity in a mining office. The galena decidedly predominates in the limonitic matrix and, if scraped off, it would probably assay over 45 per cent. in metallic lead—possibly over 60 per cent.

As the time from which the mine was abandoned until it was again drained was about two years, we have on this specimen of a 3 in. by ½ in. by ½ in. spike a very copious deposition of lead sulfide in a period of time that, geologically speaking, is utterly insignificant—in fact, "in a few geological moments." The precipitating conditions were of course exceptionally favorable, as the metallic iron would quickly decompose any soluble lead salts, whether as sulfate or bicarbonate. That the mine waters were extremely dilute seems probable, as the surrounding ore deposit was low grade, yielding 1 to 5 per cent. and was very shallow, or only 75 to 100 ft. deep.

It is probable that the mine water was in active circulation during the submergence of the mine, especially during the last part of the period of deposition. This district is so open with underground channels that mining activity in any part of the area would start a movement of the reservoir waters, even if there were no natural currents before mining was inaugurated, and considerable mining was in progress during the entire submergence of this property from 5 to 10 mi. west, or farther down on the monocline. In fact, in drilling this area with churn drills, the sludge, or comminuted drillings, are frequently washed away, on striking openings in the ore-bearing horizons, by the underground currents.

The presence of more or less chert in fragments in the adhering limonite certainly seems to indicate that the subterranean currents were sufficiently strong to move sand particles as coarse as $\frac{1}{20}$ to ¼ in., until they were trapped or arrested by the slimy limonite produced on the spike by the decomposition of the lead-bearing solution.

Another interesting feature brought out by this specimen is the fact that the large ore crystals do not necessarily mean slow growth. For while most of the galena occurs in small crystals, ranging from $\frac{1}{10}$ to ¼ in. in size, at least two crystals are ½ in. in size and one of the latter—on top of the specimen—was one of the latest to be formed, as it rests on top of a mass of the limonite-galena mixture.

The evidence of this Mission mine leaves no doubt as to the orebodies of this district being a case of descension and that the ore horizons are steadily being lowered as erosion slowly planes away the surface. Since the Mission mine was reopened in 1916, it has been necessary to drill to a depth of 300 ft. to obtain sufficient water to operate the concentration mill, although it produced a great excess at 100 ft. in 1904, which was before the mines to the west and farther down on the gentle monocline had been discovered.

The writer is indebted to Mr. J. P. McNaughton, of Miami, Okla., for this very interesting and highly instructive specimen of rapid ore deposition, as also for the historic details.

## DISCUSSION

Frank L. Nason, West Haven, Conn. (written discussion*).—The very strong conviction possesses me that even for the agile minded it is a long jump from a railroad spike to the conclusion that "The evidence of the Mission mine leaves no doubt as to the orebodies of this district being a case of descension and that the ore horizons are steadily being lowered as erosion slowly planes away the surface." I do not know whether Mr. Wheeler takes himself seriously; whether advocates of "ascension" or "descension" generally regard themselves in the same light. Assuming the serious view of this and similar discussions, I submit a few data that have come under my observation.

There is no known element that will not go into solution. The converse is also true; once in solution, any given element can be precipitated. The above general principle can be put in another form—migration. Given a metal at rest (in an orebody) glacial action or water erosion may move it and redeposit it Brought into contact with a solvent, gas or water or both, it moves with this solvent to a point when either a precipitating agent causes its deposition directly or indirectly by evaporation. Equally possible, at least, the dissolved metal may be discharged into an open stream and find its resting place in the ocean. The solvent bearing the metal may be uprising or it may be descending.

Along the New River in Virginia, five large zinc deposits have been worked—the Delton, Bertha, Fulton Farm, Austinville, and Ivanhoe. Each of these has produced large tonnages of oxidized zinc and lead ores. These were originally in limestone, presumably in the form of sulfides; when mined they were embedded in residual clay. One mine has been opened to 400 ft. (121 m.) below the surface; this is 200 ft. below the base level of drainage, New River. The oxidized ores are at the summit of a bluff, about 250 ft. above the river level not more than 1700

* Received Feb. 6, 1920.

ft. horizontally from the flood plain of the river. In spite of the migratory nature of lead and zinc ores, lime and magnesia have completely disappeared. Water channels crossing the orebodies are the sources of many springs, two long adits driven to and across the orebodies discharge a large volume of water; neither lead nor zinc is in solution, for before the mine was reopened the mining village depended on these adits for domestic water.

On the other hand, in the Kelly district, New Mexico, when potable water is obtained from springs originally in or near the zinc and lead deposits, the water seriously affects strangers or transients owing to the zinc in solution. In Tennessee, in the Powell, Clinch, and French Broad River valleys, deposits of zinc and lead oxides occur within a few hundred feet of the rivers and at elevations of from 50 to 200 ft. above the rivers. There is no evidence that these deposits have lost metal by leaching, even under the most favorable conditions for such loss. In contrast with these is the Ollie Bell mine in Wisconsin. Inferentially, at least, this was originally a large mine and a rich one. Now, while it has more than traces of both lead and zinc, it is far below economic grade. Again, inferentially, this mine has been "robbed" by solution. But the resting place of the metals has not been found.

In Mr. Wheeler's own territory, considerable pockets of zinc sulfide ore have been mined. The ore was embedded in soft, black, carboniferous shales. These deposits are on the crest of the Ozarks; carboniferous cherts cover large areas of the Ozarks. Inferentially again, the Ozarks were covered by the carboniferous, part of the remnants is zinkiferous still; inferentially, the whole may have been as highly zinkiferous as the rocks of the Joplin area. There is thus no tangible evidence that the zinc of the eroded fields enriched, much less originated, the Joplin zinc fields. Instances without number may be cited of zinc and lead deposits being undiminished in volume under the most favorable conditions for such loss; as numerous instances may be cited where, apparently at least, the bulk of these metals has been removed by solution. The metals of these deposits having been supposedly removed by solution, what has become of them?

Mr. Wheeler's railroad spike shows conclusively that a metal in solution can be rapidly precipitated when in contact with a reducing agent. But all of us know this. For ages, possibly not as far back as the "Tertiary" or the "Quarternary," men have taken commercial advantage of this fact. Cement copper has been, and is today, gathered from cupriferous mine waters by conducting these through troughs filled with scrap iron. Zinkiferous and plumbiferous waters may flow, as Mr. Wheeler states, from the Mission mine to the Admiralty mine, 3.5 miles distant, but does this flow exist without artificial disturbances of level? With this artificial stimulus, has there been a quantitative deter-

mination of zinc and lead in solution at the Mission and compared with the volume in the Admiralty mine waters?

Comparing Mr. Wheeler's premises and conclusions, it appeals to me as a very plain case of *non sequitur.* Another instance of the same line of reasoning is the Miami copper mine, which is held to be a glowing example of descensional origin. To me, another explanation appears as being tenable—capillary attraction and evaporation. It may be far fetched, but what I regard as similar, though not necessarily identical, phenomena may be observed any time in the West. Given an alkali-encrusted basin and a heavy rain, the alkali is dissolved and carried into the earth, but when the inevitable drouth follows the alkali-laden water is brought to the surface, the water evaporates, the alkali crust reforms.

James F. Kemp, New York, N. Y.—Two years ago, while passing through St. Louis, I spent an hour with Mr. Wheeler and saw this specimen. It seemed such an interesting one that I asked him to write the description for the Institute. I wish we had the spike here so that you could see it, for there might arise, in the mind of one who had not handled it, the possibility that the spike lay in some gravel on the bottom of the drift in which there were crystals of galena, which became cemented to it. As I looked at the spike, such an explanation did not seem to be as reasonable as the one that the crystals had actually grown upon the spike while it lay there and that it was oxidized where it lay. We may, therefore, perhaps assume that the crystals have positively formed upon the spike. They were fed by waters draining the mine, and its surrounding unmined ground, much of which, as Mr. Wheeler tells us, carried from 1 to 5 per cent. of lead.

These percentages afford a much more favorable case than anything that could be imagined in the original formation of orebodies. The crystals formed in 2 years. It is very interesting to have the exact facts of time and of the artificial pumping and of the stimulus to the currents which led to the formation of the crystals. Whether from this we can go so far as to infer that the lead and zinc deposits are of such recent formations as is stated in the paper is where most of us will probably differ from Mr. Wheeler's conclusions. There are two or three considerations that we ought to have in mind if we put before us such possible conclusions. We know that if a deposit of zinc blende in limestone is brought within the belt of weathering the zinc goes into solution and migrates downward and is precipitated on the uneven and unaltered surface of the bed-rock. The resulting calamine may penetrate a short distance into crevices in the underlying limestone, but it does not go far. Wherever we have an opportunity to observe the old workings that have stripped off and taken out these crusts, that conclusion is inevitable. Now, the zinc salt does not precipitate as zinc blende, so far as I know,

and if we attempted to explain our zinc-blende deposit in this way we would have to find some way of accounting for its precipitation as a sulfide under the ordinary circumstances prevailing in nature.

Let us also look at the lead for a moment. Wherever great deposits of galena in limestone have been brought within the belt of weathering, they have changed over to the sulfate and then to the carbonate ultimately and have stayed where they were. We do not find them going down, but are indebted to them for the early great sources of production of oxidized lead ores in the West. Just how this lead that we find remaining in the carbonate, or sometimes in the sulfate, is to start on its wanderings and go down, reach quiet stationary ground water, migrate extensively through it, and be again precipitated as a sulfide is one of those serious questions that would have to be satisfactorily explained in regard not only to the descension theory, but to the formation of great bodies of galena in a brief space of geological time.

I frankly confess that the Mississippi Valley lead and zinc deposits are, to my mind, about the hardest nut to crack in all the ore deposits in this country. That, however, we may lightly and easily refer them to descending currents and precipitate them again as sulfides is a questionable explanation.

Several years ago, in the Joplin district, there was found a pocket of white zinc sulfide of which fortunately I inherited a small bottleful and which I have saved as an interesting specimen from the district; but to explain the great bodies of zinc blende, the sulfide itself, precipitated by some reaction that must be so different from our ordinary experience with the descending solutions of zinc, is a pretty hard problem. On the other hand, to bring the theories of ascending waters from some igneous source in a region that is marked by the absence of igneous rocks, generally, so far as we know, takes us into the region of speculation and is a view difficult to support by any actual observation of geological facts. Nevertheless, I do believe that Mr. Wheeler has presented to us a very interesting case of what seems to be the quite rapid formation of crystals of galena around a strongly reducing agent in a period of 2 years and from currents that seem to have been stimulated by the pumping of a mine 3 or 4 miles away.

H. V. Winchell, Minneapolis, Minn.—I do not think that Mr. Wheeler suggested that the spike was sulfide, or that it was necessary in every case to find an equivalent volume or amount of pre-existent sulfides. I sent to your Chairman, some years ago, some little pebbles, nuggets, of native copper from Alaska taken from the beds of glacial streams, on which there was a certain amount of chalcocite deposited, and I do not suppose he will suggest that they came from hot solutions. Of course, that was not present in large quantity, but it seemed to be an

instance of the formation of chalcocite in recent times in gravels under a very low temperature. But I have noticed an interesting occurrence of galena in Utah, where any other theory than that of descension and deposition from cold solutions seems rather unlikely. At Ovar Canyon, Utah, there has been displacement of the rocks amounting to several hundred feet, at least. Near the top, the limestones are exposed in vertical places many hundreds of feet; the canyon, I think, is about 2000 ft. in depth. Ore has been deposited in the limestone strata, which have been subjected to slight displacements. In one place a deposit of ore was faulted comparatively recently. In working the bedded ore deposits, ore was discovered in this fault and a shaft was sunk to the depth of 800 or 900 ft. and ore was extracted. For a distance of 300 ft. below the lower limestone bedded ore, the ore in the fault had a width of about 3 ft. and was solid galena with a little zinc. Below the depth of 300 ft., the ore had a width of about 7 ft. and it was all oxidized—oxidized ores, carbonates, etc. down to 625 feet.

On the 700-ft. level, drifts were run out along this fault for 200 or 300 ft. in each direction. This ore was in a shoot not more than 500 ft. in length, perhaps not over 350 ft. in length and the fault fissure contained no ore, either oxidized or sulfide. It was an open crack, on the walls of which very finely illuminated calcareous substances of a beautiful color had been deposited, cemented in which were fragments of limestone that had fallen from above and had wedged in between the walls, which were about 4 ft. apart on that level. On the 800-ft. level, the drifts had been extended as far as the limits of the air shaft. There was no mineralization, the fissure was about 1 ft. or less in width, the calcareous deposits on the walls had a thickness of from 8 in. to 2 ft., and there was no ore. On the 900-ft. level, a crack remained, the fissure being almost entirely filled with this calcareous deposit. It had been completely filled and had to be broken out in order to run the drifts.

Now, there is a case of a mine upside down—sulfide ore on top and oxidized ore underneath. We know that the faulting is comparatively recent, and if our geological friends can tell us how that galena got in there without being deposited by downward flowing solutions, or how those solutions got heated, we would appreciate it.

W. Lindgren, Cambridge, Mass.—Iron spikes, and similar objects, with galena crystals have been found at various times in the Mississippi Valley and elsewhere; there are a number of such cases on record, some of them by the Geological Survey of Missouri.

Such a deposit can easily be accounted for by saying that the lead is present in solution in various forms and is precipitated by reducing agents. But no complete solution of the problem can be reached unless we know the character of the mine water in this particular mine. Mr.

Siebenthal considered the water in some of these mines in Oklahoma to be of deep-seated origin. He thought that they belonged to the ascending artesian circulation, although that circulation no doubt has slowed up considerably during the last period, as shown by the fact that the waters could be quite easily pumped out.

I can see no objection to the theory of the precipitation of galena by descending surface waters. I think most mining geologists present have seen zinc blende coated with galena; it is a very common thing. It shows very nicely in some of the zinc blende of the San Juan region, Colo., and lots of other places. Irving called attention to it a long time ago, and I have collected such specimens repeatedly where the galena appeared as a thin gray film and where there could be no reasonable doubt of the descending water being responsible for the precipitation.

I do not doubt that solution and precipitation in the natural ground waters offer great possibilities in the way of mineral formation. Some time ago nodules of barite with galena and pyrite were repeatedly found in the deep borings in Louisiana. They are on record in the publications of the Louisiana Geological Survey. There we have, at present, mostly chloride waters, in which barium is constantly present; this barium must be precipitated as barite when sulfate waters from a higher level come into contact with these chloride waters.

An interesting bulletin (No. 693) published recently by the Geological Survey and written by Messrs. Mills and Wells, shows how, in many cases, oil wells have been clogged by crystallized barite at the point in the oil sand where the oil issued, the precipitation occurring from the contact of two waters of different chemical composition.

I have always held that, in addition to the barium present in those chloride waters, there is also lead probably as chloride. If chloride of lead is present, a precipitation as galena might be effected by mingling with other subterranean waters, some of which, no doubt, also contained hydrogen sulfide. It does not seem to me that it is necessary to draw on the resources of the barysphere, or centrosphere, to explain the Mississippi Valley deposit, and as far as galena being deposited by descending waters is concerned I am surprised that anyone should seriously question that.

S. H. Ball, New York, N. Y.—I want to call the attention of those who are looking for an igneous origin for the ores of the lead and zinc mines of Missouri to one statement of Buckley's in his article entitled "Lead and Zinc Deposits of the Ozark Region"[1] on the occurrence of basic dikes in St. Genevieve County, Miss., which lies immediately to the northeast of the main disseminated lead mines in St. Francis County.

[1] "Types of Ore Deposits," edited by H. Foster Bain, 105. San Francisco, 1911.

He states that these dikes contain visible quantities of galena and blende, as I remember it. The fact that I make this statement does not mean that I believe that igneous rocks had anything to do with the deposition of the lead and zinc mines of Missouri, but I think it is well worth while for those who are looking for an igneous source to make a careful examination of these dikes.

I have seen, I think, every type of lead and zinc deposit in Missouri, and while I do not question for a moment that many of the so-called, using Whitney's old term, gash veins, were undoubtedly deposited by descending waters, I question very strongly whether the more important and larger orebodies are of such an origin. In Washington County, I have seen perhaps fifty shafts starting down on a foot or perhaps only a knife-edge of galena, and in each case pinching out from 20 to 30 ft. below the surface. I have no doubt that those were formed by descending waters. On the other hand, as regards the deposit from which Mr. Wheeler's specimen came, I am very strongly of the opinion that an origin by descending waters is almost impossible.

The Miami District is being prospected, or has been, by churn drills, which pass through, before reaching the ore horizon, anywhere from 20 to 200 ft. of impervious shales. The eastern edge of the shale is approximately from 5 to 10 miles to the east of Miami; that is, the older rocks are dipping westward in that region. I cannot believe but that Siebenthal was absolutely correct when he said that they were deposited probably by artesian waters, which are laterally moving and then ascending. It seems to me that possibly the deposits of this region should be broadly classed with some of the Red Bed sandstone copper deposits, in which chalcocite does not necessarily replace some other sulfide and in which I think probably the greatest amount of evidence at least indicates relatively cold waters.

Alan M. Bateman, New Haven, Conn.—This paper is of interest because it brings to our attention one more instance of deposition of sulfides by cold solutions, and data of this nature are valuable. However, I do not think we are justified in assuming an origin of ore deposits by such processes as are described, because a spike happened to become coated by crystals of galena. The conditions of natural deposition of ores are manifestly different from those which pertain under artificial conditions of mine workings.

Mr. Winchell drew attention, by way of comparison, to some chalcocite-coating native copper that he had found in gravels at the bottom of a glacial stream in Alaska and which he considered to indicate deposition under cold conditions. I also have found some chalcocite on native copper in Alaska, not, however, in the bed of a glacial stream, but in the center of a small glacier. It is obvious that under those

conditions of temperature the chalcocite could not have been coated on the native copper by means of solutions. Furthermore, it is clear that the chalcocite must have coated the native copper prior to the Glacial Period, for the specimen must have fallen on the glacier during the time of its building up in the Glacial Period. Incidentally, that glacier is a copper mine. The ore is composed of fragments of a disintegrated vein outcrop which became cemented by a gangue of ice, and the orebody has been developed by tunnels penetrating the glacier. It is, I believe, the only known instance of a glacier being mined.

I found the source of that particular specimen of chalcocite coating native copper, namely, the greenstone of the cliffs overhanging the glacier. The joints and little blowholes of the greenstone lava are filled by small pieces and slabs of chalcocite and native copper, as well as other minerals. The erosion of the cliff resulted in the falling of pieces into the glacier during its formation. The coating of chalcocite on the native copper therefore took place in pre-Glacial times when climatic conditions were not so cold. Similar pieces may have found their way into stream gravels and it is barely possible that the specimen described by Mr. Winchell may have had such an origin. If this is the case, we cannot conclude that the deposition of the chalcocite on the native copper took place in the icy cold waters of the glacial stream.

H. V. Winchell.—That is very interesting, but I do not think it applies to the samples to which I referred. This was not one pebble, nor only a few—there were pebbles of barite, pyrite, native copper, all of which had by superior gravity gone to the bottom of the gravels, and all of them were cemented together. I do not suppose that those pebbles were originally derived in the form in which I found them from a solid deposit of pebbles of barite, pyrite, and native copper formed in a single lode by the deposition of copper sulfide.

E. S. Moore,* State College, Pa. (written discussion†).—I have previously described occurrences of secondary galena in the Broken Hill orebody.[2] In this deposit lead carbonate frequently occurs in crystals shaped like arrowheads. I have, in a collection of Broken Hill minerals, several small crystals of this type. At first, they were regarded as galena pseudomorphs but when broken open they were found to be crystals with a thin coating of galena on the surface. This galena is undoubtedly secondary.

In the North mine, considerable galena occurs along fissures in the

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* Dean, School of Mines, Pennsylvania State College.

† Received Mar. 13, 1920.

[2] Observations on the Geology of the Broken Hill Lode, New South Wales. *Econ. Geol.* (1916) **11**, 327–348.

stopes and drifts where water is circulating; this is regarded as secondary in origin. Just what may have been the chemical process producing this secondary galena in these two cases is not known. Lead sulfate, coming into contact with iron pyrite under oxidizing conditions, might produce galena; or a combination of lead sulfate and zinc sulfide might produce this mineral. All these compounds are present in this orebody as well as chlorides, bromides, and iodides of silver and other secondary minerals.

Some laboratory experiments performed for me by Messrs. L. J. Youngs and C. M. Nevin showed that a 1-per cent. solution of hydrochloric acid would readily produce lead chloride from galena and that when a piece of limestone was placed in a solution of lead chloride, lead carbonate was precipitated on the limestone by reaction with the calcium carbonate. Whether this was a simple carbonate or the bicarbonate was not determined. A very dilute solution of hydrogen sulfide was next applied to the limestone carrying traces of lead carbonate and immediately the lead was thrown down as a sulfide resembling galena. As to the source of $H_2S$, if this were the agent in the Broken Hill deposit which precipitated the lead from carbonate waters, there does not seem to be any great difficulty in accounting for its presence in small amounts along fissures in an orebody which was undoubtedly deposited by igneous agencies.

H. A. Wheeler (author's reply to discussion*).—Contrary to the general assumption of that fact, the spike was not the only specimen that showed precipitated lead; all the iron tools in this mine were incrusted with galena but the spike was selected because it was more convenient to handle and had, perhaps, larger crystals than the other samples that had been saved. It was also less damaged by careless handling.

The white, pulverulent, freshly precipitated zinc sulfide that Professor Kemp alludes to was observed by W. P. Jenney more than 30 years ago, while he was studying the Joplin district for the Missouri Geological Survey; he found it in considerable amounts, but the details I do not now recall. That there is considerable zinc sulfate in solution in the waters of many of the older mines of the Joplin district is well known and the surprising thing is that freshly precipitated zinc sulfide is not more common and that it is not found in larger quantities, as the access of surface waters carrying the soluble bicarbonate of lime would precipitate it from the mine water.

While I broadly referred to all the Mississippi Valley lead and zinc deposits—at least as we now find them—as being very young, I did not intend to convey the impression that they were all due to descension, in which class I also include lateral secretion, or the horizontal movement of mineral-bearing waters that primarily started from the surface. Lateral secretion is due to the more or less horizontal movement of min-

* Received Apr. 22, 1920.

eral-bearing solutions, which may be of deep-seated or surface origin, and probably most of the Mississippi Valley lead and zinc deposits have been mainly due to horizontal moving waters, rather than to direct descent or ascent. In the Joplin and Miami districts, I think that the ore-bearing solutions started from the surface and in their very slow, tortuous movement through the pores, seams, and openings of overlying limestones and shales, they have leached out and redeposited at greater depth—and probably at a decided horizontal distance from the point of origin—more or less of their mineral contents under favorable precipitating conditions and thus have produced the present workable orebodies. I do not, however, regard all the Mississippi Valley zinc and lead deposits as being due to descension; there is a sharp, well-marked difference between the disseminated types of orebodies in southeastern Missouri, which I regard as being due to ascension, as are also the lead and zinc veins of southern Illinois and western Kentucky, which have a fluorite gangue.

The zinc and lead deposits of the Joplin region (using this term to include the Miami, Okla., Galena, Kans., Granby, Carthage, and Joplin districts of Missouri), the shallow clay deposits, pipe-veins, gash veins and "brangle" deposits of central Missouri (including Franklin, Jefferson, and Washington Counties), and the Wisconsin region, including its extensions into Iowa and northwestern Illinois, I regard as due to descension—including lateral moving solutions of primarily surface origin. The rich Miami, Okla., deposits, which occur in open, highly fractured ground, probably received their ore-bearing waters at least several miles to the eastward, where the formation outcrops, as they are immediately covered by more or less shale, as remarked by Mr. Ball; they occur on the western slope of the Ozark plateau and are on the westerly drainage of over 150 miles of limestones and dolomites that carry more or less lead and zinc, although rarely in workable amounts. The object of the paper and specimen is to show that lead deposits can be formed extremely rapidly—from the geologic standpoint—even when the ore solutions are weak and the specimen certainly suggests that the Miami deposits may be very young, possibly as recent as Quaternary.

# Mud Volcanoes of Colombia, South America

BY STANLEY C. HEROLD,* TULSA, OKLA.

(Chicago Meeting, September, 1919)

A FEW notes on the occurrence and significance of mud volcanoes in Colombia may be of interest at the present time, owing to the renewed activity in geological exploration of the coastal regions bordering the Caribbean Sea. The best-known volcanoes are found a few miles from the seacoast. One is situated to the south of Puerto Colombia; the second, to the east of Cartagena near the village of Turbaco; and the third, near the village of Puerto Escondido south of the Sinu River. Others are reported in more inaccessible localities.

Usually they stand 2½ or 3 ft. (0.762 or 0.914 m.) above the surrounding ground at the crater, though several craters from 10 to 20 ft. (3.04 to 6 09 m.) apart will often build up an extensive mound at low gradient. The shallow mound may cover ½ or ¾ acre (0.2 to 0.3 ha.) with an elevation of 12 or 15 ft. (3.65 or 4.57 m.) at the center. The craters are formed of hardened blue clay; the inside diameters vary from 14 to 24 in. (35.56 to 60.96 cm.). The rim is continuous, except on one side where there is usually a cut 3 or 4 in. deep, thus allowing an overflow.

The craters are filled to the rim-cut with a thick mixture of blue clay mud and water, which is kept in agitation by escaping gas. Fig. 1 shows the bubble in the act of bursting and Fig. 2 the appearance of the surface of the viscous fluid a second or so later. The frequency of the explosions is determined by the weight of the column of fluid in the neck of the volcano, consequently, such explosions are not synchronous in a group of craters. Ordinarily, there are two or three explosions in about 3 minutes.

The supply of gas is small but steady and sufficient pressure must accumulate before the bubble can enter the column of fluid. The water is derived from the surface and enters the sedimentary strata not far distant from the volcano. During long dry seasons, as the supply of water fails the volcano dries and geyser action ceases.

The temperature of the muddy fluid is normal; the gas is dry and inflammable. No petroleum products come with the gas. On the surface of the fluid, there is sometimes a dark brown streaky scum, which

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does not separate from the water as petroleum would naturally do. This scum is very small in quantity and apparently consists of minute micaceous particles derived from the shales.

Mud volcanoes occur in faulted localities and are closely associated with these faults, in fact, the trend of faults can sometimes be determined by the position of separated groups or single craters. Not only do these faults allow the gas to escape but undoubtedly they also offer a channel to the surface water required in their action.

FIG. 1.—MUD VOLCANO IN THE ACT OF BURSTING.

FIG. 2.—SAME A SECOND OR SO LATER.

Mud volcanoes cannot be regarded as a manifestation of economic deposits of gas, at least in the immediate vicinity, for the supply is a small one and at very shallow depth, probably within 40 ft. (12.19 m.) of the surface, judging from the time required for the bubble to reach the top after entering the fluid at the bottom. The latter event is accompanied by a tremble of the ground and a muffled rumbling. If there ever existed a considerable supply of gas nearby, it has long ago escaped by the fault. It is very likely that this gas is migrating under difficulty from a distant source and if so gas may be encountered when the proper strata are drilled into. The volcanoes indicate neither the loca-

tion of the source nor whether the amount that might be obtained would be commercial or not. They merely show that under proper sand and stratigraphic conditions, gas may somewhere be found. Buying, leasing, or drilling an estate merely because it has a mud volcano upon it is certainly not justified. As an indication of the existence of petroleum in the region, mud volcanoes should receive still less consideration. The best that can be said for them is that they are very interesting as mud volcanoes.

# Earth and Rock Pressures

BY H. G. MOULTON,* WASHINGTON, D. C.

(New York Meeting, February, 1920)

THE INCREASING scale of mining operations over the past decade, particularly in connection with the exploitation of large bodies of comparatively low-grade copper ores, has made necessary the study of the behavior of rock under failure. When extractions of as high as 40,000 tons a day are reached in steam-shovel operations and as high as 20,000 tons per day in underground mining operations, it is evident that excavations soon become of such size that the behavior of the rock will often be dependent on fundamental principles applicable to more or less homogeneous materials under stress, rather than on weathering or local planes of weakness.

In the ordinary rock excavations encountered in civil-engineering practice, such as excavations for subways, highways, railroads, etc., rock faces are not carried to depths sufficient to develop any pressures, or consequently any failures, except those due to loose masses resulting from local planes of weakness. When, however, orebodies whose tonnage is measured by hundreds of thousands are to be extracted by steam-shovel operations in open pits, it is necessary to determine if possible in advance the slopes that the sides of the pits will assume, for these slopes influence the shape of the excavations and thereby the amount of ore that may be extracted by the open-pit method. When large orebodies are mined underground at depths of 300 to 500 ft. (91 to 152 m.) by caving methods, the surface subsiding as the ore is withdrawn, it is necessary to determine in advance the outermost limits of fracture in the subsiding surface, for on the delineation of these limits depends the location of surface improvements, mills, smelters, etc. and also the location of development work, such as hoisting shafts, main haulage levels, and all other underground openings that must remain undisturbed during the period of ore extraction.

Heretofore, in civil-engineering practice, it has been assumed that earth and rock behave differently in failure. For the calculation of earth pressures, formulas are used in which the assumption is made that earth banks fail by sliding along the slope determined by the frictional resistance of the material. The line of failure and the resulting pressure is held to be dependent on the angle of repose, which is different for varying

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types of material. This necessarily leads to an assumption of an angle of repose of 90° for rock, or in other words, that except for local planes of weakness, homogeneous rock settling above an excavation would break directly over the point of failure.

Experience in earth excavations has shown the error of such assumptions as to earth pressures. Mining work carried on in New York City, in connection with the subways, has given an opportunity to study the behavior of earth under failure and has indicated that there is a common law underlying all materials possessing cohesion and not of a semifluid nature. Some years ago this subject was discussed by J. C. Meem,[1] who pointed out that pressures in timbered cuts were different from what would be expected from existing formulas for earth pressure. E. G. Haines,[2] in discussing Mr. Meem's paper, pointed out that an earth bank in failure normally broke back to a distance equal to half its height irrespective of the character of the material and expressed a belief that a cut in solid rock, if carried to a sufficient depth, would act in the same manner. This paper was inspired by reading Mr. Haines' discussion and it represents an attempt to carry on the study of failures in excavations from earth to rock so that the common laws underlying the behavior of materials failing in excavation may be developed.

During the time which has passed since Mr. Meem's paper was presented, additional data have become available with respect to sand, loam, clay, and hardpan through the construction of the new rapid transit routes in New York City. This work involved the support of open cuts and tunnels by timbers over extensive areas, and proof of the soundness of Mr. Meem's theories is to be found in the general adoption, by contractors on the work, of the types of support in the development of which he played so important a part.

For the analysis of rock pressures and the study of rock slides, it is evident that in ordinary excavations for railroads, streets, or subways depths will never be reached sufficiently great to overcome the shearing strength of the material, or to obtain results that are not simply the reflection of small local planes of weakness in the rock. There is, however, becoming available a mass of data from the large copper mines of the Southwest, where orebodies are removed by top-slicing, shrinkage-stoping, or caving, the upper part of the deposit, which is usually valueless capping, being allowed to settle after the ore is removed, with consequent subsidence of the surface and development of large cracks. Since in these operations depths of 400 to 500 ft. (121 to 152 m.) are often reached, it becomes possible to obtain measurements of the relations existing between the surface settlement cracks and the stoped areas below, which will throw light on the behavior of rock in failure.

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[1] *Trans.* Amer. Soc. Civil Engrs. (1908) **60**, 1. [2] *Idem.*, 27.

Through the kindness of Mr. Pope Yeatman and Mr. J. Parke Channing, and particularly through the coöperation of Mr. C. B. Lakenan, general manager of the Nevada Consolidated Copper Co., the writer has been able to secure cross-sections representing typical caved stopes at some of the leading prophyry copper mines and also sections of deep gravel banks in placer mines operated by the hydraulic method. These are given in connection with sections from earth tunnels upon some of the New York City subway work; there is thus obtainable, for the first time as far as the writer knows, a comparison of the behavior of material under failure in excavating operations ranging from moist sand to hard rock, and it appears that in all cases the action is independent of the character of the material, provided the depth reached is sufficient to develop pressures in excess of its shearing strength.

If the behavior of material in failure is independent of its character, the pressure from a given material is a function of its unit weight and of the depth of the excavation, and not of the character of the material. The important deduction to be drawn from this is that formulas for bank pressure should not include any functions of the angle of repose. A certain type of earth may have an angle of repose approximating 45°, whereas the angle of repose of hard rock, except for the influence of local seams and planes of weakness, is 90° from the horizontal; yet the behavior of each is identical at depths proportionate to their respective strength and they assume the same shape and break to the same angle in cases of bank failure.

## Classification of Materials

For the consideration of pressures in supported banks, materials may be divided into three general groups:

Class A, materials that act as solids. This group covers the entire range from moist sand to the firmest rock. Every type of material within this group will stand with a vertical face to a certain depth; below that depth it will fail in shear in exactly the same manner and leave exactly the same shape of bank after failure.

Class B, granular materials. In this group are included dry sand, dry gravel, broken stone, wheat, or in general any material composed of separate grains and lacking cohesion. Such materials behave in a distinctly different manner from those of the first group. Generally speaking, it is probable that they, too, will follow one general law if considered in quantities proportionate to the size of the grains and that a mountain of large stones would show the same lines of pressure and angles of repose as a bin full of wheat or a basket full of fine sand. However, the shapes of individual particles may be a factor and materials of this class may possess angles of repose determined by such shapes.

Class C, semi-liquid materials. In this group are included quicksand, wet clays, and, in general, all ground encountered in excavating that is so saturated with water that it will develop pressures of a hydrostatic nature. At depths sufficiently great, certain types of clay become plastic and flow in such a manner as to develop pressures proportionate to the depth. It is probable that if great enough depths were reached in excavating operations many types of rock would also be found to be semifluid and would flow under pressure. In certain mines in India and Brazil, there are encountered so-called "flexible sandstones," which are apparently plastic under comparatively slight pressure and will stand a considerable degree of deformation in small sections without rupture.

It is not the purpose of this article to discuss materials of either of the last two classes. Those in Class B are subject to analysis by the methods used in calculating stresses in grain bins. Material in Class C is to be considered from the standpoint of hydrostatics with due attention to its specific gravity. The behavior of material in Class A is entirely different from that of either of the other two, and when any material encountered in excavation contains sufficient cohesiveness to remove it from the class of purely granular substances and yet is not so viscous that it will flow under pressure, it must be considered as a solid and analyzed from the point of stress relations in elastic bodies, whether it be damp sand or the hardest rock.

## Tunnels in Sand and Earth

In 1916, the Degnon Contracting Co. began the construction of section 1A of route 12, a double-track subway tunnel, under Flatbush Ave., Brooklyn, N. Y. The construction required an excavation approximately 40 ft. (12 m.) wide by 21 ft. (6.5 m.) high, which was lined with concrete in the form of a double arch having a center bearing wall and umbrella section. The depth at subgrade varied from 94 ft. (28.6 m.) at the northern end of the section, to 35 ft. (10.6 m.) at the southern end. On the southern part of the section, over approximately one-third of the total distance, the tunnel was excavated by the use of a roof shield resting on concrete bearing walls at the side; for the remaining two-thirds of the length of the section a timbering method was used.

For most of the distance, the material encountered was clean sand free from silt or clay, showing a graduation in sizes that made it suitable for use in concrete mixtures. Many large boulders of trap rock were encountered. In places hardpan was found, with its top surface often coming to elevations above the upper limit of the tunnel section. The subgrade of the finished structure was above the permanent water level, though the sand was at all times damp from the surface down. This

excavation, therefore, represents a fair test of the behavior of damp sand under partial failure involving settlement above an underground excavation.

Mr. Waldo C. Briggs, engineer in charge of this operation for the Degnon Contracting Co., made careful studies and surveys of the breaks occurring in the surface as a result of the slight settlement incidental to his tunneling operations, and has furnished the writer with records from which the accompanying diagrams have been drawn. While the dia-

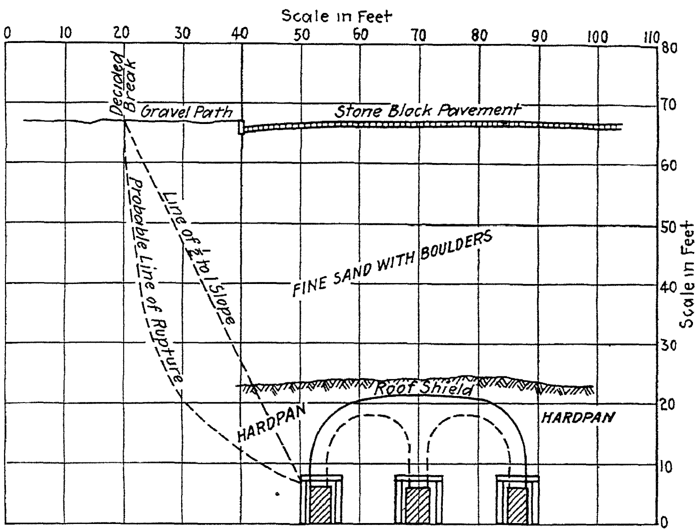


FIG. 1.—TRANSVERSE SECTION OF SUBWAY TUNNEL; ROOF-SHIELD METHOD.

grams are exact with reference to the outline of the tunnels, the location of the surface, and the relation between the excavation and the breaks in the surface, they are only approximations with respect to thickness of concrete, sizes of timbering, etc., having been drawn only to show the probable lines of rupture and the horizontal and vertical measurements of the points of subsidence with reference to the excavations causing subsidence.

Fig. 1 represents a transverse section of the tunnel near the southern part of the contract, where the roof-shield method was used. In this operation, side-wall and center drifts were carried in advance and concrete bearing walls built therein. At this point the hardpan was firm enough to arch over the top of the drifts, with very little pressure on the timbers, and had no more of the excavation than this been removed, no settlement would have occurred. Later, the rest of the section was excavated by the use of a roof shield rolling on the side walls. This operation broke

up the arching effect of the hardpan and permitted settlement at the point where the ground had been disturbed in excavating the drifts for the side

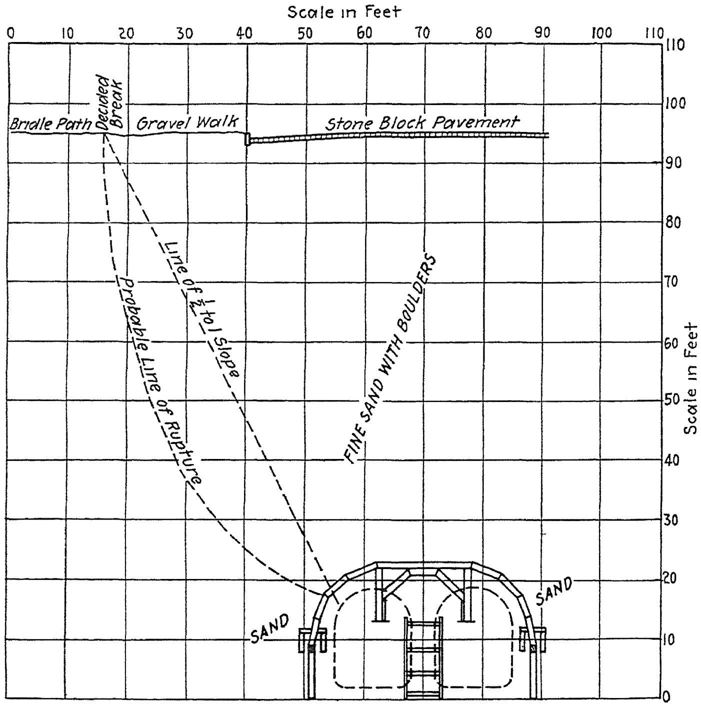


FIG. 2.—TRANSVERSE SECTION OF TIMBERED SUBWAY TUNNEL.

walls. It will be noted that a line projected on a slope of ½ to 1 from the farthest decided break meets exactly the nearest upper corner of the drift, which is the point at which settlement might be expected. The "probable line of rupture" shown is, of course, purely an assumption, there being no possible way of surveying it; yet from observations of breaks in open cuts in similar material, it is known that rupture takes place along a curved line, as shown, and that the break occurs at a distance back from the edge of the bank equal to half its height. It is a fair assumption, therefore, that rupture will occur in a similar manner through settlement above underground excavations when the relations existing between the breaks and the points of settlement below correspond. The center and side-wall drifts in Fig. 1 were driven

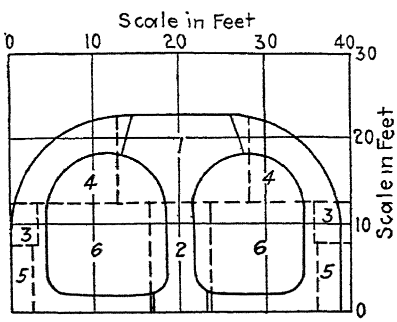


FIG. 3.—SEQUENCE OF EXCAVATING OPERATIONS.

in September, 1916. The shield passed this point in November, 1917, while the decided break in the surface was first noted on Nov. 23, 1917—or immediately after the passage of the shield broke up the arches in the hardpan over the side-wall drifts.

Fig. 2 is sketched from a transverse section near the northern end of the contract. At this point a rather unique timbering method was used. Fig. 2 shows the timbering details, and Fig. 3, the sequence of operations. The top heading was first driven and timbered with posts, caps, and knee braces. A sheeted trench was then sunk to subgrade, and, within this trench and under the timbers of the top heading, an umbrella section of concrete was built to sustain the roof. Wall-plate drifts were next driven and segmental sets placed to connect the umbrella section with the wall plate. The wall plate was then underpinned on posts; it was possible to excavate the remaining part of the material and complete the concrete lining within the timbers. The top heading and center trench were excavated in July, 1917. The wall-plate drifts and the excavation for the segments were taken out during the latter part of October and early in November, 1917. The break in the surface was first noted Nov. 23, 1917.

It is apparent that the first settlement, which would also be the maximum settlement, would be over the timbers of the top heading. After this heading was supported on the concrete umbrella section, the next point where settlement could occur would be over the wall-plate drifts, which, however, were very small and carefully lagged over top and sides. Over these small drifts the sand would probably arch and be generally self-supporting. The next operation—that of excavating over the segmental timbers—would tend to break up this arch and cause settlement, yet the thrust from the segmental timber arch would incline the resultant from the vertical and tend to throw the point of settlement, and consequent rupture, inside the wall-plate drift. As shown in Fig. 2, a line carried from the decided break in the surface downward on a slope of ½ to 1 intersects the outline of the tunnel at a point between the post of the top heading and the outer post of the wall-plate drift. Since no measurement of the actual point of rupture in the excavation is possible, this drawing does not prove the case as definitely as did the former one; yet it shows that the same angle of slope falls between the points that limit the possibilities of rupture. Fig. 2 may then be considered an approximate confirmation of the theory under discussion.

Figs. 4 to 7 show the reason for expecting the ground to break at points indicated in the sketches. Fig. 4 shows the shield just starting from the shaft. A study of its framing shows that it acts as a truss and has no arching action; there is, therefore, no force to throw the resultant out of the vertical and it tends to bring the line of settlement inside the outer line of the excavation for the side-wall drifts. Fig. 5 shows the

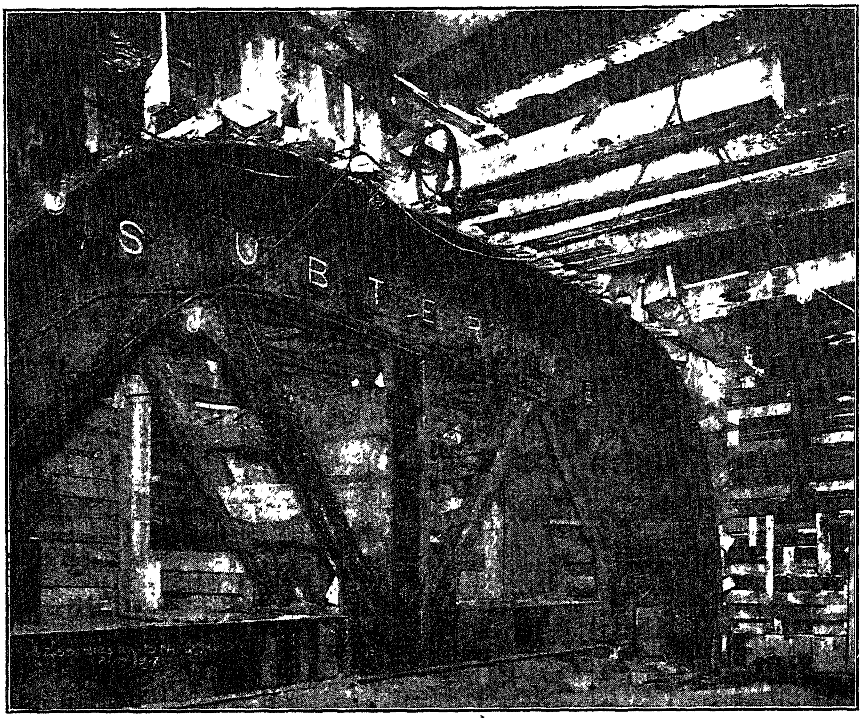

FIG. 4.—ROOF SHIELD STARTING FROM SHAFT.

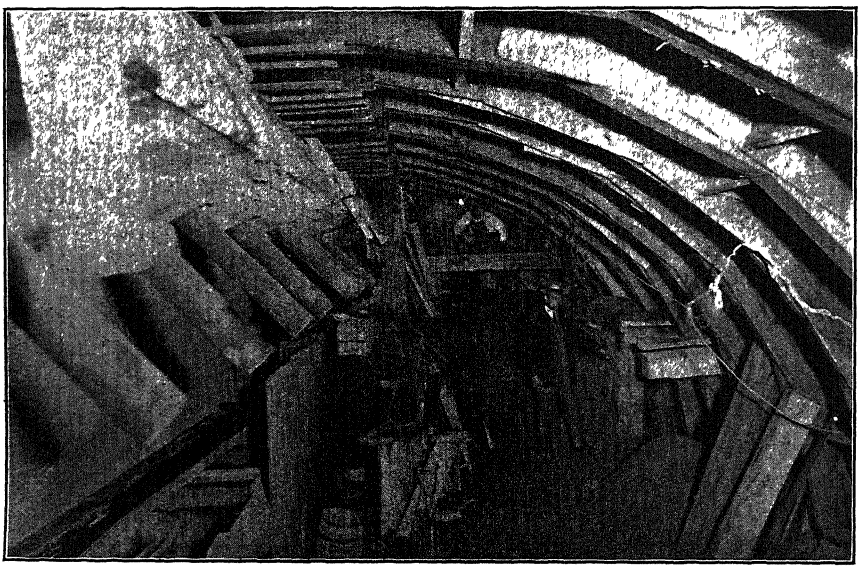

FIG. 5.—TIMBER ARCH RIBS BETWEEN CONCRETE UMBRELLA AND WALL-PLATE DRIFT.

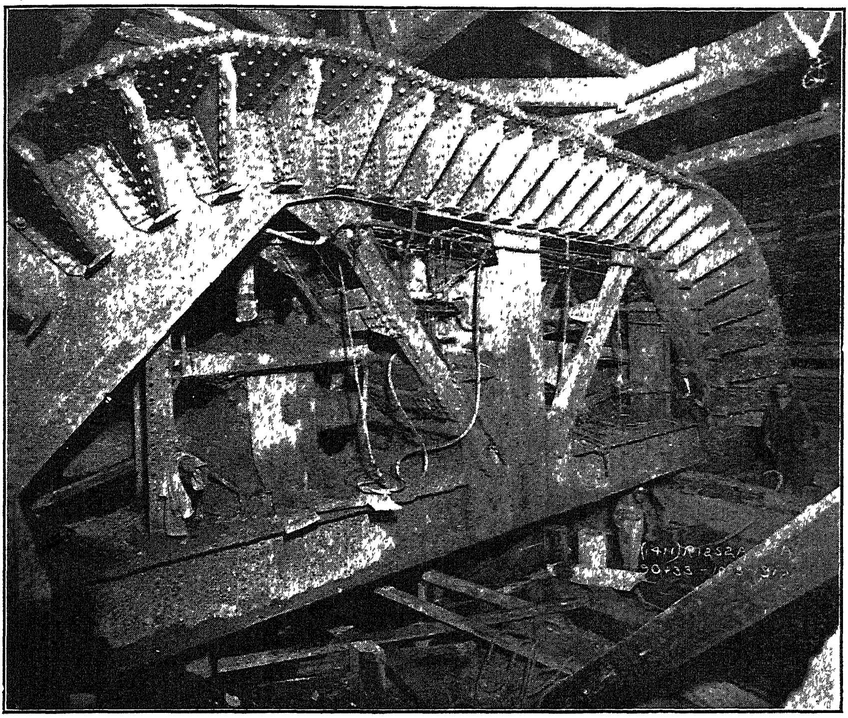

FIG. 6.—CUTTING EDGE OF ROOF SHIELD JUST AFTER BREAKING INTO SHAFT.

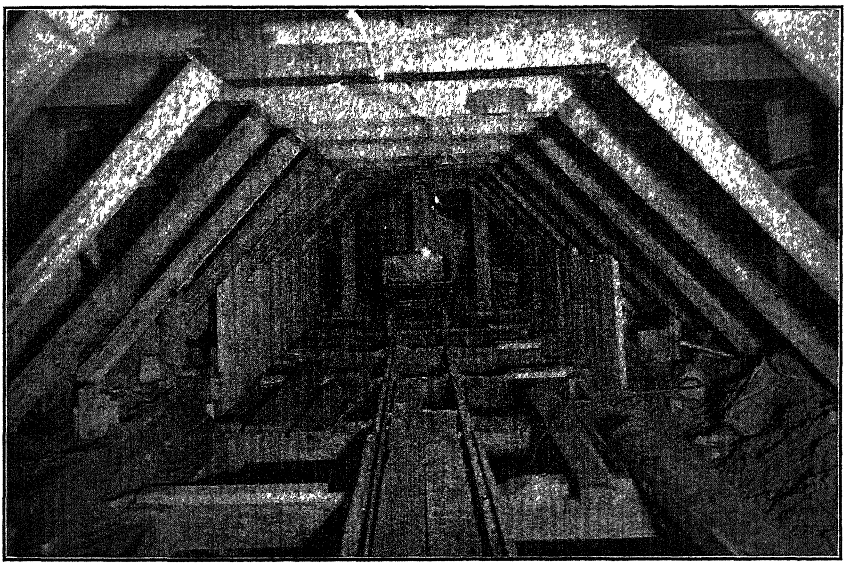

FIG. 7.—METHOD OF TIMBERING TOP HEADING AND EXCAVATING CENTER TRENCH BEFORE CONSTRUCTING CONCRETE UMBRELLA SECTION.

timber arch ribs of the timbered section between the concrete umbrella and the wall-plate drift. Due to the elasticity of the timbers and the absence of truss framing, there is a considerable outward horizontal thrust, which tends to prevent settlement at the wall plate; there is every expectation that the arch will deflect about halfway between the concrete roof and the wall plate and thus bring the line of rupture out to the point indicated in the sketch.

Figs. 6 and 7 are given to illustrate the methods of construction. Fig. 6 shows the cutting edge of the shield just after it has broken into a shaft. Fig. 7 shows the method of timbering a top heading and excavating a center trench just prior to the construction of the concrete umbrella section.

The same type of settlement, with corresponding ratios existing between horizontal and vertical measurements, has often been noted in the past, and examples are cited in the discussions accompanying Mr. Meem's paper. These two cases are of particular interest in that they are based upon exact measurements by an observant engineer and involve two types of tunnel construction—roof shield and timbered section; they are also important in that one case covers fine sand above hardpan, with subgrade at a depth of 67 ft. (20 m.), while the other is in sand throughout at a depth of 94 ft. (28.6 m.). Taken in connection with former instances and with the observed behavior of earth and sand under failure in open cuts, the mathematical relations covering the location of breaks in the surface and the shape of the probable line of rupture may be considered definitely ascertained for these classes of material.

## Gravel Banks in Placer Mines

At placer mines in which the gravel is removed by the hydraulic method, unsupported banks are often carried at heights ranging from 25 or 30 ft. (7.6 or 9 m.) in some of the smaller properties of southern Oregon or northern California, to 600 ft. (182 m.) or more at the La Grange mine in Trinity County, Calif. The method normally followed consists of playing a stream of water, under high pressure, against the base of the bank until enough material is removed to cause a cave. The gravel thus brought down is washed down the sluices, in which the gold is collected.

Nearly all gravels worked by the hydraulic process contain enough cemented material to bring them within the class of material under discussion; that is, the banks will stand vertically until a depth is reached at which the shearing strength of the material is overcome by the pressure of the overlying strata. If the theory of earth pressures on which this article is based is correct, the maximum amount of earth could be removed by exactly the process followed in this type of mining; that is, by washing out from the foot of the bank enough material to weaken its resistance and assist in caving. In very deep banks of cemented material, which offer

unusual resistance to the streams from the giant and to the shearing stresses, the process is often accelerated by drifting into the bank at the toe and turning off powder chambers in lines parallel to the line of the toe. A charge of explosives is then fired, which loosens the bank and permits it to cave more readily.

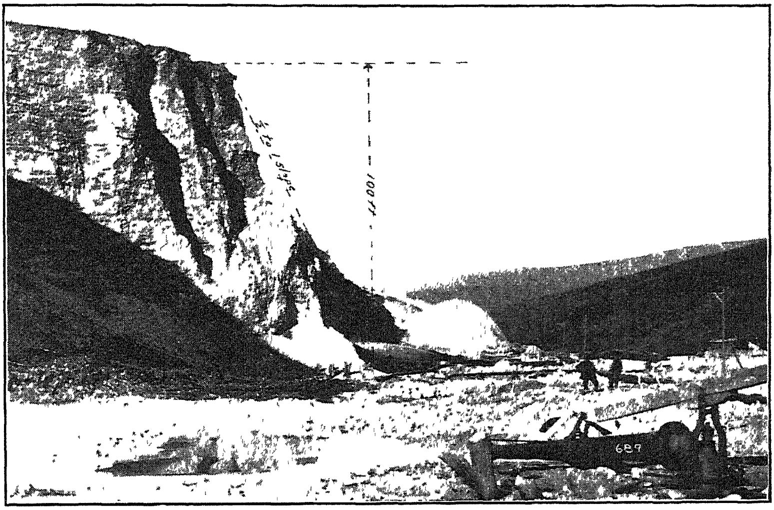

FIG. 8.—TYPICAL PLACER-MINING OPERATION.

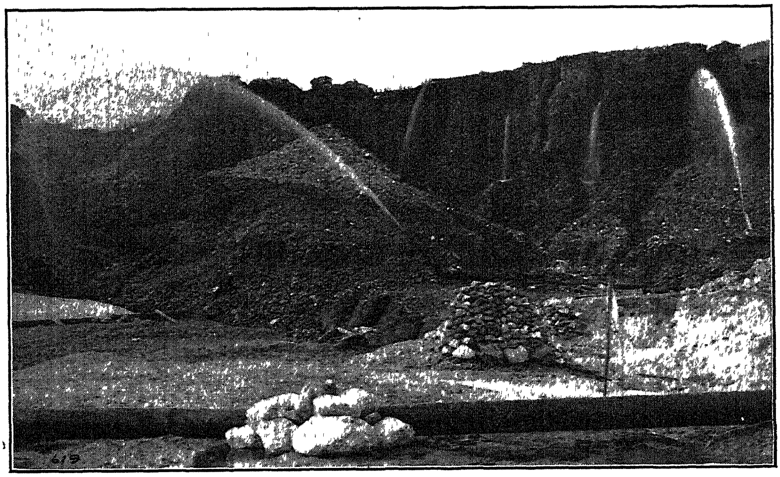

FIG 9.—EXCAVATING GRAVEL CONTAINING A GREAT AMOUNT OF ICE.

Figs. 8 and 9 show a special case in placer-mining operations, as the cementing material consists of ice. The photographs were supplied by Mr. C. H. Munro, acting general manager of the Yukon Gold Co., operating in the vicinity of Dawson, Yukon Ter. Fig. 8 shows one of the

operations of the Yukon Co., at Cheechaco Hill, near Dawson, in September, 1911. The bank is approximately 100 ft. (30 m.) high and the ground contains a small enough quantity of ice to permit its removal by the normal method of operation; that is, by removing the toe of the slope with the giant. This bank is composed of comparatively small gravel, the boulders, as a rule, being less than 6 in. (5 cm.) in diameter and seldom running above 1 ft. in diameter. The bank shows clearly the normal line of rupture, and a slope of ½ to 1 drawn from the top strikes exactly at the edge of the toe.

Fig. 9 shows a bank in the vicinity that is being mined by an entirely different method, because of the great amount of ice in the gravel. If this bank were undercut by the stream and operated as in Fig. 8, excessive amounts of material would be brought down in the form of large blocks cemented together by ice. The water is, therefore, being used as a thawing medium and is allowed to run down over the tops of the banks from ditches. The streams from the giants are also played against the sides of the bank near the top. As the ice thaws, the gravel ravels off and falls to the foot of the slope, where it is removed by streams from the giant at the bottom and is washed through the sluices. This case is the reversal of the usual method and is made necessary by abnormal conditions of frost. It is practiced only in districts affected by climatic conditions of unusual severity, such as in Yukon Territory. Fig. 8 represents a typical case in placer-mining operations and shows the exact conformity of the behavior of gravel banks to the theory under discussion.

## Rock

In ordinary civil-engineering practice, excavations are not carried to depths sufficient to develop the true behavior of rock under failure. In open cuts for railroads, subways, streets, etc., firm rock free from mud seams or other local planes of weakness will stand with vertical faces until it commences to ravel off from the effects of weather. No true test of the behavior of rock can be obtained except from excavations at depths great enough to overcome its resistance to the shearing stresses resulting from its weight. Such tests are now available from the results of caving operations carried on in extracting ores at some of the large copper properties in the southwestern part of the United States.

The Nevada Consolidated Copper Co., operating at Ely, Nev., was the pioneer among the so-called porphyry coppers, working large bodies of low-grade copper ore on an extensive scale. Three types of mining operations have been carried on at this property—steam-shovel mining in open pits, top-slicing in the Veteran orebody, and caving by the branch raise system in the Ruth orebody. Mining operations are still under way in the Ruth orebody and the surface is subsiding progressively. It will be some time before the subsiding surface will have cracked to its

ultimate limits, and the data obtained from this section of the mine at present is not conclusive. Mr. Lakenan states that, from his experience, he expects cracking to occur out to the intersection with the surface of a line projected upward from the stopes at an angle of around 60° to 70°. This corresponds very closely with the angle of a ½ to 1 slope, which is 63° 26′.

The Veteran mine, however, furnishes excellent material for study. This was the first large-scale underground mining operation among

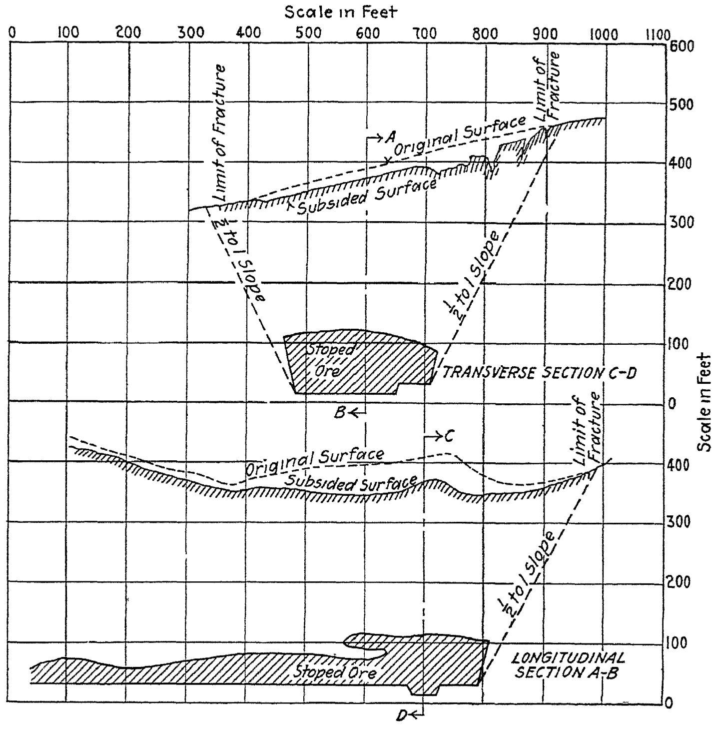


FIG. 10.—TYPICAL SECTIONS OF VETERAN MINE.

the porphyry coppers. The ore stoped averaged around 2½ to 3 per cent. copper. The copper minerals occurred principally in porphyry, and to some extent in limestone, the ground overlying the orebody being largely limestone. The limestone varied considerably in hardness and, consequently, the capping was not homogeneous and broke rather irregularly.

The Veteran orebody was completely stoped out about six years ago and sufficient time has elapsed for settlement to progress until the surface has probably reached a condition of stable equilibrium. The mine was worked by the top-slicing system. By this method, the irregular upper portion of the body is extracted with square-set timbering, one or two floors high as may be necessary to level off and start with a uniform

top surface. The lower posts are then blasted and the ground allowed to cave, bringing with it a mat of broken timber. Another slice is then taken in the form of a top heading 8 or 9 ft. high across the orebody, the timber mat above being picked up on posts. These posts are again blasted and the capping again dropped the height of the slice, this operation continuing until the bottom of the orebody is reached The mat of broken timbers follows the extraction of ore and prevents dilution with barren capping. Top-slicing is a method particularly well adapted to an orebody such as that of the Veteran mine, where the ore was of comparatively high grade and the capping so low in value that it was considered best to avoid all possible dilution. In orebodies of lower grade, which will not stand as much expense for mining, particularly where the capping contains some values so that the effects of dilution are not as serious, cheaper methods, such as caving by the branch raise system, or by shrinkage-stoping, are in more general use.

Fig. 10 gives typical transverse and longitudinal sections, showing the space formerly occupied by the orebody and also the subsided surface above the stopes. These sections show the behavior of limestone in failure at depths of from 400 to 500 ft. It will be noted that lines projected upward on a slope of ½ to 1 from the point of probable failure in the stopes intersect the surface very close to the outer limits of fracture. The errors of observation, if they may be given such a designation, are relatively small, the difference between the actual and theoretical points of outermost subsidence, as compared to the horizontal projection of the ½ to 1 slope line, are respectively 25 ft. in 155 ft. (7 6 m. in 47 m.) on the left-hand side, and 25 ft. in 215 ft. on the right-hand side of the transverse section, and 10 ft. in 195 ft. on the longitudinal section. The average error is only slightly over 10 per cent.; this case may, therefore, be taken as confirming very closely the Haines theory of the behavior of solids under failure in excavating operations.

It is interesting to note the relatively small proportion of the stoped area accounted for in the area between the original surface and the subsided surface. This is accounted for by the arching action in the broken capping, the separate blocks of ground and the voids together occupying a greater space than the original material. The usual experience in stoping ore by the shrinkage system, where the broken material is maintained at a level that will permit the miners to reach the backs of the stopes, is that 1 ton of ore must be drawn out of every 3 tons blasted—in other words, 2 tons of broken ore in the stopes occupy as much space as 3 tons in place.

Miami Copper Co. operates a large underground mine at Miami, near Globe, Ariz. Development of the company's property commenced soon after that of the Nevada Consolidated, and from a standpoint of time it was the second one of the porphyry coppers to

extract comparatively low-grade ores on a large scale by underground mining methods.

Copper ore occurs at Miami in the form of chalcocite scattered through the rock in small grains and films. Part of the orebody is in granite porphyry and the remainder in schist. The limitations of the orebody are determined by commercial considerations. In the upper part, the values

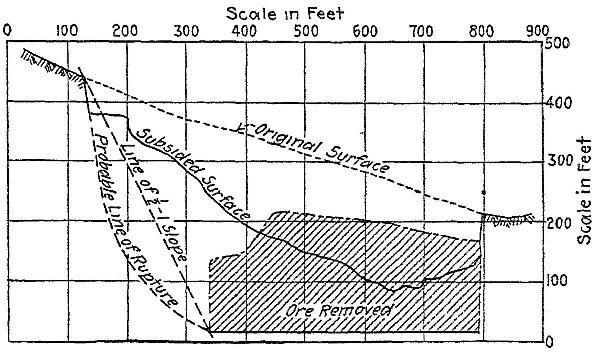


FIG. 11.—TRANSVERSE SECTION *A–B* THROUGH CAVED STOPE IN GRANITE.

have been reduced to a considerable extent by leaching, and the copper remaining has been oxidized into a form that does not lend itself to recovery by concentration. It is therefore desirable to separate the sulfide ores from the lower grade capping; and on account of the relatively low grade of the ore, the mining method must be one resulting in low costs

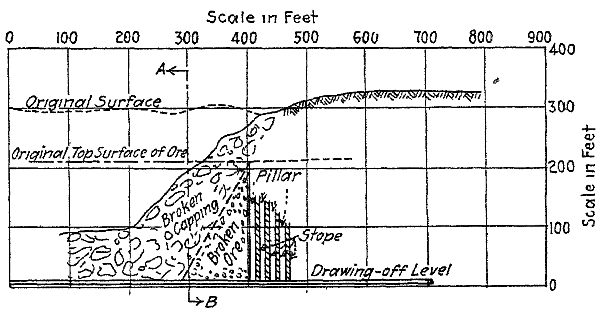


FIG. 12.—LONGITUDINAL SECTION OF SHRINKAGE STOPE.

of operation. The company's financial success has been achieved through large-scale operations with cheap mining and milling costs.

Mr. J. Parke Channing, vice-president, and Mr. F. W. Maclennan, assistant manager, have furnished surveys showing the subsidence of the ground. From these, Figs. 11 and 12 have been prepared. Two methods of mining have been used—top-slicing and shrinkage-stoping; the section given is from a part of the property where the shrinkage

system was used. Briefly outlined, the method consists of stoping out alternate bands across the deposit, drawing off only enough material to leave a space between the broken ore and the top of the stope, leaving pillars of ground between the stopes for temporary support. The stopes are spaced 25 ft. from center to center and their width is such as to leave partitions between them of from 5 to 15 ft., depending on the character of the ground. As the stopes reach the upper part of the orebody the pillars commence to crush, which action is assisted by undercutting. The weight of the material above breaks the ore into pieces small enough to be drawn out in cars. In this method of mining, it is only necessary to drill and blast about 20 per cent. of the ore, the remainder being crushed by its own weight and the weight of the capping. The broken ore is drawn out until the dilution with capping lowers its grade below the commercial limitations of value, and the broken capping settles as the ore is drawn, filling the space left by the stope. The longitudinal section, Fig. 12, gives a general idea of the method; it is not complete in detail, for the various sublevel entrances and shafts are not shown.

The transverse section, Fig. 11, shows the relation existing between the underground excavation and the breaks in the surface. On the right-hand side of the sketch the wall stands vertical; this is due to the fact that the excavation at this point is only 200 ft. (60 m.) deep; or not deep enough to overcome the shearing strength of the rock. On this side of the orebody the material is harder On the left-hand side of the sketch, where the excavation is deep enough to develop the true behavior of the material and where the rock also has a lower shearing strength, the ½ to 1 slope projected downwards from the break in the ground surface intersects exactly the bottom edge of the stoped area. If the broken capping that now fills the excavation were removed, there is no reason to doubt that the line of rupture would be found as outlined on the sketch.

Comparing a transverse section of the Miami Copper Co. mine with that of the Veteran mine of the Nevada Consolidated Copper Co. shows the difference in the type of breaks. In Fig. 11 the ground is broken sharply and the surface has slipped down, leaving a sheer wall at the point of fracture, while in the case of the Veteran mine subsidence is not as sharply marked and the outermost fracture is only a small crack. The different types of subsidence are due partly to the fact that the ratio of the height of the orebody and the distance from the top to the surface are much greater in the case of the Veteran mine, and the capping therefore has an opportunity to settle slowly and develop an arching action. The mining method employed also results in much quicker settlement at the Miami property, the ground being undercut and caved and not let down gradually, step by step, as it were, as in the top-slicing method used

at the Veteran. Fig. 13 shows the subsided surface at the point covered by the transverse section in Fig. 11, and shows the decided breaks and steep slopes resulting from quick settlement.

The results from the Miami Co.'s operation check exactly the theory under discussion; and taken in connection with those from the Nevada Consolidated mine show that the behavior of porphyry undercut and caved by the shrinkage-stoping method in Arizona is the same as that of limestone settling over stopes worked by top-slicing in Nevada.

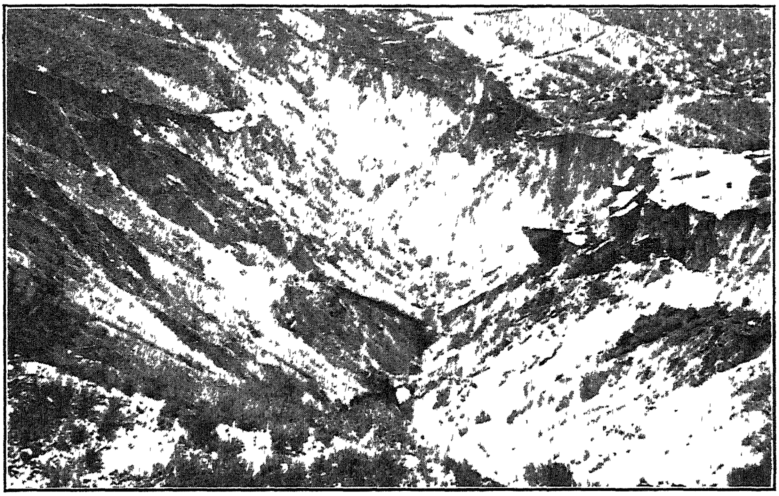

FIG. 13.—SURFACE AT POINT COVERED BY TRANSVERSE SECTION IN FIG. 11.

Mr. Louis Cates, general manager of the Ray Consolidated Copper Co., at Ray, Ariz., made for use in this paper extensive surveys and studies of the subsidence over the stopes of his property. Unfortunately the results so far are of little value for comparative purposes and only show that no inference can be drawn as to the behavior of rock under pressure unless predicted upon results from excavations carried to depths sufficient for the weight of the material to overcome its shearing strength.

The Ray Consolidated Copper property is somewhat similar to the Miami, and its underground orebody is mined by a similar shrinkage stope method. Stoping is proceeding on several levels, and the surface is still subsiding and has not reached an equilibrium. One small orebody has been completely removed, but this is only approximately 100 ft. below the surface. The situation is complicated by the occurrence of numerous hard crystalline ribs, which cause an irregular line of fracture. The shape of the stopes is also very irregular in the horizontal plan, and as a result the angle between the outermost fracture and the nearest edge of the stope varies considerably. There is also a tendency for breakage to occur at the junction of the hard ribs and the softer material.

From the sections furnished by Mr. Cates, it is possible to obtain almost any result from a vertical out to an angle of 60° from the horizontal. Even these unsatisfactory results from the only worked out orebody of the Ray Consolidated property are, however, enough in themselves to show the fallacy of formulas that employ functions of the angle of repose, since for the purpose of such formulas it would be necessary to assume the angle of repose of solid rock at 90°, and the sections furnished by Mr. Cates show at several points lines of fracture at the surface located as far out as the intersection of a ½ to 1 slope from the vertical plane of the nearest edge of the stopes.

## Application to Mining Problems

If it were possible to remove, within the limitations of unit cost that govern the particular mining operation under consideration, all of the material lying within the lines of rupture, the sides of an open cut would have a shape similar to that drawn on several of the foregoing illustrations; that is, the walls would break practically straight down for approximately half their height, curving thence inward to the bottom, and a slope of ½ to 1 drawn from the top of the bank would intersect the junction of the line of fracture and the bottom of the excavation. Evidently this is the maximum limit of stability for an open pit of great depth. In steam-shovel operations, however, many other factors limit the boundaries of an open pit to slopes much flatter. In the first place, such excavations proceed in benches, each bench being wide enough to contain the steam shovel and the tracks over which the excavated material is removed. Economies of steam-shovel operation require that the line of cars pass the shovel and therefore the bench must be wide enough to contain at least two lines of tracks. Furthermore, the method of mining involves blasting in the successive faces, particularly near the bottom of the slope, and the rock dislodged by the blast assumes an angle of repose due to its then being in loose particles and not in a mass possessing cohesion. The slope assumed by the broken rock under these conditions is approximately 45°, so that the ultimate slopes of the boundaries of steam-shovel pits, being determined by practical problems of operation, may be roughly calculated as the angle of which the tangent is the total vertical depth of the excavation divided by this same figure plus the sum of the widths of the benches. The writer has not had the opportunity to make any extensive studies of steam-shovel pits of this type, but judging from data that Mr. Lakenan has submitted, a slope of 40° would be a good one to use in estimating approximately the amount of ore recoverable by operations of this type. In cases where it is possible to carry unusually high banks above the benches and the ground is sufficiently homogeneous so that slides will take place only as a result of failure from shearing stresses, and not along local planes of weakness,

lines of maximum fracture may be plotted in accordance with the principles discussed earlier in this paper. Perhaps, through the application of these principles, it might be possible to work out methods of open-pit excavation in which the benches would be at much greater differences of elevation, crushing of the rock being accelerated by blasting at the foot of the bank so that it would cave from above, the fallen material being removed by the steam shovel up to the line of rupture. In other words, with the knowledge that rock and gravel both behave the same under failure when excavations are carried to depths sufficient to develop pressures in excess of the shearing stress of the material, it might be possible, at least in certain types of rock, to work out mining methods somewhat similar to those used in placer-mining operations, the material being brought down in the same way but removed by steam shovel and cars instead of by the hydraulic giant.

In planning the development of underground-mining operations involving stopes of large area and at considerable depth, slopes of approximately 70° (½ to 1) should be projected upwards from the location of the outer boundaries of the stopes at controlling points and lines drawn connecting the intersection of such slopes with the surface. Within these lines will be contained the entire area of ground that will be disturbed by settlement cracks resulting from subsidence over the stopes, providing the rock is fairly homogeneous and there are no extensive fault planes or contacts between rocks differing greatly in hardness. Where extensive fault planes occur, consideration must be given to the possibility of extensive movement along such planes. In most cases, however, surface improvements and underground developments will be free from disturbance if located outside of the area delineated, but they are practically sure to be in danger if located within that area. As a matter of fact, those responsible for the development of large-scale underground mining operations probably already act more or less in accordance with this idea, having noted by observation the settlement of ground outside of the vertical boundaries of the stoped area.

## Earth Pressures

The foregoing series of comparisons of failures in materials ranging from damp sand to rock appear, to the writer, indications of the existence of a general law governing the line of rupture for all materials of Class A (materials possessing cohesion). The position of the line of rupture and its relation to the dimensions of the excavation appear to be independent of the angle of repose of the material and also independent of the amount of cohesion. It would also appear that the only material that can exert pressure upon a retaining wall or upon the sheeting of a cut is that which lies between the line of rupture and the retaining wall or sheeting. For a given depth of cut, always assuming the depth to be great enough to

overcome the shearing strength of the material, the volume will be constant irrespective of the character of the material.

The influence of cohesion on earth pressures is beginning to receive more consideration from the engineering profession; it has been discussed by Mr. William Cain in recent papers before the American Society of Civil Engineers. Even he, however, is inclined to employ functions of the angle of repose in his calculations of earth pressure, while the writer feels that for all materials possessing cohesion variations in the angle of repose have no effect upon pressures.

It is not the purpose of this article to attempt to develop a new formula for earth pressures. It is a matter of common observation among contractors and engineers in charge of excavating operations that the existing formulas are erroneous. Mr. Haines and Mr. Meem have made valuable contributions toward the study of earth pressures, and the fundamental principles they have discovered should be used as a basis for the determination of a suitable formula. In the meantime, it is unfortunate that textbooks and reference books still give for the guidance of the engineer formulas so utterly at variance with observed results; for example, the American Civil Engineers' Pocketbook, 1916 edition, gives the Coulomb formula, which results in an estimate of pressure directly proportionate to the unit weight of material, the depth, and a function of the angle of repose. The Public Service Commission for the First District, State of New York, uses a modification of this formula in calculating pressures on subway structures, said modification being in effect nothing but the assumption of a constant weight per cubic foot and angle of repose for all classes of earth materials.

Unfortunately there are practically no extensive authoritative measurements of earth pressures available against which calculations may be checked. To determine the true pressures from any material, it would probably be necessary to dig a trench, preferably timbered with horizontal sheeting, for a length of at least 1000 ft. (304 m.) and to a depth great enough to develop pressures in excess of the shearing strength of the material, and take regular extensometer readings on the transverse braces as excavation progressed. The operations would have to be under careful control, and the rangers compressed with screw jacks on temporary braces before the permanent braces were installed, so that none of the actual pressure of the earth would be employed in compressing the rangers and sheeting and that no compression would be driven into the bracing by wedging.

The most valuable series of tests with which the writer is acquainted was made by Max Miller, junior engineer, 7th division, Public Service Commission for the First District, State of New York, in a subway cut on Flatbush Avenue at Eastern Parkway in 1916. In these tests, the earth pressures were computed by measuring the deflection of rangers

over a bank area 22 ft. (6.7 m.) horizontally and 55 ft. (16.7 m.) vertically. The excavation at this point was 85 ft. wide and 80 ft. deep in a soil consisting of coarse sand mixed with about 20 to 30 per cent. of clay and some gravel. Because the lower 20 ft. of the cut was occupied by the completed structure, the measurements have no particular value below a depth of 60 ft. The graphic chart of pressures indicates that very nearly the maximum was reached at a depth of 15 ft.; also that the maximum pressure encountered was at a depth of 25 ft. and continued at about the same amount to 48 ft., when it began to decrease again. The results of these tests, which were carried on in Class A material, which should act as a solid under failure, indicate that earth pressures reach a maximum at a point slightly higher than half the depth of the cut, maintaining about the same amount of pressure over a distance equal to approximately one-quarter of the height of the bank, and that the total pressure on the timbers in the upper part of the excavation is greater than the pressure on the timbers in the lower part of the excavation. At a depth of between 15 ft. and 45 ft. below the surface, the pressure in this instance remained practically constant at around 900 lb. per sq. ft.

The following tabulation shows the extreme variance between pressures as calculated by the usual formulas and the results observed at the Flatbush Avenue cut. The angle of repose is taken at the usual figure for soil (36° 53′), and the weight of earth at 80 lb. per cu. ft. The pressures are given in pounds per square foot of area.

| Depth, Feet | Calculated Pressures | | Actual Pressures |
|---|---|---|---|
| | Coulomb Formula | P. S. C Formula | |
| 15 | 300 | 500 | 900 |
| 45 | 900 | 1500 | 900 |
| 55 | 1100 | 1833 | 300 |

The formulas used in these calculations are:

$$\textit{Coulomb's Formula.}—P = Wh \times \tan^2\left(45° - \frac{\phi}{2}\right)$$

$$\textit{Public Service Commission Formula.}—P = \frac{100\,h}{3}$$

in which $P$ = pressure per square foot at depth $h$;
$W$ = weight of earth per cubic foot;
$\phi$ = angle of repose.

In considering the nature and distribution of the pressures exerted on the timbers supporting a vertical bank in an earth cut, where the material is of Class A, the following points may be considered as well established.

1. The pressure on the timbers reaches a maximum at or slightly above a point half way between the top and the bottom, and the total pressure in the upper half of the cut is greater than the total pressure in the lower half of the cut. This is a matter of common knowledge among contractors excavating earth and supporting vertical banks with sheeting and braces.

2. It has been shown by Mr Haines that the line of rupture in an earth bank after failure is roughly formed by a one-quarter circle meeting the bottom of the cut, extending vertically in a tangent intersecting the surface at a distance back from the edge equal to half of the height. It has been the purpose of the foregoing to extend Mr. Haines' observations and prove that the relation he discovered is the expression of a law that applies to all cohesive materials considered under Class A. It also appears that no pressures should come from any of the material lying behind the line of rupture, but that all effective pressures must be due to the action of the material lying between the supported face of the bank and the line of rupture.

3. Mr. Meem has shown, by several experiments, that sand contained in a box can be made to support its own weight and even additional weight by arching action when all support is removed from a loose section of the bottom of the box. This arching action appears to be effective only when the lower part of the sand remains in place to act as centering, but little if any direct vertical pressure comes upon the sand which acts as centering. The fact that earth materials develop such arching action on a large scale in actual construction is shown by the fact that the timbering in tunnels and drifts does not show more than a small proportion of the pressure that would come upon it if it were required to support the entire weight of material from the roof to the surface. The writer's observations of construction work in New York City and elsewhere have wholly satisfied him as to the soundness of Mr. Meem's theories in this respect.

Granting the above premises, which appear fully sustained by observation and experiments, the deduction is that the pressures in a sheeted earth cut come from the arching action of the material lying between the line of rupture and the sheeting. This material is always broken and has settled to varying degrees, though often so slightly that settlement may not be apparent to the eye. Its cohesion has been at least in a large part destroyed and its behavior is different from that of the cohesive undisturbed material lying back of the line of rupture.

Fig. 14 is a possible explanation of the observed behavior of timber cuts. This figure represents an idealized cross-section of one side of an earth cut in Class A material, timbered with horizontal sheeting, vertical rangers, and horizontal transverse braces or struts at regular intervals. The timbering dimensions and locations of members are to be taken as

illustrative, and not as necessarily representative of suitable design for a particular case. If the pressures act as indicated in the sketch, the explanation of heaviest pressure in the upper part of the cut and little or none near the bottom is apparent. It is also apparent that a run of ground at the bottom would remove the centering, destroy the earth arches, and result in a collapse of the system. It is also apparent that the collapse would commence at the top, since it would be the result of the heavier top pressures of the opposite side. As a matter of fact, this is exactly the type of collapse that occurs when timbering supporting the sides of an excavation in sand fails as a result of a run at the bottom. It therefore follows that the material at the bottom transmits practically no pressure against the timbers, and yet it must at all times be supported

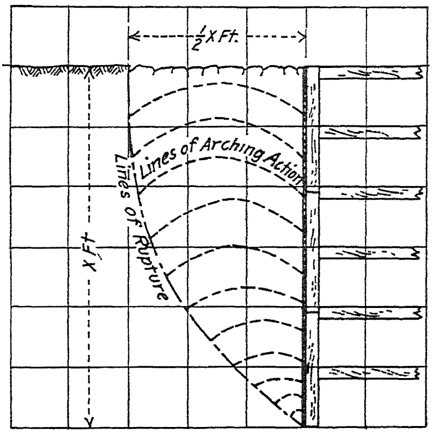


Fig. 14.—Idealized section of earth cut and timbering.

and held in place. Very light timbering will suffice at the bottom, yet it must be close and securely held. The use of knee braces, or any other form of support at the bottom, except definite transverse bracing, should be discouraged.

It is clearly understood that all of the foregoing refers only to materials possessing cohesion and acting as solids, here called Class A materials, these materials ranging in character from damp sand to rock. The analysis of a sheeted cut in wet clay or any other material that flows under pressure is distinctly different. In such materials pressures will increase regularly from the top toward the bottom and will be heaviest at the bottom The writer has observed certain cuts carrying clay near the bottom, in which the material acted as outlined in Fig. 14 for a considerable period, but after standing 18 months or more, the clay began to squeeze and pressures later developed at the bottom to such an extent as to bow the lower transverse braces out of line and crack the sheeting in places.

Materials are not always clearly in any one of the three classes arbitrarily assumed for the purposes of this paper. The classes shade imperceptibly into one another, and material encountered in excavation may partake of the nature of any two classes. In case of doubt, it is advisable to timber cuts uniformly with substantial transverse braces of the same size at regular intervals from top to bottom.

Considering earth pressures in the light of the theory set forth, a new light is thrown upon an interesting question sometimes encountered in subway construction. Assume that a cut has been excavated to a depth of 60 ft. (18 m.) in earth and that the transverse timber braces in the upper part of the cut, where the pressures are heaviest, are beginning to show signs of heavy loading to the extent that reinforcement will soon be required. Assume further that a steel and concrete structure is then placed in the lower part of the cut, up to a height of 40 ft. from the bottom, taking all of the pressure from the sides up to that height. The question arises as to whether or not this relieves any of the side pressure and makes unnecessary the provision of additional braces in the upper 20 ft. If it is true that pressures come from arching action, the answer is that there is no relief, that the timbers in the upper 20 ft. of the cut are still under the same pressure and require the same amount of reinforcement as if the permanent structure had not been placed in the lower 40 ft.

It is also interesting to give at least speculative consideration to the difference between material existing in a natural bank, or even loose material behind a retaining wall which becomes more or less compacted as it is placed, and the same material as it is piled loosely. A bank of sand and loam may stand for years, unless affected by rain and wind, at a slope corresponding to the line of rupture discussed in earlier paragraphs, yet material taken from the same bank and loosely piled will assume comparatively flat slopes, or, in other words, an angle of repose. The writer obtained a good illustration of this point in October, 1919, in the plant of the Chelsea Lumber & Box Co. near Klamath Falls, Ore. This company had a sawdust pile approximately 30 ft. high, which had accumulated during the summer of 1919 (each summer's accumulation is burned during the following winter). The material, which consisted in part of grains of sawdust from cut-off and other saws and in part of small chips from planing machines, assumed a slope of 45° or flatter as piled. During the late summer and early fall, one face of the pile was excavated with scrapers and the material used for filling in adjacent marsh lands. As the scrapers worked at the bottom, the sawdust caved and fell from above and the face assumed exactly the line of rupture common to the materials considered under Class A. Vertical breaks appeared in the top of the pile, these continuing roughly half way down and sloping outward toward the bottom. The bank of sawdust gave the writer excellent illustrations of the type of conchoidal fracture that Mr. E. G. Haines

recognized in earth excavations and discussed before the American Society of Civil Engineers 12 years ago. Even this loose sawdust, possessing so little cohesion and having been compacted only by its own weight over a period of a few months, shows clearly the difference in slopes assumed by any particular material piled loosely and the same material failing in the bank of an excavation.

This point has an important bearing on underpinning problems and all other problems involving the determination of the safety of structures adjacent to excavations. To determine the safety of a structure adjacent to an excavation, it is necessary to develop the boundaries of the material that will be affected by the failure of the banks of the excavation. Since the behavior of material in a bank is different from that of the same material piled loosely, considering of course only materials that in the bank possessed a certain amount of cohesion, it is evident that the area of disturbance cannot be determined by plotting slopes obtained by the study of the material when loosely piled. The use of a slope or angle of repose based upon the study of loose material will lead to erroneous results. As a matter of fact, there is probably a small area beneath such a line in which the ground will be disturbed and a much larger area above the line in which the ground will not be disturbed. We are much nearer a solution of underpinning problems, when considering Class A material, if we realize that the line of fracture is made up of practically a vertical line extending approximately half the depth of the excavation and thence curving outward to the bottom than we are if we assume it as a uniform slope determined by the study of the loose material.

While a proper formula of earth pressures is yet to be developed, at least this much is apparent: that the existing formulas that make use of an angle of repose and result in an estimate of pressure greater at the bottom of a cut do not hold for earth materials in Class A. The ideas developed by Mr. Meem and Mr. Haines are too important to be forgotten by the profession, and while this paper has not added to their work much of material importance, or anything new except perhaps the compilation of a series of comparisons of earth and rock under failure, its purpose will be served if the subject is again brought up for discussion and review by mining and civil engineers and the work of these two men given further consideration and amplification.

## DISCUSSION

E. G. Haines, Brooklyn, N. Y. (written discussion*). —When 12 years ago I made the statement that, carried to a sufficient depth, a cut in solid rock would fail and behave in precisely the same manner as one in earth, I was taking a shot at a target I could but dimly see, but the

* Received Feb. 18, 1920.

statement was made in good faith and I am very glad that Mr. Moulton has found the results so close to the prediction. Frankly, I should have been much surprised had he found them otherwise, for the statement was made after long observation of the manner of failure of earth, under widely differing conditions, as a result of which I was firmly convinced that the commonly accepted methods of determining earth pressures were erroneous.

Pressures are usually determined on the basis of Coulomb's formula, or some modification of the hydrostatic formula, where factors for the friction of a sliding mass are introduced. This requires the assumption of a sliding plane, or angle of repose. As a matter of fact, in an undisturbed bank of earth there is no such thing as an angle of repose. Even in materials that break into a fine granular mass and will stand only at very flat slopes when piled, an excavation can often be carried down vertically for some little distance before failure occurs, even though the material has been piled but a short time.

When an excavation is made through most classes of earth, it can always be carried to a certain depth before the bank fails. When failure occurs, it is not at the angle of repose of the removed material. Instead, it shows a curved line, usually closely approximating a circular curve in the lower portion; the upper portion often is vertical and tangent to the curve of the lower portion. In a long trench carried to uniform depth, there will usually be a number of short slides, the lengths seldom exceeding very much the depth of the excavation. Such slides, or failures, are purely local matters. They may be due to local weakness but are more likely to be an indication of the fact that the excavation has reached the critical depth and is on the verge of failure. The distance back from the excavation to the upper edge of the failure (or the crack, in case the excavation is timbered) usually approximates one-half the depth of the excavation at the time of failure.

Mr. Moulton has given me credit for bringing these matters to the attention of engineers. I can hardly claim that honor. The relation between height and width of bank failures was mentioned by the late Sir Benjamin Baker, in 1880, in a paper[3] before the Institution of Civil Engineers in connection with the excavation for the London underground railroads. He was, however, unable to describe a reason for that relation, but did point out that it was inconsistent with the usual theory and with the angle of repose for the particular material encountered. The curve, which is a noticeable feature in the fracture of earth banks, was also mentioned by Mr. Wilfrid Airy in the discussion of that paper. My attention has recently been called to a paper[4] presented by Mr. Airy, in 1878, before the Institution of Civil Engineers, in which he pointed

[3] Actual Lateral Pressure of Earthwork. *Proc.* Inst. Civ. Engrs. (1880) **65**, 140.
[4] On the Slopes of Cuttings. *Proc.* Inst. Civ. Engrs. (1878) **55**, 241.

out that excavation could be economized and safety insured by dressing the banks to a curve tangent to the vertical at the top, and tangent to the angle of repose at the bottom, instead of extending the line of the angle of repose to the top of the bank. Neither of these men, however, advanced any reason for the relation between height and width, or the curved form of the fracture, both of which are of the highest importance and give an indication of the cause of failure.

In my discussion of Mr. Meem's paper before the American Society of Civil Engineers, I presented an explanation of both these features and stated what I believed to be a simple natural law governing both the form of the fracture and its location which is essentially as follows:

In the case of earth or other homogeneous granular masses, failure takes place, to state the matter in a very elementary way, simply because the force exceeds the resistance. The force, in the case of an earth bank, is composed of the sum of the weights of the individual grains of the material acting vertically downward; there is no other active element. The resistance may be called cohesion, adhesion, shear, or anything one chooses; I use the term shear, which it most closely resembles in its action. The entire resistance equals the sum of the resistances of the individual grains over the entire surface that fractures. It may also be stated, as an elementary axiom, that failure will take place at the instant when the sum of the forces exceeds the sum of the resistance units. It may further be stated that failure will occur in that form which allows the greatest possible weight to be opposed to the smallest possible resistance. From geometry, we know that this condition is satisfied only when the weight is in the form of a sphere. This condition at once accounts for the relation of height to depth and for the curved form of the fracture in the earth, as the fracture is hemispherical in form. The fact that the upper part of the fracture is often vertical is due simply to the fact that the tangent from the curve to the surface of the earth is shorter than the arc in the upper quadrant, and forms therefore a path of least resistance. The volume of a sphere varies as the cube of its diameter, whereas its surface varies only as the square; it is therefore evident that at some depth failure must occur. This applies as well to a hard granular material, such as rock, as it does to moist sand.

At the time this theory was propounded, to test its application, a number of cases and conditions were cited, which it is unnecessary to repeat here. I would state, however, that in the past 12 years I have had an opportunity to verify those statements on construction work involving the removal of several million yards of excavation in shafts, tunnels, and subways and see no reason to modify any of the statements I at that time made. On the contrary, I have every reason to believe that they have been confirmed.

While heretofore neglected, the position and form of the fracture

curve is of the greatest importance in connection with construction work, especially work similar to the subways, where numerous large buildings and other structures are adjacent. Mr. Moulton has mentioned this fact in its relation to underpinning of buildings. In the past, the safety of a building adjacent to excavation has been determined largely by the position of its foundations with reference to an assumed angle of repose from the bottom of the proposed excavation. This line has often been taken as one vertical to one horizontal, and in some doubtful cases as one vertical to two horizontal. On neither of these lines can it be determined whether or not a building will be disturbed by an adjacent excavation On both lines, it is possible that a building may be either safe or unsafe, depending on the depth of its foundation below the surface and the distance from the excavation. In the case of a 2 to 1 slope, for instance, a building with a shallow foundation, which naturally places it farther from the excavation, will probably be safe; whereas if it stands closer to the excavation, say at a distance one-half the depth of excavation, it may or may not be safe; while if close to the excavation it is most certainly unsafe. This condition is due to the fact that the curve of earth fracture is, theoretically and usually in fact, nearly horizontal at the bottom of the excavation.

During the past 20 years, in connection with subway construction, I have investigated the conditions affecting the stability of several thousand buildings adjacent to the excavation. A large percentage of these have been underpinned by various methods with varying degrees of success. The influence of the fracture line on earth banks has been most marked in connection with this work and I am convinced that the determination of the degrees of safety based on any assumed angle of repose is an improper and dangerous method. In a number of cases buildings, which would probably have been safe without underpinning, have been badly cracked and injured, after underpinning, as excavation proceeded. The underpinning piers added weight to the bank near its critical line and the excavation for the piers weakened the bank at the same line, both of which had a tendency to cause the bank to shear at a greater distance from the excavation than would otherwise have been the case.

Certain of the illustrations given in Mr. Moulton's paper referred to deep mining operations, in which the overlying material is described as limestone or granite. One would appear to indicate that a considerable amount of earth was over the rock. One of the objections stated against the theory I have proposed for soil failure is that it would not apply to stratified material. I wish to take exception to that statement. As stated at the time it was proposed, the theory applies to any and all granular materials. The fact that the material is stratified does not materially influence the problem. Failure is due simply to the summation of weights and resistance; and while the height and width of the

fractured volume will be controlled by the predominant material, earth weighs approximately two-thirds as much as rock and where they occur together is usually much less in amount. The condition of maximum force and minimum surface will still approximate a spherical fracture.

Robert Ridgway, New York, N. Y. (written discussion*).—It is clearly pointed out in the paper that the author is only discussing the behavior of materials, rock and earth, that act as a solid, called by him "Class A" material. This must be borne in mind by those who study the paper and the slopes or lines of fracture under discussion must not be confused with those assumed by the same material after a long exposure to erosion and to the weather. The flattened slopes resulting from long exposure approximate the angle of repose and are due to entirely different causes than the pressures discussed in the paper. We know that steam shovels naturally excavate in Class A earth to slopes that approximate the author's line of rupture and that these slopes stand for long periods of time without caving, generally until erosion or weathering flattens them.

One frequently sees a timbered trench in which the timbering of the upper half indicates stress while the lower timbers do not. Recently, a trench 55 ft. deep and about 40 ft. wide was timbered in the usual way. After several months, the timbering of the upper portion showed increasing signs of stress in spite of the fact that the steel structure had been erected in the trench and the side pressures transferred to it for a height of about 30 ft., leaving approximately 25 ft. of the upper portion supported solely by the timbering. The dividers, or cross-timbers, were bowed under the pressure and were biting into the rangers and the sheeting was badly bowed between the wales. The support given by the steel structure in the lower 30 ft. did not appreciably reduce the pressure on the timbering in the upper 25 ft. so it was necessary to place additional dividers there to prevent a possible collapse.

Over 30 years ago, the speaker was connected as assistant engineer with the construction of what was then called the New Croton Gate House at the intake to the New Croton Aqueduct. This structure required the excavation of about 125,000 cu. yd. of material, nearly all of which was rock, the stripping being relatively light; the location was in a steep side hill. A deep recess had to be excavated in the hill at the level of the existing aqueduct about elevation 174 and in this recess a pit about 40 ft. deep was sunk, in which the massive gate chambers were built. The bottom of this pit was at elevation 134 or a little lower, the floor of the gate chambers at elevation 210 and the ground surface at the back or deepest part of the cut was about elevation 345. In other words the greatest depth of the cut (elevation 345–134) was about 211 ft.

* Received Feb. 18, 1920.

These figures are given from memory as the records are not available at this time, to the speaker, but they are approximately correct.

According to the contract drawings, the excavation in rock was to have been made with vertical slopes below elevation 270 to the bottom, a distance of 136 ft. At elevation 270, a berm was planned about 10 ft. in width with rock slopes above it of eight vertical to one horizontal. The work had not proceeded very far before it was found impracticable to maintain such steep slopes, so they were flattened from time to time until when the excavation was completed to elevation 134 the rock slope from top to bottom averaged about two to one. The gate chambers were then built, affording masonry or backfilling support to the sides of the cut up to elevation 210, leaving unsupported the remainder of the cut with a maximum height of about 135 ft. The slopes above elevation 210 have been exposed to the weather for over 30 years; and while it has been necessary from time to time to trim the small masses of rock loosened by frost and other causes, there has been, so far as the speaker knows, no trouble from any large movements of the rock. Had it been possible to excavate and maintain the cut during construction to the original steep slope lines the speaker is of the opinion that there would later have been very extensive movement for the reasons given by Mr. Moulton.

J. C. Meem, Brooklyn, N. Y. (written discussion*).—There are three points in which the writer differs from the author. The first point is that the breaking line, or plane of rupture, of the ground is not in any way connected with or dependent on the plane or angle of repose. The second is "that formulas for bank pressure should not include any functions of the angle of repose. A certain type of earth may have an angle of repose approximating 45°, whereas the angle of repose of hard rock, except for the influence of local seams and planes of weakness, is 90° from the horizontal; yet the behavior of each is identical, etc." The third point is the classification, which is not sufficiently comprehensive.

The writer, in a paper read before the Brooklyn Engineers' Club, made five instead of three classifications, as follows:

Class A, cohesive material, such as solid rock, hard clay, etc.

Class B, semicohesive material: normally dry granular materials; *i.e.*, all soils not included in the other four classes.

Class C, noncohesive material, such as bone-dry hot sand, grain, etc.

Class D, semi-aqueous material: all submerged or supersaturated soils not included in class E; that is, all soils, such as sand, gravel, etc.

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* Received Feb. 18, 1920.

the characteristics of which are not changed by submergence, as in the case of clays.

Class E, aqueous material, such as pure water or other liquids, plastic clay, pure quicksand, grout, etc.

Class E is eliminated from this discussion because the laws of aqueous or hydrostatic pressure are well known and need not be considered here, except that the fact should always be kept in mind that the pressure is due to that of the specific gravity of the basic liquid and not the weight per volume. Class A is eliminated, except where operations are on a scale sufficiently large to bring the material into class B, as noted later. Class C is eliminated except in the notes on experiments later on.

Coming now to the first point of difference—the relation between the breaking line and the angle of repose. It should be borne in mind that the break that appears above the tunnel or back of the cut may not fairly represent the condition of stress or pressure that existed before the disturbance or break occurred. The function of the engineer is to brace against the possible break, and the stresses developed by his timber or structure are very different from those measured by the area of the mass affected if his structure fails, or if he purposely removes the support. The fact that the break occurs along cleavage lines, or conchoidally or in some other manner, establishes the fact, rather than denies it, that there is a plane of repose which may be approximated to if not found to be a straight line. For instance, between the breaking lines or planes above a tunnel is included a large mass of material whose weight does not in any way stress the tunnel structure. Even during the gradual subsidence of the material a large part of its weight is taken on the so-called haunches corresponding to the planes of repose as noted later; that is, unless the width of the structure is relatively greater than the thickness of cover (or key), in which case the natural arch fails and virtually the entire weight comes on the structure or the timbering.

This leaves for final consideration the second point, that formulas based on angles of repose fail because that of solid rock is assumed to be 90°. It is unfortunate that the author has fallen into this error, since further on he states that under large operations the angle of repose of rock is equivalent to that of boulders or other aggregate. This being true, why or how can it be 90° under small operations? As a matter of fact, in such instances its angle of repose is not affected and should not be considered unless affected. In other words, the angle at which rock will break (when it breaks with everything clear) should be found and that only established as the angle of repose.

In this connection the writer wishes to explode one other fallacy of those engineers who approach somewhat gingerly the angle of repose; that is, that the angle of repose of water is zero, which is no more true than that that of rock is 90°. The fair definition for an aggregate having

an angle of repose of zero is that it must give full pressure on a vertical wall and none on a tunnel roof; while an aggregate having an angle of repose of 90° must give full pressure on the horizontal roof of a tunnel and none on the vertical face of a wall.

While these conditions do not exist in nature, the writer believes they are approximated in the case of bone-dry sand or grain in one case and hard clay with lubricated vertical cleavage planes in the other. Thus, the clay with its angle of repose of 90° would give full pressure on a tunnel and very little against a wall; whereas the hot dry sand, with its angle of 33° ±, would give the maximum pressure on a wall and minimum pressure on a tunnel roof. Theoretically, these conditions can be met by

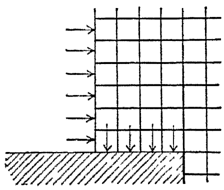

FIG 15

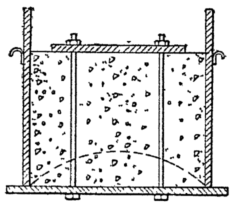

FIG. 17.

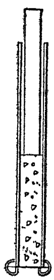

FIG. 16.

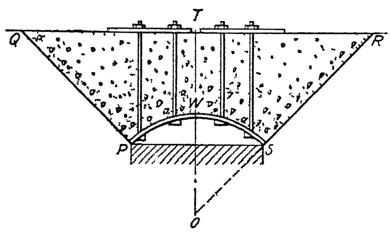


FIG. 18.

assuming a mass of iron filings drawn horizontally by a powerful magnet, as representing an angle of repose of zero and cubes of frictionless marble superimposed vertically as representing an angle of repose of 90°, as shown in Fig. 15.

The broad question with which the engineer is concerned, however, is: how much stress must be calculated for and met in the design of the timbering, the structure, or the operation. Manifestly something is fundamentally wrong with the old formulas and the ordinary course is to use them and allow such factors as are dictated by expediency and experience. In this discussion the writer approaches the problem not with the confidence of solution, but in the hope that his suggestions may help pave the way, through experiment on a large scale and through the com-

pilation of such data as are set forth in this paper and otherwise, to a final solution of these vexed problems.

His conclusions are based on these fundamentals: (1) That pressure is transmitted laterally through soil; (2) that the tendency of all soil is to arch due to this lateral transmission of pressure; (3) that pressure through soil is not normally cumulative; that is, a 10-ft. square tunnel would have no more stress on its roof (in homogeneous soil free from water) at a depth of 1000 ft. than it would at a depth of 100 ft. or 20 ft.

A few years ago the writer showed, before the American Society of Civil Engineers, two experiments; one was suggested in an old English book and the other was original. By fastening a sheet of tissue paper across one end, he converted a 2-in pipe 24 in. long into a container, which he filled, for about 10 in., with hot dry sand. A loosely fitting plug, to act as a piston bearing on the sand, was then put in as shown in Fig. 16. It was found that ordinary pressure, such as the weight of a man, or the fairly delivered blow of a sledge hammer failed to break the paper at the bottom. The pressure on the sand is distributed laterally against the sides and as the friction increases in proportion to the pressure, the stress cannot be transmitted vertically unless the pressure is sufficient to solidify the sand, in which case the pipe would probably burst before the sand yielded. The writer believes that this principle applies to grain bins and other similar structures, however large. He also believes that this principle accounts for the fact that a hollow steel pile, when being driven, will pick up a sand plug which prevents farther progress of the pile (unless cleaned out), or else it will be carried down with the pile, leaving the upper chamber empty.

In the second experiment, the bottom of a small box was replaced by a bottom somewhat larger than the box, as shown in Fig. 17, bolts bearing on large washers above passed through it. The box was partly filled with hot dry sand, as shown, and the nuts keyed down hard on the washers on the sand. When all was tightened in place, the box could be lifted and carried about without danger of collapse. The writer has made this experiment with success on a box equivalent to a 3-ft. cube and does not doubt that the principle would apply to boxes of an indefinitely larger size as well. When sufficiently tight, the box may be filled with water and carried about indefinitely, as long as there is not enough leakage to discharge soil with the water.

In the case of the box of dry sand, it is apparent that all the conditions of the arch are fulfilled. The third condition, cohesion, in the arch members is usually lost sight of because arches are composed of members of solid material and internal cohesion is implied. In the case of the sand arch, this cohesion is supplied by confining the material below and externally, thereby causing the sand arch to supply its own centering. If

this experiment can be duplicated on a large scale, there is no reason to doubt that centering can be introduced above a tunnel and by supporting it, by means of long bolts keyed to washers on the surface, virtually relieve the tunnel of any load on its roof, except that below the centering *PWS*, as shown in Fig. 18. In practice it is impracticable to do this, but it is not necessary, for when the so-called centering is properly maintained above the tunnel roof the soil arch above relieves the tunnel roof of nearly all or a large part of the rest of the load.

The same principle applies if the tunnel is eliminated and a wall erected along the line *OWT*; the soil above the half centering *WS* will be carried by the arch between *TW* and RS. This may be shown more clearly in Fig. 19. In this it is assumed that the soil arches between the

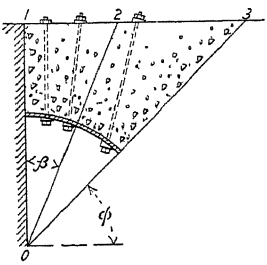


FIG. 19.

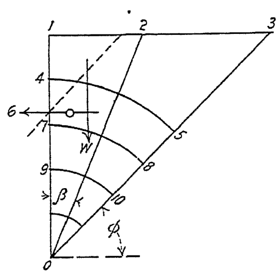


FIG. 20.

planes 1–0 and 3–0 (the last being the assumed plane of repose) and that some line or plane, as 0–2, marks the division between that carried by the wall and that by the plane of repose.

In Fig. 20, the solid wedge 1–0–3, which slides freely on a frictionless plane 0–3 but is held in place by a retaining wall 1–0, is the equivalent of a series of solid arches 4–5, 7–8, 9–10, etc. Some plane, as 0–2, will divide the two areas, so that one will result in a thrust against the wall and the other as a weight on the plane 0–3. The writer has arbitrarily assumed that this line 0–2 will bisect the angle 1–0–3; that is, the angle 1–0–2, or $\beta$, will be $\frac{90° - \phi}{2}$. The weight of the material accounted for will be all that in the area 1–0–2. The thrust will be the weight divided by the tangent $\phi$, and the lever arm will be the height of the center of gravity of the area 1–0–2 above the tangent, or $\frac{2}{3}H$.

With the exception of this last member (the lever arm), the formula for a sliding aggregate will be the same as for the sliding solid and need not therefore be further developed.

In the case of the sliding aggregate, which corresponds to Class B, or semicohesive, the aggregates are assumed to slide along planes 3–2, 5–4, 7–6, etc. (parallel to the plane of repose), Fig. 21, until they have

developed sufficient depth and stability to arch along the assumed lines 4–19, 6–20, etc., when some plane, as 0–7, will mark the division between the thrust area against the wall and the area supported entirely by the plane of repose 0–18. The writer continues to assume that this area of thrust is measured by the angle $\beta$, or $\frac{90° - \phi}{2}$. The thrust of the soil being downward instead of out, as in Fig. 20, the center of pressure will probably be at 6, where the line through the center of gravity at $A$ parallel to the slope line meets the vertical.

If $H$ is the height 0–1, the horizontal distance (radial) of $A$ from the wall is $\frac{H \tan\beta}{3}$ and the vertical distance of $A$ above 6 (tangential) is $\frac{H \tan\beta}{3} \times \tan\phi$.

Therefore if $R$ is the lever arm,

$$R = \frac{2}{3}H - \frac{H \tan\beta \tan\phi}{3}$$

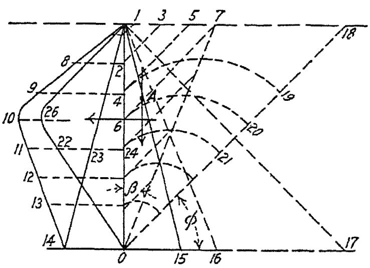

FIG. 21.

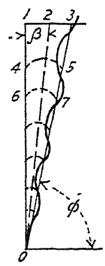

FIG. 22.

If $\Sigma$ represents the relation to the height $H$ (noted in the sliding solid as $\frac{2}{3}$),

$$R = \Sigma H = \tfrac{2}{3}H - \frac{H \tan\beta \tan\phi}{3}$$

or

$$\Sigma = \tfrac{2}{3} - \frac{\tan\beta \tan\phi}{3}$$

In the case of $\phi = 45°$, $\Sigma = 0.53$.

Let $W$ = weight of soil per cubic foot (volume).

$$\text{Area (volume)} = A = H \tan\beta \times \frac{H}{2} = \frac{H^2 \tan\beta}{2}$$

$$\text{Thrust} = \frac{W}{\tan\phi} = T \text{ and } R = \Sigma H.$$

$$\text{Then } M = ATR = \frac{H^2 \tan\beta}{2} \times \frac{W}{\tan\phi} \times \Sigma H = \Sigma \frac{WH^3 \tan\beta}{2 \tan\phi}$$

If the ordinates of the pressure lines 3–2, 5–4, etc., are plotted the pressure curve 1–26–0 can be drawn to the left of the wall 1–0 in Fig. 21.

Fig. 22 probably represents more nearly conditions the engineer will normally meet, in which the angle of repose is steep, irregular and poorly defined, and in which the surface break will probably show at 2 rather than 3, more certainly if the ground is hard and the trench well braced.

In the matter of pressures on tunnels, the writer assumes that all the soil in the area 5–4–6, Fig. 23, is carried by the tunnel roof 5–15–6, while all the soil above the area 5–3–6 is carried by the normal arching of the soil. Some mean area between these two marks the dividing line that separates the soil stressing the tunnel roof from that which is supported by the normal arching of the soil. For safety, the writer assumes that

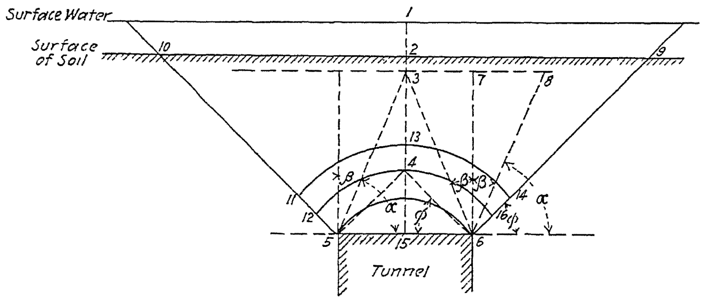


FIG. 23

the planes 5–3–6 measure the area of the soil whose weight is carried by the tunnel roof. In this $\alpha$ is taken to equal $90° - \beta$ where $\beta = \frac{90° - \phi}{2}$ and the area 5–3–6 $= A = \frac{L}{2} \times \tan \alpha \times \frac{L}{2} = A = \frac{L^2}{4} \tan \alpha$. The pressure on the roof per linear foot is

$$P = \frac{L^2}{4} \tan \alpha W = \frac{WL^2}{4} \tan \alpha$$

where $P$ = pressure on roof, per linear foot;
$L$ = width of roof or span;
and $W$ = weight of soil per cubic foot (volume).

In practice the area below the line 12–4–16 will more probably represent the actual pressure, with all below 11–13–14 representing the safe maximum. In his practice the writer has always used the formula just given.

When the depth 2–15 is less than the span $L$, the full vertical pressure of the soil should be assumed as coming on the tunnel roof.

In these instances of pressures of semicohesive soils on trench and tunnel, it should be noted that the flatter, or smaller, the angle of repose the greater is the pressure on the trench or wall and the less is the pressure on a tunnel roof; while with a steeper or larger angle, the reverse condi-

tions prevail. In all cases, therefore, in which the angle of repose cannot be determined or even fairly estimated the engineer should use the maximum (probable) in designing a tunnel and the minimum (probable) in a wall design.

In those cases where the ground is supersaturated, *i.e.*, Class D or semi-aqueous soils, it should be noted that the pressures, though combined, act in separate units. That is, the status of the soil is not affected by the pressure of the water. It should also be noted that this does not apply to what may be called aqueous soils, such as those containing a larger percentage of clay or other lubricating soil—such as pure quicksand, plastic clay—but it does apply to all soils in which the granular element predominates—such as sand, gravel—under normal conditions and not under velocity head due to flow of water, etc. The only effect of submergence on these soils is to change the cohesive factor and the possible angle of repose.

It is not possible to have full pressure of water on the whole area of a structure and at the same time have full soil pressure on the same area. For instance, if we assume that soil presses on the roof of a structure and through imaginary columns, this pressure is extended vertically so the water cannot also exert pressure on this area. If those columns have not been affected by the water and extend sufficiently high to exert their normal arching properties, we have in the case of a tunnel (with the same notation as before), if $D = 1.15$, or depth of water; $d = 2.15$, or, depth of soil (where $d$ is greater than 5.6); $PC$ = combined pressure, and $V$ = weight per cubic foot (volume) of water

$$PC = \frac{WL^2}{4} \tan \alpha + SLDV$$

Where $S$ = percentage of effective voids in the soil; for safety it may be taken as 50 per cent. or 0.5. Where $d$ is less than $L$,

$$PC = dLW + (DLV - dLV) = dLW + LV(D - d)$$

When the engineer has reason to fear that full hydrostatic pressure may result and the weight $PC$ as deduced is less than $Pw$ (the full pressure of the water disregarding the soil) the structure should be designed to resist stress as indicated by $PC = DLV$. But at no time is the engineer justified in taking full soil or full water pressure combined.

In the case of combined pressure against a wall, Fig. 21, if the curve of normally dry pressure is assumed to be 1–26–0, we may produce the curve 1–10–14 by assuming 1–17 as the line of full normal water pressure; 1–16 will be that due to the reduction of area by reason of the voids and 1–15 will represent the correlated pressure due to the difference between the weight of soil (100 lb.) and water (62.5 lb.). Drawing 1–14 equivalent to 1–15 and extending the ordinates between 1–0 and 1–14, beyond

1–26–0, the curve 1–10–14 is drawn, representing the combined pressure. Thus we have 1–14 representing the pressure due to water, 1–26–0 the pressure due to soil, and 1–9–10–11–14 the combined pressure.

The radius of gyration, or lever arm of the moment of this combined pressure, is a proportionate mean easily found between 0.53 $H$, due to soil, and 0.33 $H$, due to water.

The writer does not assume that he has solved, finally and conclusively, all or even any of the vexed problems noted; but he does hope that he has shown progress along the road leading thereto.

Until experiments are made on a large scale, no final or definite solution may be expected. Nothing can be gained by punching holes in sheeted trenches or cofferdams and measuring the pressure thereat. One might as well expect to measure the weight of sand in an hour glass by determining that at the orifice, for no matter how large or small the volume of the hour glass, on equal orifices the pressure is always the same.

In conclusion the writer wishes to say that while pressures can be made cumulative by deepening a cut or enlarging (either vertically or horizontally) a tunnel prism, they are not normally cumulative in ground. That is, tunnels of equal cross-section in normally dry soil will develop the same pressure at whatever depth they may be placed. Tunneling operations would not be possible in normally dry soils except with closed full shields, nor could operations be carried on at the bottom of a deep cut if pressures were cumulative. Pits of small cross-section (4 or 5 ft. square) may be sunk indefinitely in normally dry soil without increasing the cross-section of the sheeting, due to the fact that the horizontal arching of the soil prevents the pressure from becoming cumulative, as it would do in a long cut.

Fig. 24 shows the operation of jacking up sections 4 ft. by 10 ft. of the roof of the Battery tunnel (New York City) under Joralemon St. Upwards of 160 ft. of tunnel roof was jacked up 30 in., allowing a 16-in. ring of brick to be built and giving an additional clearance of 14 in. This could not have been possible had the theory of cumulative pressures prevailed. It may also be of interest to note that the bottom of the Battery tunnel was cut out for considerable distances and replaced by one at a lower gradient. When this work was done above the water line there was no evidence on the exposed bottom of pressure.

Fig. 25 shows a sectional shield emerging into a shaft. The sectional shield is designed to be used at any depth without change of cross-section, as long as water is not present under pressure. The theory of its design is that the pressure per square foot is proportionately less on small areas aggregating a total large area than it is on the same large area.

This discussion is submitted in confirmation, or corroboration, of the author's belief that there is great need of revision of the theories and formulas now in use. These theories and formulas are still taught in

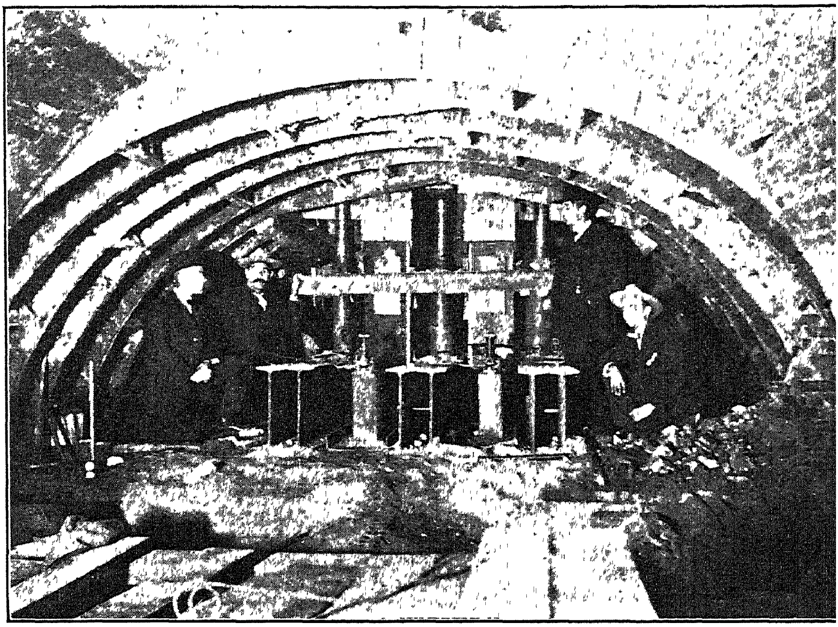

FIG. 24.—JACKING UP TUNNEL ROOF.

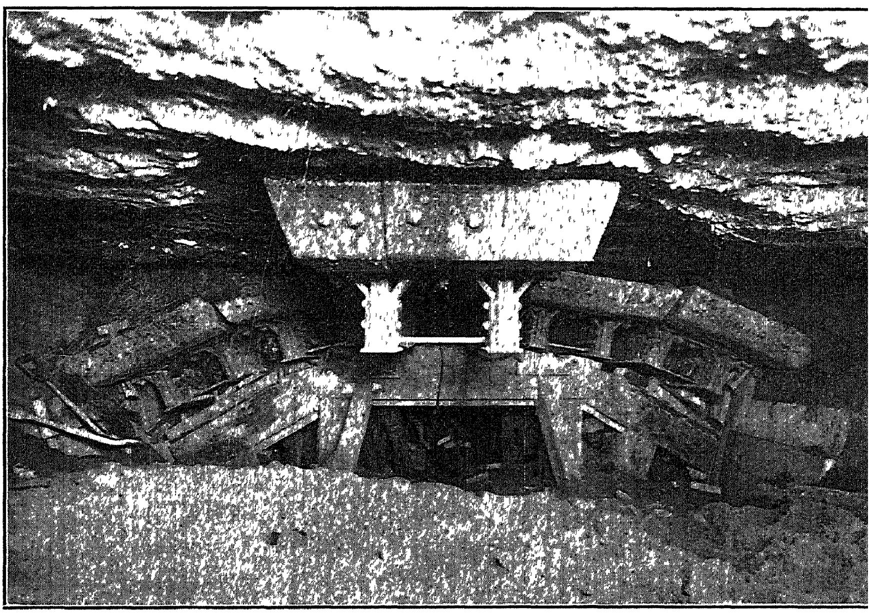

FIG. 25.—SECTIONAL SHIELD EMERGING INTO A SHAFT.

the schools and, unfortunately, are used by many engineers, though the results by them are usually modified and safeguarded by factors of safety.

HARRISON SOUDER,* Cornwall, Pa.—Many engineers have been studying the matter of slopes, in connection with large open-cut mining operations and it has been difficult to decide what slopes rock would take and what slopes should be allowed in mining iron ore in such open cuts. A practical application of Mr. Moulton's theory can be made in the design of ore pockets or bins.

We have some sticky ore that packs in the bins and is very tenacious but the lump ore flows freely. The sticky ore assumes practically the same slope Mr. Moulton brings out in his paper. It seems to me that there is a possibility of making some saving in designing ore pockets, knowing that sticky ore is to be put into them, a slope of 70° would answer the purpose.

B. F. TILLSON, Franklin, N. J.—If a small drift or narrow stope is driven in an orebody that has a tendency to slough off in slabs, due to strain of former geological conditions, or due to the intensely granular nature of the ore and the weight above it, we will find a tendency for the roof to arch until it reaches a certain form in which it maintains a condition of equilibrium. Now, what is the relation of that excavation, for certain classes of material, to its distance below the surface in order that the arching effect and the voids that come with the change of material from a solid to a broken state will cause no disturbance of surface conditions?

If we have a foot wall and a hanging wall and are caving beneath the surface, the solid being mined is covered with a so-called matting of timber, brush, etc. We start mining in slices beneath that matting with a burden of broken rock and fill placed there to provide a large volume of material to distribute the stresses and shock produced by the fall of any large masses from the hanging wall, or from the solid ground over a lens that does not outcrop; and we come back in certain steps. A certain number of sets are left wide for convenience in operation. As we mine and leave more sets of timber open, the stresses increase in the timbers, their failure becomes quite serious, and the danger of operation increases. But if by shooting out the posts in those sets we diminish the extent of open chamber until, say, two or three sets are left open to solid pillars, those stresses are such as permit the timber to support the matting regardless of the depth of the overburden. In the operation of the caving system, the production per man in any working place is more limited than in other systems of mining and the natural result is to gain from a certain size of orebody a maximum production by

* General Manager, Cornwall Ore Bank Co.

bringing back in any one pillar a number of these steps, thereby permitting a multiple unit of gangs that can work in any one pillar.

The distance between retreating sublevels or slices is of high importance. If one face approaches very close to the other, there is a greatly increased strain in the supporting timbers in the matting. Is that strain due simply to the increased length or span of the arch formed in the filling material and in the matting that helps to hold it in place?

When we have sought to replace timber sets with steel sets, in shrinkage stoping, in which we are mining a pillar of material by back stoping (working on top of the broken ore and drawing off the excess with a solid back above and a pile of broken ore below) we have found that, as we go up a considerable distance, we do not get increased breakage of the timber because of the arching effect between the sides of that stope and this broken ore. But as we increase the span of that stope we get increased pressures. If we had more definite figures it would be easier to design steel framework to replace timber to withstand such stresses. We have been fortunate in designing steel sets that replaced timber sets and satisfactorily supported many thousands of tons of broken ore without getting into excessive steel weight. These steel sets support very heavy ore for indefinite lifts and have been previously described.[5]

C. E. Arnold, New York, N. Y.—Concerning the work of the Miami Copper Co. at Miami, Ariz., I should like to say that this company has made what seems to me to be an interesting application of the principles that Mr. Moulton formulates. My recollection is that in the earlier stages of its caving operations that company carried the shrinkage stopes almost to the top of the ore, assuming this to be necessary for obtaining the proper crushing action on the pillars between the stopes. When the pillars started to crush, it was noticed that they tipped sidewise toward the open pit resulting from the ore extraction. Later it was found that the stopes could be carried to approximately half their original height with equally good crushing action, the conclusion being that the natural lateral pressure developed was sufficient to cause the crushing.

Mr. Moulton has sketched for us his conception of the relative values of the horizontal components of this lateral rock pressure, and it is noticeable that of these components the greatest are at approximately the elevation that corresponds to the center line of the ore zone which the Miami Copper Co. found unnecessary to stope. This appears to me to be good proof that Mr. Moulton's theory is sound, and that it offers broad possibilities in its commercial application.

Waldo C. Briggs, Brooklyn, N. Y. (written discussion*).—Recently a trench at the street level was excavated for the purpose of laying a bank

[5] Zinc Mining at Franklin, N. J. *Trans.* (1917) **57**, 720.

* Received Feb. 14, 1920.

of ducts to be used in connection with the operation of the tunnel. At four points along the line of this trench, which was 10 ft. deep, an intersection with the line of "decided break" occurred. In the four intersections, the break was decidedly evident and extended vertically to the bottom of the trench. This would seem to confirm the opinion of the author of the paper in regard to the form of the curve of rupture.

In the first sentence on page 348, the author says: "The pressure on the timbers reaches a maximum at or slightly above a point halfway between the top and the bottom . . . ." This statement was confirmed by observations made in one of the shafts built by the Degnon Contracting Co., referred to on page 330. This shaft was about 22 ft. by 40 ft. in section and 100 ft. deep in a mixture of sand and clay and stood for over 2 yr. before the permanent lining was placed. During this time the timber lining deteriorated materially and heavy loads were developed; so that the cross-braces cut deeply into the "soldiers" (vertical rangers placed against the horizontal sheeting). The indications of approaching failure were all above the midway point in depth and greatest at a point about 20 ft. in depth.

H. G. Moulton.—The question as to the critical depth at which the surface would not be affected by a given size of tunnel is one that cannot be determined until much more data are collected and some of the underlying principles governing the failure of rock at great depth can be developed; but, generally speaking, as the widths of a stope increase, there comes a point at which the lines of arching action are broken and the collapse extends to the surface. Somewhere between the small tunnel, which, as Mr. Meem points out, is a simple case of rock arches, and the shrinkage stope that breaks through to the surface, there is a critical relation between width and depth, dependent also on the strength of the rock. I hope that by the collection of all possible data, with the coöperation of the members of the Institute, it may be possible to determine some of these matters with a reasonable approach to accuracy.

As to the design of timber or steel sets over a main haulage way in filled ground, the basis of assumption is to strike the probable lines of arching action through the broken rock and figure that the load on the sets is only that coming from the rock under the arches. If the posts are on solid rock, the arches in the broken rock may take all of the load except for the material that lies beneath the arch whose lines touch the corners of the set. If the set rests on filled ground, and if for this reason or because of irregularities in the sides of the stope the arches are higher, it might be advisable to assume that the weight coming upon the caps is due to the material lying under the arch whose abutments are on a level

with the post footings, or even higher. In such a case it would be advisable, after making an assumption as to the position and shape of the lowest arch line which might be expected to relieve pressures, to draw lines upward from the position of the ends of the cap at a slope of ½ to 1 to intersect the arch line, and assume that the weight coming on the cap was that bounded by these lines and the arch line. Until more data are available, it would be necessary to make so many assumptions that the calculations would be of little value, but I believe that the principles along which we should work in developing a method of calculating pressures on sets in filled ground should be as outlined.

# Examination of Ores and Metals in Polarized Light

BY FRED. E. WRIGHT,* WASHINGTON, D. C.

(New York Meeting, February, 1920)

IN A recent paper[1] a detailed discussion is given of the possibilities of using polarized light in the examination of opaque substances. The factors underlying the problem are there treated from the viewpoint of the electromagnetic theory of light and the presentation is necessarily mathematical in form. The practical methods resulting from the discussion are, however, relatively simple and promise to be of value in the determination of certain ores and metals when studied in reflected light under the microscope.

The results of the theoretical investigation referred to may be summarized by stating that in general the optical constants, such as refractive indices and absorption indices, cannot be satisfactorily ascertained on small, random, polished sections; that the use of polarized light enables the observer ordinarily to determine whether the crystal plate is isotropic or anisotropic, and also to ascertain the degree of anisotropism; that for this purpose methods based either on the contrast in intensity between the two reflected components of non-polarized light, or on the amount of rotation of the plane of polarization, on reflection, of vertically incident, plane-polarized light, may be employed; that methods based on the phase difference between the two reflected components are in general of little value because of the small differences in phase which ordinarily result for a relatively large change in birefringence or biabsorption.

The measurement of the optical properties of transparent minerals, even in minute irregular grains, is a simple task with modern petrographic microscope methods and is accomplished by petrologists as part of their ordinary routine work. But the determination of the optical constants of opaque substances is difficult and is rarely attempted; all observations are necessarily made in reflected light and are restricted commonly to the determination of color, of hardness, of the character of crystallization, and of the behavior of the mineral or metal plate toward reagents and abrasives. With practice, the observer becomes expert in recognizing details of this kind and determines with a high degree of certainty the different

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* Petrologist, Geophysical Laboratory.

[1] F. E. Wright: Polarized Light in the Study of Ores and Metals. *Proc.* Am. Phil. Soc. (1919) 401.

substances in the plate before him. There are cases, however, where additional criteria would be welcome, and for these the user of the petrographic microscope naturally turns to the phenomena of polarized light; if methods were available by means of which the optical constants of opaque substances in fine particles could be readily ascertained, these methods would be of great value not only to students of ores and opaque minerals, but also to metallographers in the study of metal alloys.

In the present paper a brief description is given of the several methods, new and old, available for the determination of the degree of anisotropism of a plate of an opaque substance when examined in normally reflected light. The plate may be illuminated either by natural (non-polarized) incident light or by plane-polarized, incident light. In the first case, anisotropism is recognized by a difference in the intensity of the reflected components of the light; in the second, by a rotation of the plane of polarization of incident, plane-polarized light waves after reflection.

## Methods Based on Intensity Contrast

The source of illumination for all methods included in this group is natural (non-polarized) light; the light may be white, colored, or monochromatic. Non-polarized light on reflection from a birefracting, biabsorbing plate suffers changes that may be detected with proper apparatus. The reflected light is, in general, elliptically polarized; a phase difference exists between the reflected components of the light; the amplitudes of the vibration, and hence the intensities of the two reflected components, are different. The phase difference between the components is ordinarily not sufficiently large to be readily measured and is, therefore, of little value as a diagnostic feature. In case the reflected light waves are plane-polarized, one of the components is more intense than the second and the excess of intensity in one direction gives rise to a certain amount of polarized light in the reflected light waves; this polarized light can be detected by methods similar to those that have long been in use for the detection and measurement of polarized light in the sky.

### *Koenigsberger's Method*

In a series of articles[2] Koenigsberger, adopting the arrangement of the Wild photometer, suggested the use of a Savart plate in conjunction with an analyzing nicol and a telescope for the detection of polarized light in light reflected from a crystal surface, the incident light to be non-polarized and to impinge vertically on the plate. The arrangement

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[2] *Centralblatt für Mineralogie, Geologie u. Paläontologie* (1901) 195–197; (1908) 565–569 597–605; (1909) 245–250; (1910) 712–713

proposed by Koenigsberger is illustrated in Fig 1. The non-polarized light (monochromatic or white) enters the system along $B$, passes through a contrast plate $Q$, consisting of a biplate of smoky quartz cut parallel to the axis and mounted so that the principal axis of the one plate is normal to that of the second, a lens $L$, for imaging the contrast plate in the image plane $H$ of the telescope $T$. The light passes through $L$ to a small reflecting prism $R$, thence through the objective $O$ to the crystal plate $C$ where it is reflected back through $O$ past $R$, through a plane-parallel, rotatable glass plate $P$ of known refractive index ($n = 1.515$), through the Savart plate $S$ and the nicol $N$ into the telescope $T$.

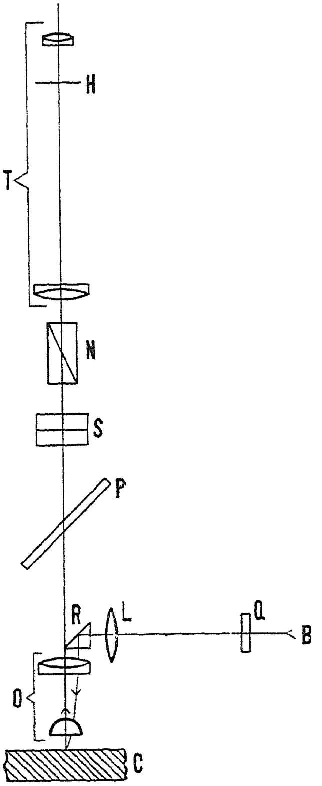


FIG. 1.—APPARATUS USED BY KOENIGSBERGER FOR MEASUREMENT OF DEGREE OF ANISOTROPISM IN OPAQUE CRYSTAL SECTIONS. PLATE IS ILLUMINATED BY VERTICALLY INCIDENT, NON-POLARIZED LIGHT.

The Savart plate consists of two plane-parallel plates of calcite or quartz cut at an angle with the optic axis sufficiently large (45°) that, when examined in convergent polarized light, the isochromatic curves of the interference figure cross the field as practically straight lines. The two plates are of equal thickness and are superimposed so that the horizontal projection of the axis in the one is at right angles to that of the axis in the second. If the combination is observed between crossed nicols when its axes include an angle of 45° with the principal nicol planes, a series of parallel, vertical, dark, interference bands appears in the field, if monochromatic light is used. If a white light source is used, the bands are colored, except one central band. These bands are similar in appearance to the bands in a quartz wedge or a Babinet compensator; but their mode of formation is different because they depend on very slightly convergent polarized light for their formation while in the quartz wedge the birefringence introduces the required path difference. From the mode of formation, it is evident that if non-polarized light is used no such bands will appear, but that with a small percentage of polarized light present there is superimposed on the white field a series of colored bands the intensity of which increases with the amount of polarized light present. Under the best conditions of setting, the maximum intensity is obtained at the center of the light bands, namely one-half of the total non-polarized light

and one-half of the total polarized light; whereas at the center of the dark central band only the non-polarized portion is transmitted. At intermediate points the non-polarized light and a part of the polarized component are transmitted. The field, therefore, alternates in intensity; the least amount of polarized light that can be detected depends obviously on the least perceptible increment in intensity that the eye is able to detect. The data of Koenig and Brodhun place this difference-limen at 1.6 to 2 per cent. for favorable intensities of illumination. Pickering and others[3] estimate that in sky polarization about 1 per cent. of polarized light can be detected. Koenigsberger asserts that 0.3 per cent. difference can be detected, but experiments by the writer using a rotatable, plane-parallel plate and different sources and intensities of illumination indicate that the figures of Koenig and Brodhun are more nearly correct and that Koenigsberger's statement of the precision attainable is about five times too great. The difference between different settings, especially for large intensity differences, is of course considerably less than the least perceptible increment, but this difference may not be considered to be the least perceptible increment itself.

In order to render more readily visible the Savart bands and the point of exact compensation, Koenigsberger employs a contrast biplate of smoky quartz, which introduces a difference in intensity between the components (result of pleochroitic absorption) and thus produces a shift of the lines in the halves of the field; these lines are then further shifted by the changes in intensity resulting on reflection from an anisotropic crystal plate.

In the practical application of this method, it is essential that the plate to be examined be well polished and normal to the axis of the microscope; that the reflecting prism be not too large; that the rotating glass plate $P$ be mounted with its axis at 45° with the upper nicol plane; that the Savart plate be accurately constructed and be normal to the microscope axis; that the telescope be accurately focused on infinity (in order that convergent light of only very small angular aperture pass through the Savart plate). To measure the degree of anisotropism, the glass plate $P$ is rotated until the effect of the crystal plate is exactly compensated and the Savart bands disappear or are separated by exactly half a band if the contrast arrangement is used.

The tilted plate employed by Koenigsberger introduces differences of intensity normal and parallel with the plane of symmetry. If $i$ is the angle of incidence of the light waves entering the tilted plate, and $r$ is the angle of refraction, the ratio of intensities of the two components parallel and normal to the plane of symmetry is equal to $\cos^4 (i - r)$. For any given angle of tilting of the glass plate the intensity ratio can be ascertained from this expression.

[3] Report of U. S. Naval Observatory on Total Eclipse of July 29, 1878.

Photometrically, the gradation in intensity between the different Savart bands is less advantageous than the juxtaposition of two or more evenly illuminated fields—the one at minimum intensity, the next at maximum intensity. This principle is used in measurements of sky polarization and can be readily applied to the present problem.

*New Method*

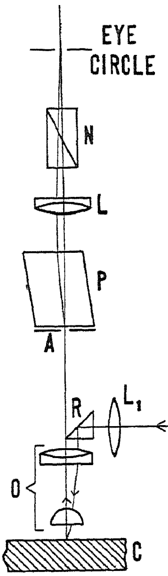


FIG. 2.—APPARATUS EMPLOYING CALCITE RHOMB FOR MEASUREMENT OF DEGREE OF ANISOTROPISM OF OPAQUE CRYSTAL PLATE. PLATE IS ILLUMINATED BY VERTICALLY INCIDENT, NON-POLARIZED LIGHT.

This method uses a cleavage plate of calcite of such thickness and an aperture of such width that the fields from the two rays just touch (Fig. 2) as in the well-known Haidinger lens used by mineralogists for observing pleochroism in minerals. The calcite rhomb should be protected on each end by a cover slip cemented to it.

In Fig. 2 the non-polarized, incident light is reflected by the prism $R$ through the objective $O$ to the opaque plate $C$, whence it is reflected back through the objective $O$ and the aperture $A$ of the calcite cleavage piece $P$. In this plate, two images of the aperture $A$ are formed as a result of the double refraction of calcite; these are then observed through a nicol prism and low-power lens or through a low-power microscope consisting of a weak objective lens $L$ located below the nicol prism $N$ and a low-power eyepiece. The plate $C$ is imaged by the objective on the aperture $A$, which may then be viewed directly by a low-power eye lens or be re-imaged by the lens $L$ into the image plane of an eyepiece (not shown in Fig. 2). Because of the calcite plate, high magnifications should not be used in the eyepiece system. With this arrangement, the plate is imaged in the field of view and the phenomena are observed that it alone produces. A still better field can be obtained by adopting an arrangement[4] in which two biprisms serve to produce a satisfactory junction line between the two fields. For the same purpose, a Koenig-Martens photometer may be employed to advantage, and was, in fact, used by the writer for most of the tests.

Equipped with any one of the foregoing devices, the observer illuminates the specimen to be examined and studies the character of the light that is reflected. On rotating an isotropic plate, no differences in intensity between the two halves of the field are observed; a birefracting and

[4] For a discussion of these types of photometers see F. E. Wright: *Jnl* Optical Society of America (1920).

biabsorbing plate, on the other hand, polarizes to a certain extent the light on reflection, and this gives rise to a difference in intensity of the reflected components.

To ascertain the amount of polarized light present in the reflected light: (1) Set the analyzer so that its plane includes an angle of 45° with the principal plane of the calcite rhomb or the wollaston cube of the photometer. In this position the two fields are equally intense when the photometer is pointed at a source of non-polarized light. The photometer is now mounted on the microscope and the illuminated opaque plate rotated until the two fields in the photometer are equal in intensity. (2) Turn the stage 45° from this position and observe maximal contrast of photometer fields. In this position the plane of vibration of the polarized light includes an angle of 45° with the principal plane of the analyzer; the polarized light enters only one field of the photometer and contributes its share to the intensity of this field. The intensity of the second half of the field is due solely to the non-polarized light or $\frac{I_n}{2}$; that of the first half both to the non-polarized light and to the polarized light or $\frac{I_n}{2} + I_p$. (3) The analyzer is now turned to a position $\alpha$ at which the two fields appear equally illuminated. For this position the equation

$$\frac{I_n}{2} : \left(\frac{I_n}{2} + I_p\right) = \tan^2 \alpha$$

is valid; from this we deduce readily

$$\frac{I_p}{I_n + I_p} = \cos 2\alpha$$

This expression enables the observer to ascertain directly the amount of polarized light present.

In case the arrangement of the photometer is such that, when it is pointed at a source of non-polarized light, equal field intensities are observed when the principal plane of the analyzer includes an angle $\alpha_1$ different from 45°, the ratio of the intensity of the polarized light to the total light (polarized and non-polarized) is

$$\frac{I_p}{I_n + I_p} = \frac{\tan^2 \alpha_1 - \tan^2 \alpha}{\tan^2 \alpha_1 (1 + \tan^2 \alpha)} = \frac{\sin(\alpha_1 + \alpha)\sin(\alpha_1 - \alpha)}{\sin^2 \alpha_1}$$

The above procedure enables the observer to ascertain not only the positions of the polarization axes of the opaque plate but also to measure the degree of its departure from isotropism (degree of biabsorption and birefringence).

These new methods, based on intensity contrast, allow the observer to view the image of the plate directly, while in the Koenigsberger method

the Savart bands are not readily distinguished unless the plate is brought out of sharp focus and even then the irregularities of the image tend to render indistinct and uncertain the faint Savart bands.

### Methods Based on Detection of Rotation of Plane of Polarization

These methods have been long in use by petrologists and the principles underlying the construction of the different devices need not be repeated here.[5] Suffice it to state that normally incident, plane-polarized light impinging on the plate with its plane of polarization at 45° to the optical ellipsoidal axes of the section suffers on reflection a rotation of the plane of polarization, the angular amount of which depends on the degree of the birefringence and the biabsorption.

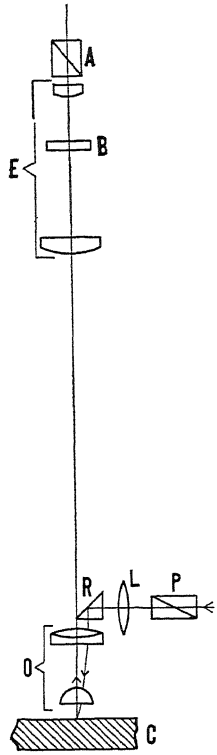


Fig. 3.—Apparatus employing bi-quartz-wedge-plate $B$, for measurement of degree of anisotropism in opaque crystal plates. Plate is illuminated by vertically incident, plane polarized light.

#### *Koenigsberger's Method*[6]

In this method, Koenigsberger adopted the arrangement shown in Fig. 3. The incident light passes first through the polarizing prism $P$ (vibration plane horizontal or vertical), the total reflecting prism $R$, and the objective $O$, to the opaque crystal plate $C$, whence it travels, after reflection, through the objective $O$, the field lens of the eyepiece, the Biot sensitive tint plate $B$ (of quartz cut normal to the optic axis and of such thickness, 3.75 mm., that it shows the sensitive tint between crossed nicols) the eye lens, and the analyzer $A$, to the eye of the observer. The rotation of the plane of polarization on reflection is detected by the change in interference color on rotating the crystal plate. Koenigsberger emphasizes the fact that this method is not so sensitive as his first method and is only qualitative in nature.

#### *Hanemann's Method*

In 1913, Hanemann[7] improved Koenigsberger's method by substituting a Soleil-Biot plate (two adjacent Biot quartz plates 3.75 mm. thick cut normal to the axis, the one of right-handed, the second of left-handed quartz) for the single Biot

[5] These are treated at length by F. E. Wright in the Methods of Petrographic Microscopic Research, Carnegie Institution of Washington *Publication* 158 (1911) 115–148.

[6] *Centralblatt Miner. Geol., Paläontol.* (1909) 245–250; (1910) 712–713; *Metallurgie* (1909) **4**, 605–608.

[7] K. Endell and H. Hanemann: *Stahl und Eisen* (1913) 40; *Zeit. anorg. Chem* (1913) **83**, 267–274; (1914) **88**, 265–268.

plate. This arrangement introduces the feature of color contrast in adjacent fields and is accordingly more sensitive than the Koenigsberger method. The Bertrand eyepiece which is in common use by petrologists would serve the purpose as well as, and possibly better than, the Soleil-Biot plate.

### *New Method*

The methods of Koenigsberger and of Hanemann are based on changes in color (sensitive tint) on rotation of an anisotropic plate. In the case of strongly colored substances, especially yellow minerals, the natural color dominates the field so effectively that the sensitive tint is no longer present as such and the slight changes in hue on rotation of the stage are only with difficulty detected and may be overlooked. The sensitiveness of these methods varies, therefore, with the color of the substance under examination. It is also difficult to obtain a uniformly colored field under average conditions of illumination, especially near the junction lines in the field.

In measurements of this nature in which the conditions of observation vary within wide limits while the sensitiveness of the eye (threshold vision) is more or less fixed, it is essential that the sensitiveness of the device employed for the measurement be variable so that the most sensitive conditions of observation can be obtained. It is on this principle of variable sensibility that the writer's bi-quartz-wedge-plate[8] was constructed.

On substituting the bi-quartz-wedge-plate for the Biot plate of Koenigsberger or the Biot-Soleil plate of Hanemann, the most sensitive arrangement for the detection of anisotropism in opaque substances is obtained. The method is exceedingly simple and requires no apparatus in addition to that furnished with a research-model, petrographic microscope. The observations are made either in white or in monochromatic light. The degree of anisotropism is indicated by the amount of angular rotation of the analyzer required to produce equal intensity of illumination in the adjacent halves of the bi-quartz-wedge-plate.

Other devices, such as the Lippich half-shade prism, may be substituted for any one of the rotating quartz plates and wedges; but such devices are not recommended for practical work because the apparatus is more complicated and the attainable accuracy is not increased.

### *Effect of Reflecting Prism or Plate on Character of Light*

An extended series of experiments with different kinds of reflectors, including total reflecting prisms, silvered surfaces, silver glass mirrors with central aperture or a small disk mirror in center of field, or an Abbe reflecting cube, was carried out to ascertain, if possible, which type is the

[8] *Amer. Jnl. Sci.* (1908) [4] **26**, 377–378; Carnegie Institution of Washington, *Publication* 158 (1911) 140–142.

best for such work, especially for use with the second group of methods. The general equations state that in case the azimuth of the plane of polarization of the incident wave is zero or 90°, the reflected beam is still plane-polarized; practical experience with such reflectors shows, however, that the reflected beam always reveals traces of elliptical polarization even when the plane of polarization is horizontal or vertical. This is no doubt due, in the case of reflecting glass surfaces, to internal reflections; with a blank metal reflecting surface, such as the fresh silver side of a silvered mirror, it may result from strains in the outer film. Be the cause what it may, in no experiment was the elliptic polarization entirely removed and the accuracy of the settings was correspondingly diminished. Experience with the different types of reflectors did not demonstrate marked superiority of any one particular type. It is, of course, essential that the plane of polarization of the incident beam be strictly horizontal or vertical in order to reduce the amount of elliptic polarization present to a minimum.

### *Effect of Thin Surface Films on Character of Reflected Light*

Drude[9] was the first to emphasize the importance of thin surface films in effecting the polarization relations of reflected and of transmitted light waves. In the case of opaque substances (ores and metals), the surface film effects may be serious and cause an isotropic mineral to show phenomena of anisotropism. Thus a sheet of copper or other isotropic metal shows distinct anisotropic effects. The rolled surface of the metal is striated and part of the polarization effects may be of the nature of grating polarization phenomena, or may result from reflection at oblique surfaces. Many ground surfaces of isotropic pyrite are decidedly anisotropic. In certain cases this is probably due, as Koenigsberger suggested,[10] to grinding striæ; but in other pyrite sections the anisotropism persists in spite of care taken to eliminate grinding striæ. This may be the result of surface film effects or of strain birefringence in the pyrite. Its presence in random sections, even carefully ground, serves as a danger signal, warning against the drawing of too positive conclusions from a single section. As a general rule, pyrite sections are isotropic and the conclusion is warranted that it is isometric, as crystallographic measurements prove it to be. The statement of Koenigsberger that surface films affect only the phase difference and not the amplitudes of the reflected beams is borne out neither by theory nor by practical experience. In general, it may be stated that surface film effects are not so pronounced as to affect seriously the validity of the determination.

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[9] *Ann. Phys.* (1891) **43,** 146; *Lehrbuch Optik* (1906) 272–280; Winkelmann's *Handbuch Phys.*, **6.**

[10] *Centralblatt Miner., Geol., Paläontol.* (1908) 597.

### *Study of Opaque Sections by Foregoing Methods*

For the successful examination of opaque sections by the methods of this class, it is essential that the principal plane of the polarizer be exactly horizontal or vertical; that the reflecting surface of the opaque crystal be carefully prepared, free from scratches and grinding grooves; that the principal plane of the analyzer be at right angles to that of the polarizer—this is best tested by using a glass or other test plate of isotropic substance and observing uniform field intensities or colors in the bi-quartz-wedge-plate or the Bertrand eyepiece. The opaque mineral plate is now placed on the stage of the microscope and illuminated. As the stage is rotated, the changes are observed in the color or relative intensities of the quadrants of the Bertrand eyepiece or the halves of the bi-quartz-wedge-plate, respectively. At four positions during a rotation of 360°, the field is of uniform color or intensity. In this position, one of the optical ellipsoidal axes of the plate coincides with the principal plane of the polarizer. From this position, the plate is now rotated 45°. With the bi-quartz-wedge-plate, the maximum contrast is here obtained. The analyzer is now rotated through a small angle to restore equality of intensity or color over the entire field. The angle of rotation is a measure of the departure of the plate from isotropism.

This method depends for its success on the ability of the eye to determine the exact position of total extinction. With a strong source of light and a bi-quartz-wedge-plate, it is not difficult to determine the position of extinction within a few minutes of arc, thus rendering this method considerably more sensitive than any other method at present available

## Summary

To the user of the petrographic microscope, the meager amount of information that can be gathered from the use of polarized light in the study of opaque mineral or metal sections is disappointing, but even the results obtainable are important diagnostic features and should not be disregarded. In special cases where crystallographic symmetry relations of the plate under examination prescribe the positions of the principal axes of refringence and of absorption, as in isotropic, uniaxial, and the principal planes of orthorhombic crystals, the reflected waves are plane-polarized and not elliptically polarized, as in the general case. As a result, normally incident, plane-polarized, light waves whose vibration directions are either parallel or normal to the plane of symmetry are reflected as plane-polarized waves; but the intensities of the two reflected waves are different because of the difference in the refractive and absorption indices parallel and normal to the plane of symmetry.

Normally incident, non-polarized light contains after reflection a certain amount of plane-polarized light and this amount increases with the strength of the birefringence and of the biabsorption in the crystal

plate. The presence of plane-polarized light in natural light can be detected by any one of the well-known physical methods, such as are commonly used in determinations of sky polarization. Of these methods Koenigsberger adopted the Savart plate with rotating glass compensator; a second method is suggested, which employs either a single calcite cleavage plate with proper aperture (after the manner of the Haidinger lens or the Pickering photometer) or a small portable Koenig-Martens photometer. This method is superior to the first in being simpler in adjustment and in manipulating. It is based on a photometrically better principle, namely, contrast of two or a series of illuminated fields, the first at minimum intensity, the adjacent field at maximum intensity with a sharp line of demarcation, which disappears when the intensity of illumination in the adjacent fields is the same.

The attainable accuracy depends on the least perceptible difference in intensity of illumination that the eye can detect; this is about 1.5 to 2 per cent. under ordinary conditions of illumination. An opaque crystal therefore of such weak birefringence or weak biabsorption that the difference in intensity between the reflected components is less than 1.5 per cent. appears isotropic. In transmitted light, differences in birefringence of only 0.02 per cent. of the refractive index are readily detectible. The methods based on differences in intensity of the reflected components of vertically incident light are therefore fifty or more times less sensitive in the detection of anisotropism than the methods based on the phenomena of plane-polarized, transmitted, light waves.

In case the vertically incident light is plane-polarized the difference in amplitude between the reflected components normal and parallel to the plane of symmetry causes a rotation of the plane of polarization; this can be detected and measured by any one of a number of devices in common use by petrologists. Koenigsberger used a Biot sensitive-tint quartz plate in a qualitative way to detect anisotropism. Hanemann improved Koenigsberger's method by employing the Biot-Soleil sensitive-tint, quartz bi-plate and obtained thereby a color contrast in two adjacent fields. The ordinary Bertrand eyepiece used by petrologists is suggested as a still further and equally simple method for detecting anisotropism.

A still more sensitive and better method is, however, to use the writer's bi-quartz-wedge-plate, in which the sensibility is variable and can be adjusted to meet the conditions of illumination and thus to produce the most favorable conditions for extreme sensitiveness and consequent accuracy of results of measurement. This method is simple and is the most sensitive at present available for detecting and measuring anisotropism in opaque substances. The degree of anisotropism is indicated by the amount of angular rotation, on reflection, of the plane of polarization of vertically incident light waves. The accuracy of these methods depends on the threshold limit of vision (least intensity of light that the eye can

detect); this is different from the least perceptible difference in intensity of illumination of two adjacent fields (contrast sensibility of the eye) on which the sensitiveness of the first group of methods depends.

The sensitiveness of the second group of methods can be increased by increasing the intensity of the light source and also by introducing some contrast device, such as the bi-quartz-wedge-plate, or the Lippich biprism. In general it may be stated that, for practical purposes, the methods of the second group are simpler in manipulation and more sensitive than those of the first and therefore better. A disturbing factor is the introduction of a small amount of elliptically polarized light by the reflecting prism or plate. The phase differences between the two components of the reflected waves are, in general, small and do not seriously affect measurements by the methods outlined above which are based on the amplitude differences between the two components. The phase differences can be measured by one of several devices of which the Babinet compensator is the best and most widely used; but for the detection and measurement of anisotropism, methods based on phase differences between the components after reflection of vertically incident, plane-polarized light, are not, in most cases, of practical value.

The use of immersion liquids and a monochromatic light source for obtaining the greatest differences in intensity between the reflected components and thus to increase the sensitiveness of the methods for detecting anisotropism is possible; but the gains derived therefrom do not appear to warrant the bother which their use incurs.

The theoretical investigation has demonstrated that it is, in general, not possible to measure both the refractive indices and the absorption indices of an opaque body by means of the phenomena resulting on the reflection of vertically incident light waves; but it is possible in special cases to determine the degree of anisotropism and also the positions of the polarization directions in a given crystal plate. These data taken in conjunction with crystallographic data enable the observer, just as do birefringence and extinction angles in transparent crystals, to draw conclusions regarding the crystal system of the substance under investigation. But in opaque substances, the precision attainable is relatively low and the phenomena which can be observed are few and restricted in scope. As a result one cannot expect from the application of polarized light to such substances the harvest of optical data which has been gathered from transparent crystals. For opaque bodies the possibilities are few, the limitations are great, and recourse must be had in practical diagnosis to other methods of determination, such as behavior of the substance toward abrasives and solvents, color, density, hardness, etc., in addition to the determination of the degree of anisotropism.

# Geology and Mining Methods at Pilares Mine*

BY W ROGERS WADE† AND ALFRED WANDTKE,‡ PILARES DE NACOZARI, SONORA, MEX.

(Chicago Meeting, September, 1919)

THE Pilares mine of the Moctezuma Copper Co. is situated at Los Pilares de Nacozari, Sonora, Mexico, about 75 mi. (120.7 km.) south of the international boundary and about 7 mi. (11.26 km.) east of the town of Nacozari de Garcia. It is situated at an elevation of 5000 ft. (1524 m.) in one of the northward trending ridges that form part of the mountain mass that culminates in the Sierra Madre de Mexico. To the east, south, and west, the topography is rugged and diversified; to the north less diversity exists, ranges of mountains rise sharply out of almost level ground and give the basin range structure characteristic of the Southwest.

The climate is dry, the average rainfall of 15 in. (38 cm.) being confined almost entirely to the months of June, July, and August. The winter consists of a few days of disagreeable weather when a little snow may fall; the remainder of the year is thoroughly delightful. Vegetation is typical of a semi-arid region. In the cañons near the mine, where water flows intermittently, willows thrive; on the northern slopes of some of the mountains and in some of the broader cañons pines may even become abundant.

The dominant rocks are flows and pyroclastics which are cut by dykes, stocks, and batholiths. The mountain just west of Nacozari is, in part, composed of limestone of an unknown age which occurs as a syncline and is undoubtedly a part of a prevolcanic land surface. Overlying the limestone and faulted against it, are the volcanics; cutting the limestone is a coarse biotite granite. Whether the granite is older or younger than the volcanics has not been definitely established.

No attempt has been made to differentiate the rock types of the volcanics, of which there are exposed 3000 ft. (914 m.) in thickness of basic and acid flows (andesites, latites, and rhyolites) and andesite and latite tuffs and breccias. The latite breccias, the youngest of these volcanics, cover the highest mountains in the immediate vicinity. The

* In the preparation of this paper access has been had to the report made by John M. Boutwell, in 1911, on the geology and ore occurrence at Pilares. So little that is really new concerning the ore and its occurrence has been discovered since then that the paper must be considered an abstract of Mr. Boutwell's report.

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age of the volcanics has not been definitely determined; Boutwell placed their age as post-Carboniferous and probably late Cretaceous. South of the international boundary, the volcanics appear to rest on upturned Cretaceous rocks, hence the volcanics at Pilares, which are to be cor-

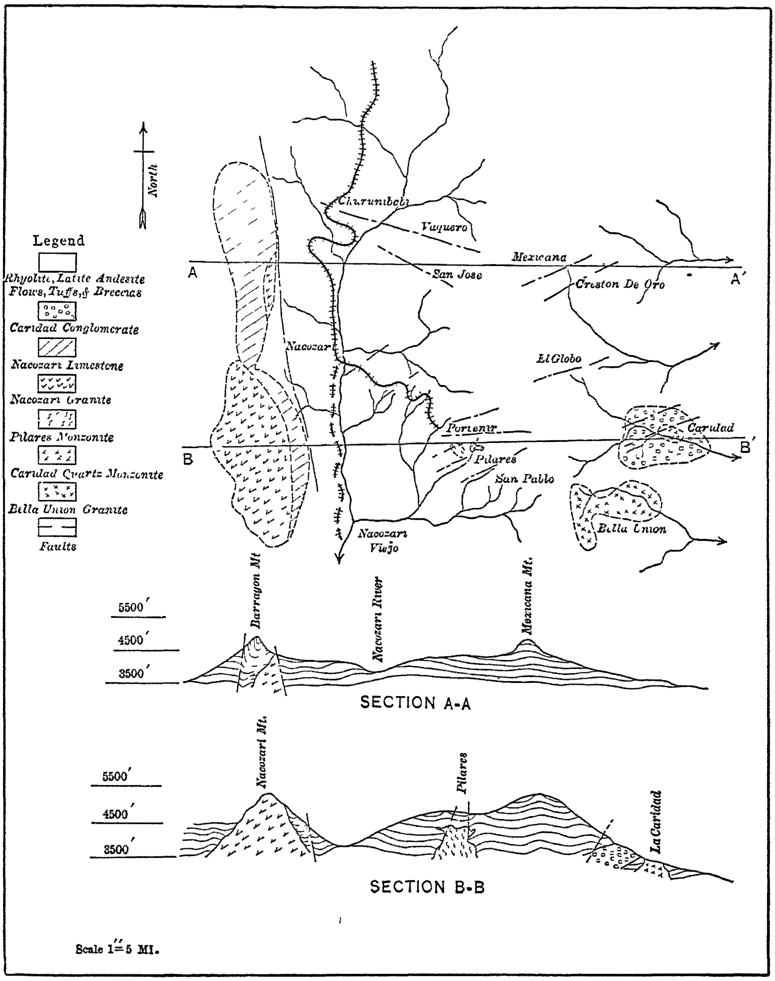


FIG. 1.—PILARES MINE AND VICINITY.

related with those south of the international boundary, may be post-Cretaceous and perhaps of early Tertiary age.

The period of volcanic activity was apparently followed by crustal warping which, however, folded the rocks but slightly. This period of folding was succeeded by a period of igneous intrusion, during which time diorites, monzonites, quartz-monzonites, granites (?), and basaltic dikes were developed. Following the period of igneous intrusions came

a time of renewed crustal warping, which was accompanied by faulting. Erosion followed, leaving many mountains as monadnocks and filling some of the valleys with gravels.

As viewed 15 mi. (24.14 km.) north of Nacozari, and from Nacozari Mountain, there seem to be indicated at least three cycles of erosion: One prior to the deposition of volcanics; a second following the gentle doming of the volcanics, for an erosion surface cuts across the volcanics, leaving mountain ranges as monadnocks; and a third, the present cañon cutting cycle, that to the south has carved wide and steep walled valleys, but in the vicinity of Pilares and Nacozari is represented by V-shaped gorges.

### *Geology at Pilares*[1]

At the Pilares mine, the chief rocks are latite and andesite tuffs and breccias. These formations are cut by intrusions of monzonite, diorite, quartz-porphyry, and a bifurcating diabase dike, locally known as "caliche;" Figs. 2 and 3 show the relationship of these rocks. The sections permit of the following generalizations: A period of volcanic activity was followed by a period of gentle crustal movement that flexed the volcanics and produced several easterly and northwesterly faults. The tilted beds were then cut by the monzonite and diorite intrusions, the diorite being apparently a chill phase of the monzonite. Further faulting and shearing developed the elliptic plan of the orebody and ushered in the period of mineralization that resulted in the localization of orebodies wherever the invaded rock was favorable. These favorable localities were undoubtedly largely determined by faulting and fracturing, which produced open spaces and gave ingress to mineral-bearing solutions. The result is that orebodies occur wherever the pre-ore shattering prepared the ground and where the conditions of temperature, pressure, and concentration were favorable to ore depositions. The deposition of the ore was followed by a period of fracturing and dike intrusion, the "caliche" dike being a representative of this period. After the dike came renewed faulting with movement along all of the shear zones, but particularly manifesting itself along the Esperanza and Luna shear zones, where horizontal and vertical displacements of about 100 ft. (30.48 m.) are recorded.

As a result, there has been produced a body of mineralized ground cylindroid in shape, the outline being determined by fault planes. The walls of the orebody in the upper workings are determined by fault planes in the latite and andesite breccias; below the 400-ft. level (121.9 m.) the wall rock is monzonite. In depth, this rock always forms the outer wall

[1] The deposit has been briefly described by S. F. Emmons [*Econ. Geol.* (1906) **1**, 629–643], by W. Lindgren ["Mineral Deposits" (1913) 496], and by W. H. Emmons [Enrichment of Ore Deposits, *Bull.* 625, U. S. Geol. Survey (1917) 225–226].

of the oval. Within the core of the oval are numerous tongues of diorite, which are, no doubt, chill phases of the monzonite. The periphery of the oval almost invariably bears ore; within the oval are lenses of ore irregularly scattered.

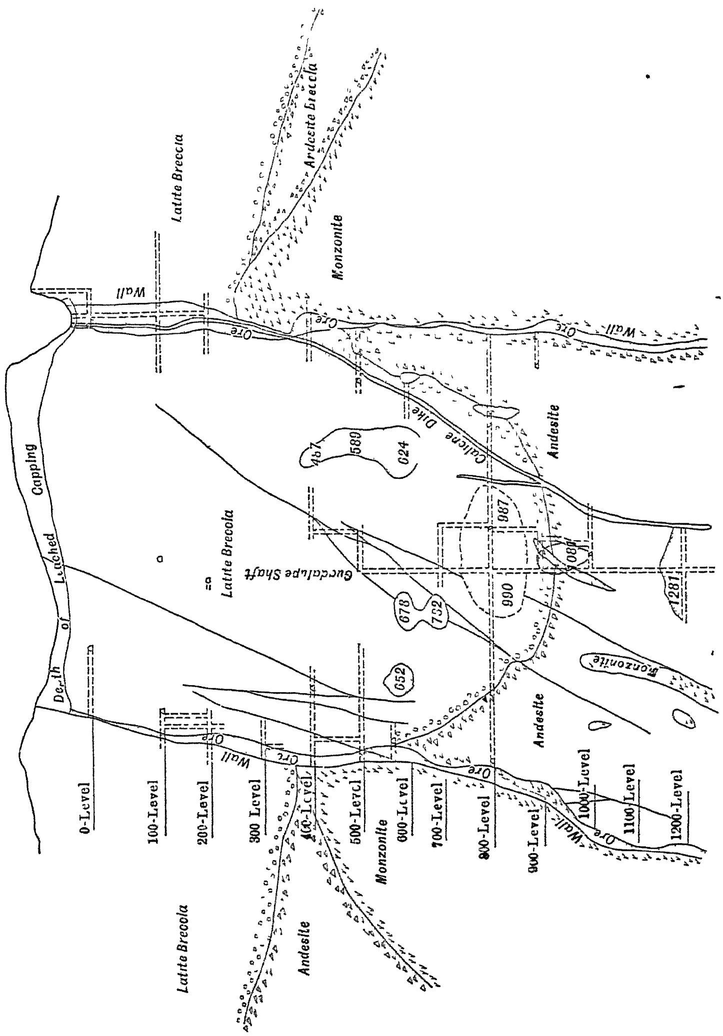


FIG. 2.—VERTICAL SECTION AT RIGHT ANGLES TO MINE OVAL THROUGH GUADALUPE SHAFT.

*Structure.*—In the vicinity of the mine, the volcanic rocks appear to form part of an unsymmetrical syncline, inclined to the east, which strikes northwest and pitches southeast. Two main sets of faults are responsible for the shape of the orebody, mark the zones of shattering that permitted the entrance of the mineral-bearing solutions, and indicate the directions of movement that followed the period of dike intrusion. The

first, or easterly set of faults, has an approximate N 70° E trend; the second, or northwesterly set, has an approximate N 40° W trend. Variations of 10° to 20° from these directions occur. The easterly set of faults, with the exception of the Esperanza fault zone and a few minor slips, dip steeply to the south; the northwesterly faults dip, as a rule, steeply to the southwest.

Tangential movement, causing a fracturing of the corners of the parallelogram formed by the intersection of the two sets of faults, would account for the oval outline of the mine. The corners of the oval, other things being equal, containing the greatest number of fractures, would

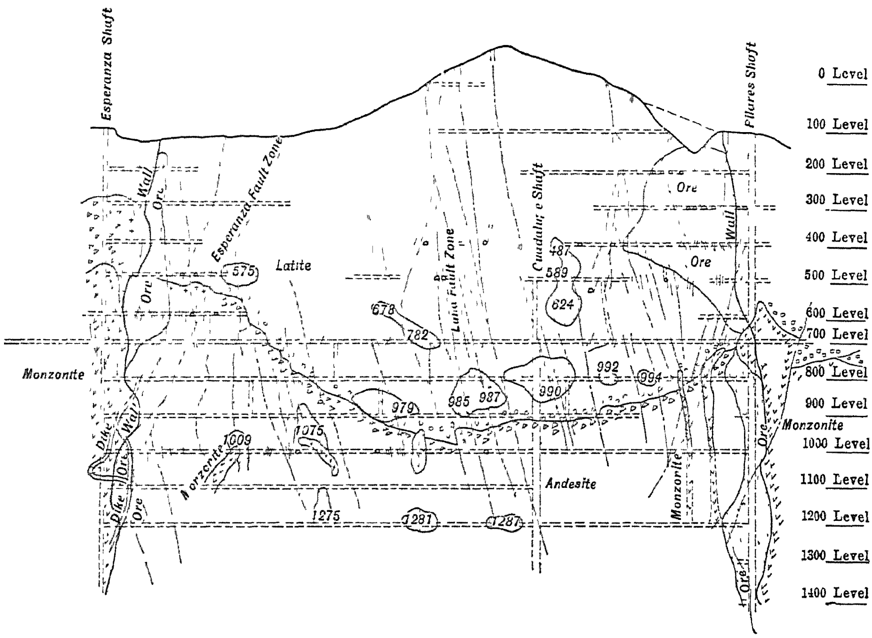


FIG. 3.—VERTICAL SECTION THROUGH MINE OVAL FROM ESPERANZA TO PILARES SHAFTS.

also become the most favorable places for ore deposition. Post-ore movement appears to have been along two directions: the earlier, which the diabase followed, was northwest; the later, which cuts the dike and offsets the orebodies, was easterly. The combination of a soft dike and pronounced fault zones has produced a real difficulty in the method of mining.

*Ore-bearing Rocks.*—The only volcanic rocks that have been cut by the mine workings are the latite and andesite breccias. The latite breccia has a field habit that, in places, is anything but that suggestive of a breccia. On Paulina Mountain, the breccia habit is clearly defined, but masses of rock resembling parts of a flow are frequently encountered

underground. Some of these masses, where they are cut by mine workings, are fully 40 ft. (12 m.) across, and at least 10 ft. (3 m.) in the other directions. In the exposed portions of the glory holes in Pilares Mountain, apparent boulders can be seen that measure 30 by 8 ft. (9 by 2 m.) with the remaining dimension unknown. Although the usual habit of the latite is that of a breccia composed of fragments not over 10 in. (25 cm.) in diameter, volcanic bombs and a lathe-like habit are not unusual. The bombs, which have attracted the attention of the Mexican miner, who calls them "calaveras," may be round or spheroidal and are always encased in a number of shells, resembling the layers of an onion. The sulfides frequently replace these shells or are deposited between two shells, forming an ore that is striking in appearance. Where the latite has a lathe-like habit, the structure would almost seem to be a primary brecciated flow structure in the latite. Where this type of rock has been mineralized and has been replaced by the ore sulfides, it has a peculiarly beautiful appearance.

The andesite has no unusual field habits. The mine workings clearly show that this rock underlies the latite breccia. Any theory of ore deposition that suggests that the mine occupies a volcanic pipe can, therefore, be dismissed as untenable.

The monzonite is the important intrusive from an economic standpoint, for it undoubtedly "brought in" the ore. The field relations of the rock are shown in the mine sections. The rock is dark, fine-grained holocrystalline, and of granitic texture, with the habit of a diorite rather than a monzonite. The intrusive tongues are slightly porphyritic and resemble dioritic dike rocks. The occurrence[2] is strongly suggestive of a stock of which the top has been depressed by shrinkage due to cooling and the weight of the superincumbent mass. Contact effects, due to heat of intrusion, are not pronounced. Those due to solutions given off by the intrusive are of economic, rather than petrologic interest.

*Metamorphism.*—Owing to the disposition of the ore-bearing members of the mine, an exceptional opportunity is given for the study of the alteration of a set of rocks when due to the heat of intrusion of an igneous body and when due to the action of abundant solutions that deposited ore. Although numerous contacts between monzonite-latite and monzonite-andesite are exposed, in none of them do the invaded rocks show well-marked alteration, such as recrystallization, due to the heat of the intrusive. Almost all of the alteration that can be observed without the aid of the microscope appears to have been produced by invading solutions that carried the ore. Each of the ore-bearing members suffered characteristic alterations and, because of its chemical unlikeness to the other

[2] See the discussion on block subsidence by R. A. Daly, "Igneous Rocks and Their Origin," 196. 1914.

members, each underwent a special change, though some of the new minerals introduced became prominent in all.

Latite breccia is not only the most siliceous of the different rock types, but it differs from them in physical properties, as regards fineness of grain and minuteness of comminution. The most characteristic change is to sericite; less characteristic is the alteration due to specularite, quartz, and scheelite. Of economic importance is the complete or partial replacement of the rock by chalcopyrite, which completely replaces masses of the latite weighing a ton or more without visible alterations having been produced in the adjacent latite. Where pyrite enters, the rock at once becomes soft and sericitic. There are in the mine large volumes of ground that contain 2 to 5 per cent. of pyrite and of which the latite has been largely altered to sericite.

The specularite is confined more or less to the immediate vicinity of the strong shear zones, although it also occurs disseminated. Whereever specularite enters, the rock becomes silicified.

Although not a dominant type of alteration, scheelite has been introduced in sufficient quantity to warrant mentioning the process of scheelitization. Locally the scheelite has become sufficiently abundant so that a serious attempt was made to mine one stope for its scheelite content. As a rule, chalcopyrite is more abundant than pyrite in the scheelite-bearing rock.

The andesite breccia, being more basic than the latite breccia, has undergone changes which are correspondingly different. The chief alterations are to sericite and chlorite; quartz, specularite, and scheelite are found in but a minor degree. As in the latite, so in the andesite, the sericite is accompanied by pyrite, the limits of orebodies being frequently determined by the entrance of sericite which acts as a signal that pyrite is present. The andesite becomes, moreover, soft and greenish due to the entrance of chlorite, a mineral that is but sparingly found in the latite. Upon the lower levels, large volumes of ground are seen to be chloritic and correspondingly poor in chalcopyrite. Up to the present time, chloritic ground has failed to yield an orebody, although large volumes of chloritized andesite contain 0.7 to 1.2 per cent. of copper. It seems, therefore, to be a rule that chloritic andesite forms poor ground in which to prospect. Frequently the brecciated monzonite, which contains excellent ore, will be bounded by chloritized andesite breccia, lean in chalcopyrite and containing a little pyrite.

The monzonite is undoubtedly the igneous rock to which the ore is genetically related. Up to the present time but few orebodies have been found in this rock. The ore that does occur is found where the monzonite has been strongly fractured. The changes suffered by the rock are similar to those affecting the andesite—sericitization, chloritization, calcitization, silicification, and scheelitization being the types of altera-

tion. As with the andesite, both sericite and chlorite signify a meager copper content, whereas the quartz and scheelite indicate good copper values. Of economic interest is the fact that, where sulfides occur in monzonite, there appears to be a preferential development of chalcopyrite. On the 1200-ft. level (365 m.), in a drift that penetrates the monzonite for 100 ft. (30 m.), there was found a vein of gypsum associated with pyrite. Near by was found the only megascopic epidote as yet observed in the mine.

As an indicator of ore, the degree or the kind of metamorphism may naturally become an important guide, but no infallible rule or set of rules has as yet been established. In general, a marked sericitization of the latite and andesite indicates pyrite, but in some of the wall orebodies sericitized latite carries good copper values wherever scheelite and quartz become accessory minerals, but even these orebodies grade into an assay wall of sericitized latite and pyrite. In the andesite, sericite also indicates pyrite; but in this rock both chlorite and specularite become important diagnostic minerals, for the soft chloritized rock almost never carries sufficient chalcopyrite to warrant the term ore. When the andesite, even though chloritized, is also silicified or carries specularite, then chalcopyrite becomes more abundant and bodies of minable ore are found.

Inasmuch as few orebodies have been developed in the monzonite, its characteristic alterations are not entirely apparent. The prime requisite, as with the other rocks, is shattering, and where this is pronounced, the monzonite makes ore. As has been stated, the rock shows no striking megascopic alterations, and when sulfides do occur, chalcopyrite is usually the more abundant. As with the other rocks, sericitization signifies pyrite, whereas silicification and scheelitization indicate copper.

Another feature, the significance of which is not entirely apparent, is the vuggy character of the latite and of the andesite, to a lesser degree. These open spaces evidently resulted from solution being a more pronounced process than deposition. The druzzy lining of the vugs generally consists of calcite, to a lesser degree of quartz, chlorite and specularite, and rarely of the sulfides of iron and copper. The meager data available would seem to indicate that the vuggy rock is found on the peripheral part of an orebody.

*Classification of Orebody.*—Lindgren[3] has classified the Pilares orebody as belonging to those deposits definitely related to igneous rocks, but found near the surface. Since, however, a great amount of exploitation has been done since the observations were made on which Lindgren based his classification, it seems well to reclassify the occurrence in the light of the more recent work. The ores are intimately associated with a

[3] *Op. cit.*

monzonite intrusive and the ores were undoubtedly precipitated out of the emanations given off by this intrusive. The mineral association and the type of rock alteration are those characteristic of deposits formed at intermediate depths. In their relation to an intrusive, the ores resemble contact metamorphic deposits,[4] but at Los Pilares the country rock is latite and andesite instead of limestone, and so the characteristic contact minerals are missing. The deposit is placed, therefore, among those formed at intermediate depths due to direct emanations from an igneous intrusive to which the deposit stands in intimate relationship.

*Ore and Associated Minerals at Pilares.*—The chief ore mineral is chalcopyrite; the lesser ore minerals are chalcocite, covellite, and bornite. Both tetrahedrite and native copper are rare. Although native copper has been observed on the 100-ft. (40-m.) level, it has been found more abundantly on the 400-ft. and 500-ft. levels. The tetrahedrite has been found in a few narrow veins, about 1 in. (25.4 mm.) wide, on the 800-ft. level. Chalcocite and covellite are only important as ore minerals above the 500-ft. level. Bornite is rare and has been observed in but one part of the mine—the 35 block on the 800-ft. level. In the stope where bornite occurs, the mineral is associated with pyrite, chalcopyrite, and a little tetrahedrite. The bornite is concentrated in a small irregular body and at one time a few carloads of high-grade ore were shipped from this stope direct to the smelter.

Associated with the copper minerals are pyrite, sphalerite, scheelite, sericite, chlorite, calcite, quartz, specularite, epidote, gypsum, barite, and kaolin. The pyrite occurs intergrown with the chalcopyrite, or replaces the latite and andesite over large volumes of ground, which are almost barren of copper. The mineral appears to have had greater powers of migration than did the chalcopyrite, and so is found forming the inner wall of almost all of the wall orebodies and the peripheral or assay wall of the central orebodies. It is frequently found well crystallized, the common habit being a combination of cube and pyritohedron.

Hematite, variety specularite, occurs in both latite and andesite, but particularly in the latite above the 600-ft. level. Associated are abundant quartz and lesser amounts of chlorite, chalcopyrite, and pyrite. The specularite is found not only along the walls (the Luna wall in particular), but is also found along the fractures of the Luna fault zone. The huge pillars that form conspicuous croppings are parts of the hematite-

[4] The occurrence of ore in volcanic rocks close to an intrusive and resembling in many respects the contact metamorphic occurrences in limestone have seldom been described. One other similar deposit has been examined by the writer in the Copper River district of Alaska. The Alaskan occurrence on Clear Creek, a tributary of the Kuskulana river, is in the Nikolai greenstone and is characterized by chalcopyrite associated with pyrite with a gangue of andradite garnet, calcite, epidote, and magnetite. The intrusives are quartz monzonite as a stock, and dikes of quartz porphyry.

bearing ground and perhaps owe their endurance to their superior hardness due, in part, to the silicification of the latite. The specularite is found not only at the surface, but also in the deepest workings. Since this mineral can be traced from the surface downwards and since wherever found the relationships remain the same, the specularite is held to be entirely primary in origin.

Sphalerite is but sparingly found and is rarely observed associated with ore. The mineral occurs not only at the surface but also in one of the central orebodies on the 1000-ft. level. In the zone of secondary enrichment, it is replaced by covellite and chalcocite. Chlorite, although found in the latite, is more abundantly developed in the andesite and is a characteristic primary alteration product of this rock. Both calcite and quartz are abundant throughout the mine, with calcite seeming to prefer the andesite to the latite. The mineral also occurs in the ore-bearing monzonite.

Barite is a minor mineral and is observed as a rule in small vugs. On the 400-ft. level it occurs intimately intergrown with the chalcopyrite. Minute well-formed crystals have been observed on the 1000-ft. level. Epidote and gypsum are of rare occurrence; both are found on the 1200-ft. level in the sheeted monzonite.

The presence of scheelite in copper mines has been rarely described. In the Pilares mine the mineral seems to prefer the wall orebodies. The mineral occurs massive and in crystals showing the first and second order pyramids of the tetragonal system. The mineral has been found most abundantly along the Santa Cruz wall of the oval. With the scheelite are associated quartz, pyrite, chalcopyrite, specularite, and sericite. Some of the best primary ore of the mine occurs in ground carrying a noticeable amount of scheelite. Wherever found the mineral appears to have been one of the first minerals developed.

*Distribution of Ore.*—The ore of the mine consists of chalcopyrite and chalcocite, the latter mineral being important as an ore mineral above the 400-ft. level. The sulfides forming orebodies are disposed in two ways: (1) as a more or less continuous zone of ore about the walls of the oval, a body resulting that varies from almost nothing to 300 ft. (91 m.) in width; (2) as large plum-shaped bodies within the more central portion of the oval. In either case, the localization of ore is due to fracturing; aside from differences in rock texture, the more abundant the fracturing, the more abundant the ore.

Inasmuch as fracturing has been, to a large extent, the chief cause for the localization of the ore, the best orebodies of the wall should be found where the fracturing was most complete. As is shown in Fig. 4, the heaviest fault zones determining the walls occur at the Pilares and Esperanza ends of the mine; at these ends, large orebodies have been developed. Along other parts of the periphery, bodies of ore have been

worked but all are smaller than the two orebodies at opposite ends of the oval.

The Esperanza orebody measured approximately 1000 ft. (304 m.) long, 300 ft. (91 m.) wide, and has been mined for a depth of 800 ft. (243 m.). How much of the orebody has been eroded cannot, of course, be stated. The mining of this orebody began under the capping and extended to the 900-ft. level. In its upper portions, secondary enrichment increased the copper content, but this process becomes negligible below the 300-ft. level. As far as values go, some of the stopes of primary ore (6 to 10 per cent. in grade) are as rich as those in which the copper content was increased by secondary enrichment. In depth, pyrite becomes more abundant and the size of the minable orebody becomes smaller and smaller. Looked at in a broad way, this orebody is a part of a huge lens, the apparent width of which has resulted, in part, from duplication by faulting.

At the Pilares end of the mine, the strongly fractured ground carries an orebody the greatest length of which is about 600 ft., and greatest width about 300 ft. The ore has been developed from near the surface to the 1400-ft. level. Below the 500-ft. level, the orebody narrows markedly. About the walls are other orebodies lenslike in shape. On the 400-ft. level, these lenses come together, so that the entire periphery of the mine is mineralized, giving a truly remarkable body of ore as far as size is concerned.

All of these wall orebodies show both horizontal and vertical alterations in grade. As seen in horizontal section, the ore begins at the barren wall and shades off into an assay wall of pyrite toward the center of the oval. In a vertical direction, the different cuts taken in stoping vary in grade, a cut of 2.5 per cent. ore being followed by 4 per cent. ore.

The central orebodies, discoveries of recent years, occur in an irregular manner within the oval. Although in the outcrop the pillars that have made the mine a local landmark occupy the center of the oval, all exploration directly beneath the pillars, prior to 1915, had failed to reveal a body of sulfide sufficiently concentrated to be termed ore, and as a result it was thought that the ground beneath the pillars was barren. As work was extended in depth, as haulage ways were laid out through the center of the oval, more and more sulfide was found and the large orebodies of the central part of the oval were, at last, discovered.[5] These orebodies are localized in zones of shearing and, as is shown in Fig. 4, the orebodies are of relatively large dimensions, but extremely irregular in outline. It seems that wherever fracturing was intense and conditions of temperature, concentration, and pressure right, the ore began depositing, not along any particular fracture, but spreading from a number of

[5] In 1911, John M. Boutwell advocated prospecting these pillars in depth.

fractures out over a large volume, so that the central orebodies have no defined wall, but shade off into sulfide, which is at present of too low grade to be mined.

As is to be expected, there are wide vertical and horizontal variations in the grade of the ore, to say nothing of horses of barren rock that must be sorted. The barren rock resembles, in part, huge volcanic bombs, of which the peripheral part is layered as the skins of an onion.

*Croppings.*—The superficial indications of ore at Pilares are rather marked. On approaching Pilares Mountain by the mine railway, the eye is attracted by the reddish-brown knobs that rise conspicuously

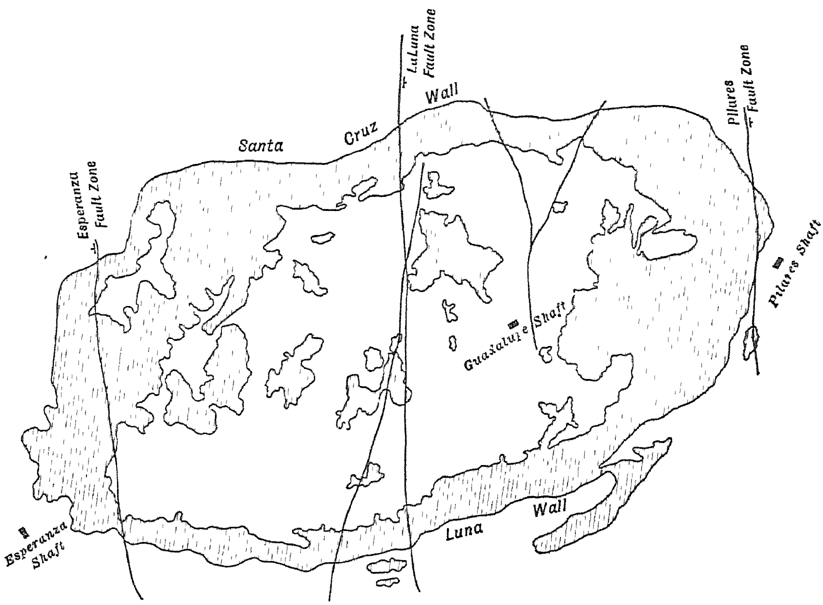


FIG. 4 —PILARES OREBODIES PROJECTED ON ONE HORIZONTAL PLANE.

above a red stained area, which stands in contrast to the gray of the surrounding latite. The knobs or pillars attain fully 50 ft. (15 m.) in section and may have a height of about 30 ft. (9 m.) An examination discloses that these pillars contain specularite, quartz, sphalerite, chalcopyrite and cuprite. In addition to the stains of limonite and manganese oxide they are smeared with malachite. They represent, therefore, excellent indicators of ore-bearing solutions and it is not to be wondered that the early prospectors gouged holes into these promising looking croppings. Today one of the main fill holes of the mine has cut away about half of one of these pillares and the exposure resulting is no more promising than were the gougings of the prospector. The inside of the pillar is light, almost white in color, carries specularite, kaolin, and quartz and is almost barren of even a trace of copper. Even after ore had been

discovered on the property, faith in the pillars was not lost, as is shown by the tunnels driven to cut the pillars in depth, but instead of copper, specularite and quartz were discovered. At last the pillar ground of the mine was given up as being unprofitable and development work continued around the walls of the oval. Eventually haulage drives had to be laid out through the center of the oval and one of these on the 600-ft. level disclosed ore. Development work showed a big orebody and a stope of 85,000 tons of 3 per cent. ore has been opened in the pillar ground. On the 700-ft., 800-ft., 900-ft., and 1200-ft. levels, good-sized stopes (3.5 and 4 per cent. in grade) have been opened beneath the conspicuous pillar croppings. At Pilares, therefore, we have a case of ore occurring beneath a cropping that has every appearance of being above ore-bearing ground, but the ore was discovered at a depth that long ago discouraged the prospector.

The croppings above the wall orebodies stand in contrast to the pillar croppings. The walls are marked by a slight depression, instead of an elevation. The glory holes in which the fill is quarried give excellent sections of the wall croppings. On the surface, these croppings have a reddish brown color and show a distinct limonitization, which in several instances becomes massive. All traces of copper are absent for a depth of 50 ft. (15 m.) where a little malachite becomes visible. The malachite gives way in a few feet to the zone of secondary enrichment which, although strongly marked for about 50 ft., gradually shades off into the primary sulfides.

A third type of cropping is that of the barren pyrite variety. These croppings, as in many other parts of the Southwest, are light red in color and give way in a few feet, and frequently almost at the surface, to pyrite.

*Secondary Enrichment.*—The zone of secondary enrichment is relatively narrow and shallow. Along the walls, both pyrite and chalcopyrite may be entirely replaced by chalcocite for a depth of between 100 and 200 ft. (30 and 60 m.). Below the 200-ft. level, the amount of enrichment is almost negligible. Along shear zones, which offered channel ways for solutions, the chalcopyrite is frequently tarnished down to the 900-ft. level. It may be of interest to note that the bornite occurrence on the 800-ft. level, even though in a fractured zone, shows no evidence of chalcocite enrichment, whereas the associated chalcopyrite is frequently well tarnished. On the 500-ft. level, crystalline native copper in a dendritic pattern occurs in seams in the latite. Associated are chalcocite, kaolin, and limonite, all apparently secondary in origin.

From records of the mine it appears that water was first encountered at a depth of 400 ft. The main haulage tunnel, at a depth of 700 ft., encountered abundant water when it was being driven, but since that time this tunnel has practically drained dry the overlying country. When

the winze of the Guadalupe shaft was being sunk, water at one time stood about 4 ft. below the collar of this winze. After work in the mine had progressed to the 1200-ft. level it became necessary, in the revolutionary days, to close down the mine for two months and water rose in both the Pilares and Esperanza shafts to within a few feet of the 1000-ft. level.[6] A churn drill hole on the Esperanza side of the mine and about ½ mi. (0.8 km.) distant from the oval, and a prospect shaft on the Pilares side of the mine and about ½ mi. distant from the oval, both encountered standing water within 100 ft. of depth. At present the mine makes about 50 gal. (189 l.) of water per minute and as depth is gained the upper levels become dry. It seems, therefore, probable that the original water level stood at 400 ft., a depth which limits the zone of important secondary enrichment, and that the water level has constantly been lowered by pumping.

*Summary.*—The Pilares mine is of the pyrite-chalcopyrite type occurring in latite and andesite breccias perhaps of early Tertiary age. The ores are genetically related to monzonitic intrusives. In many respects the occurrence of the ore resembles that of the contact metamorphic deposits in limestone. The oval outline of the cylindroid mass of mineralized ground is due to the intersection of easterly and northwesterly fault planes. The chief ore mineral is chalcopyrite; bornite is but sparingly found. The gangue is characterized by specularite, which is held to be primary, scheelite, chlorite, calcite, epidote, sphalerite, sericite, gypsum, and pyrite. Enrichment of commercial importance is confined to the upper 200 ft. of the workings. The cropping is conspicuous because of several large hematite-bearing and limonite-stained pillars that carry but a trace of copper and overlie important orebodies, which do not extend much above the 500-ft. level.

## Mining Methods

### *Equipment*

The Pilares mine is connected with the mill at Nacozari by a 5-mi. (8 km.) steam railroad and the 5000-ft. (1524-m.) Porvenir tunnel. The railroad gage is 36 in. (91.4 cm.) and the track is laid with 80-lb. (36.287-kg.) rails. The ore train is made up at the switching yards at Porvenir and turned over to the steam locomotive at this point. The railroad transports 75,000 tons of ore to the mill per month in 25-ton bottom dump cars. The ore is delivered to the railroad cars in the Porvenir tunnel from large chutes made by widening raises to 15 by 15 ft. with the bottom of these raises, just above the tunnel, enlarged into the main pockets. These raises run up to the upper levels and,

[6] Statement by Bert Hore, Mine Foreman.

by inclines cut into them, ore can be dumped in at each level. The openings where the mine cars dump into these chutes are covered by automatically swinging steel doors to prevent the dust returning into the mine workings. There are ten of these railroad chutes and their combined capacity is around 20,000 tons. The ore below the tunnel level is hoisted to the railroad pockets.

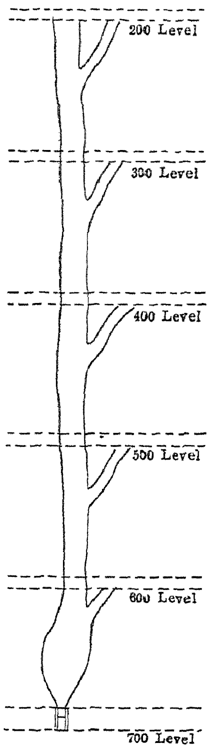

FIG. 5.—ILLUSTRATING RAILROAD ORE CHUTES.

The mine has been opened to the 1500-ft. level by three shafts. At the north end of the oval in which the ore occurs there is the three-compartment Esperanza shaft. The compartments are 4.5 by 5 ft., using three cages; this shaft handles material and men to the Esperanza and Guadalupe divisions. At the Pilares, or south end of the oval, is the three-compartment Pilares shaft with three cages, which handle men and supplies to the Santa Cruz and Pilares divisions. The Guadalupe shaft, which is the main ore-hoisting shaft, is situated in the center of the oval and extends only to the 500-ft. level.

At Esperanza, a double-drum, Nordberg electric hoist with a 22-ton flywheel generator set handles two cages. These cages have a height of 9 ft., so that long timber can be loaded on them without putting up the bonnet.

At Pilares, a new five-compartment shaft is being sunk 400 ft. beyond the old shaft, as the old shaft is in ore on the lower levels. This new shaft will have one big cage, with a 5 ft. 8 in. by 9 ft. deck, and two standard cages in the 4.5 by 5 ft. compartments. The fourth compartment will be used for a counter-weight for the big cage, and for pipe lines, while the fifth will be used for a manway. The shaft was opened as follows: drifts were run out from each level and 4 by 6 ft. raises run up; then, starting on every second level the shaft was widened and timbered up full size, one compartment being lined inside and used as a chute to handle the broken rock. The shaft timber is 12 by 12 in. for the wall plates, 8 by 12 in. for the main dividers, and 6 by 12 in. for the sub-dividers. All timber is California redwood, because of its greater resistance to decay and fire.

The main ore-hoisting shaft is the Guadalupe, situated in the center of the oval. This shaft uses skips of 2.5 tons capacity in two compartments, and has a third compartment for a manway. The ore is hoisted from pockets and the skips are charged by means of a loading cartridge, which holds exactly one skipload. The dumping of the skips is so ar-

ranged, by means of a door, that the ore is deflected to the mine-railroad ore chutes, while by throwing the door the other way, waste can be hoisted and dumped into a separate chute. This door is on the 550-ft. level, but by means of an air valve it can be thrown either way by the skip tender at the 1000-ft. or 1200-ft. levels. The door is connected with white and red electric lights, so that when it is set for ore, it shows a white light at every level; and when set for waste, a red light. The electric device is very simple and positive and shows the skip tender just how the door is set.

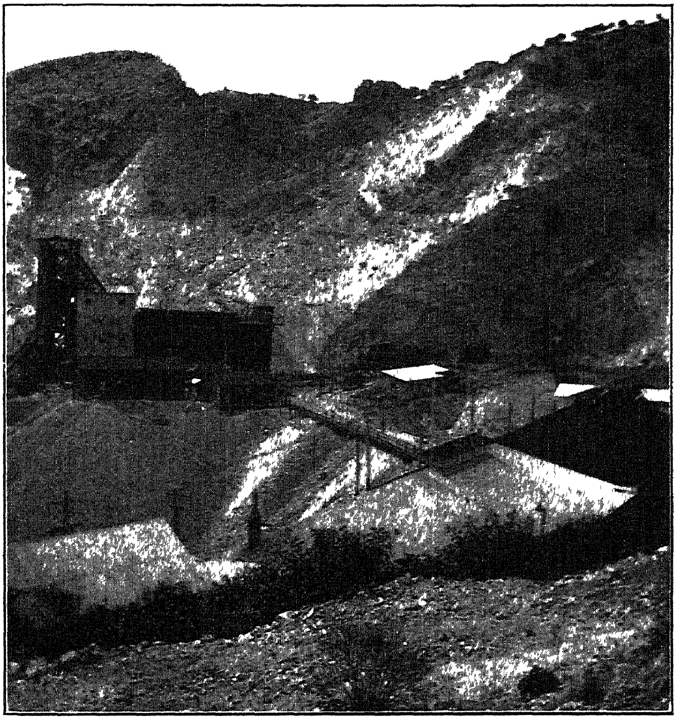

FIG. 6.—PILARES SHAFT; OUTCROP AT TOP OF MOUNTAIN IS ONE OF PILLARS FROM WHICH MINE IS NAMED.

Tramming is done by both hand and electricity. Hand tramming cars have a capacity of 22 cu. ft. (0.623 cu. m.) and hold approximately 1 ton. They are equipped with Hyatt roller bearings. The electric tram cars are side rocker dump, of 1.5 ton capacity, with Hyatt roller-bearing wheels. Five-ton bar frame Westinghouse electric locomotives are used, operating at 260 volts from an overhead trolley wire. The standard track gage on all levels, except the 700-ft. main railroad tunnel level, for both hand and electric tramming is 20 in. (50.8 cm.). Hand tramming tracks use 16-lb. (7.23 kg.) rails and electric tracks use 40-lb. (18.14-kg.) rails.

The mine employs about 1300 men underground and 300 men on the surface. All labor is Mexican. The only Americans are the division foremen, engineers, and heads of departments. All development work, mucking, and tramming is done by contract at so much per foot or per car. A considerable number of the stopes are also operated on contract. In drifting and cross-cutting, Leyner & Waugh 60 Dreadnought and Waugh Turbo machines are used, mounted on 8-ft. (2.43-m.) columns. The standard drive is 8.5 ft. by 5 ft. wide for machine work, and 7 ft. high

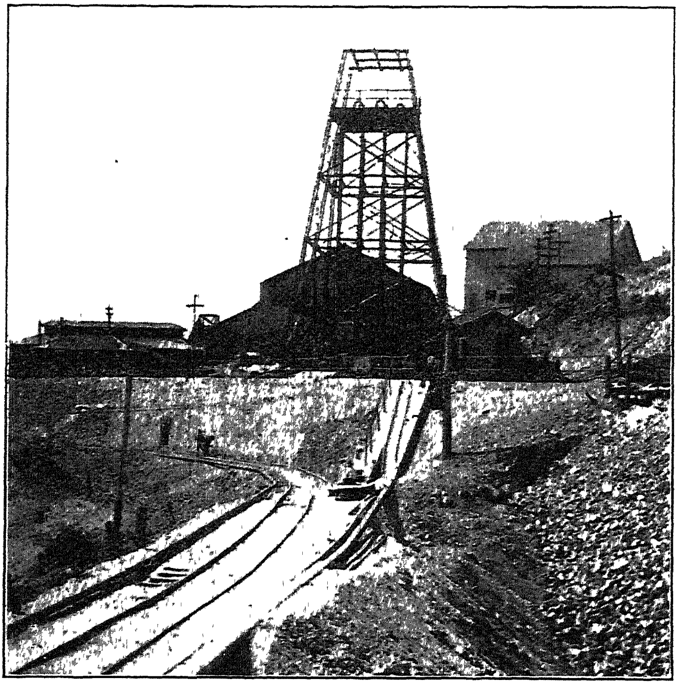

FIG. 7.—ESPERANZA SHAFT HEAD-FRAME WITH COMPRESSOR HOUSE AND HOIST HOUSE IN BACKGROUND.

by 5 ft. wide for hand work. The standard drift round is shown in the illustration. Stoping is done with 16-V Waugh and CC-11 Ingersoll-Rand stopers. The men running stoping drills are paid by the foot of drill hole put in per shift.

*Stoping Methods*

We have in operation the following methods of stoping orebodies: Hard ore in ground that stands exceptionally well is mined by the flat cut-and-fill method and an incline shrinkage-and-fill method.

*Cut-and-fill Method.*—The flat cut-and-fill method, shown in Fig. 8, has been the standard method of Pilares for several years and is the method

that, at present, produces the largest percentage of the ore. The ground stands exceptionally well in the majority of the orebodies. Some of these flat cut-and-fill stopes often reach very large dimensions, as, for example, the 646 stope, which is 60 ft. (18.28 m.) wide by 600 ft. (182.88 m.) long. The standard practice is to sill such a stope on the level, then take one

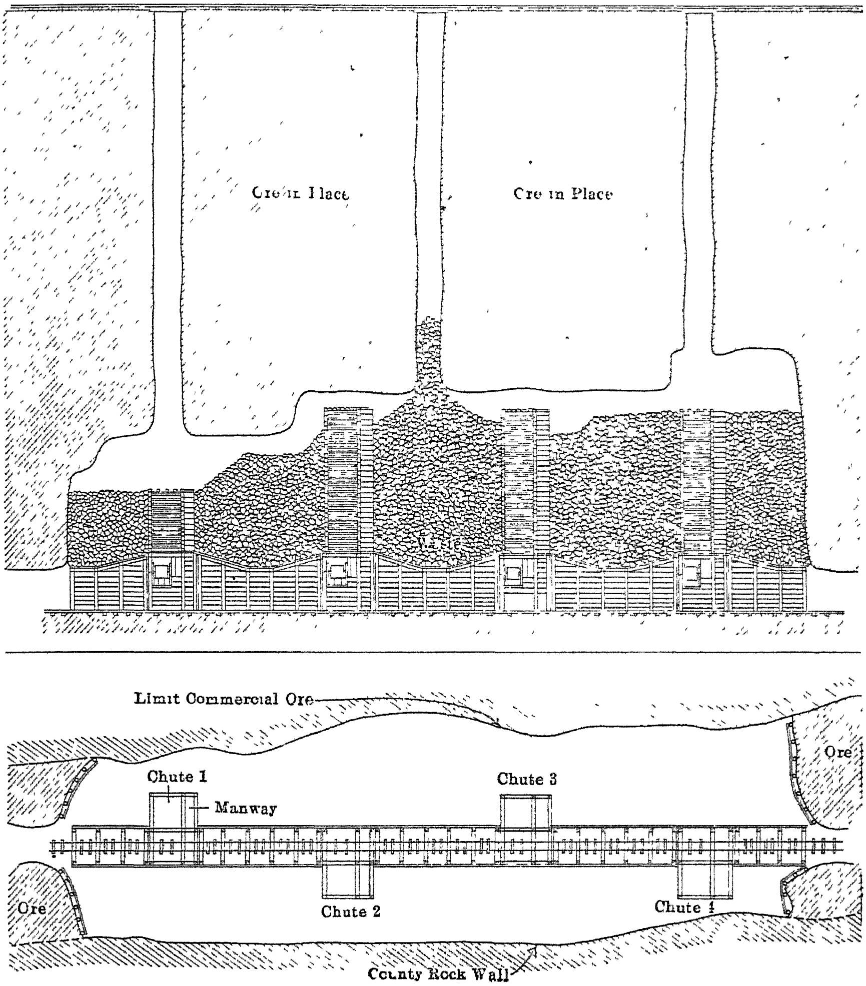


FIG. 8.—FLAT CUT-AND-FILL STOPE.

7-ft. (2.13-m.) cut off the back, then run up fill hole raises 50 ft. (15.24 m.) apart along the center line of the stope to the level above, stand a timber line of 10 by 10 in (25.4 cm.) sawed timber, making the drift 5 by 8 ft. (1.52 by 2.438 m.) in the clear; double lag over the caps and single lag on the sides. Chutes with manways are provided every 30 ft. (9.144 m.). These

chutes are 4 ft 6 in. square (1.37 m.) inside measurement, and are lined with 2 by 10 in. (5 by 25.4 cm.) plank. The chutes have arc doors 30 by 30 in. (76.2 by 76.2 cm.) in size. Waste is now poured down the fill holes covering the timber line and spread level. This filling work is done on contract by the cubic foot, the engineers first measuring the space to be filled and computing the volume. The cut is then started at one end of the stope around a fill hole and carried 7 ft. (2.13 m.) high the length of the stope. When the first cut has advanced part way down the stope,

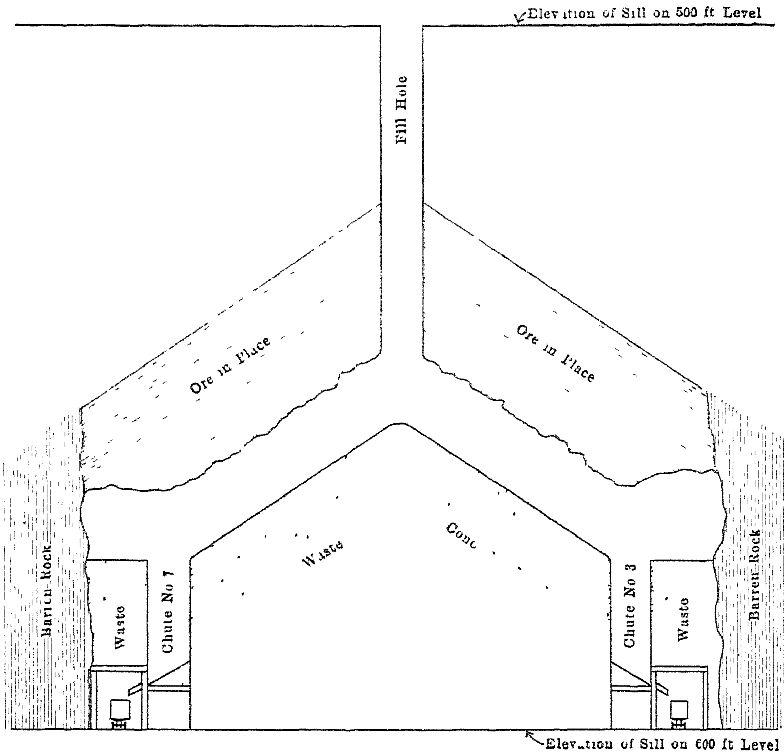

FIG. 9.—INCLINE SHRINKAGE WITH INCLINE FILL STOPE.

fill is poured in the end fill hole and spread and another cut at the next elevation is started behind the first cut. A long stope will have two to three cuts going on at once. Cribs of 6 in. by 6 in. by 6 ft. timber are built up to support the back at all points that appear heavy. Most of the timber in these cribs is saved and used over again.

This flat cut-and-fill method is a cheap method and well adapted to the nature of the deposits, as a very considerable part of the ore contains large waste boulders. These are more easily sorted out and disposed of in a flat cut-and-fill stope than in an incline shrinkage-with-fill stope. However, the incline shrinkage-and-fill stopes are proving to be cheaper

by about 15 c. per ton and new stopes are being laid out to run on this system, providing they are in a block of ground in which the ore is not too badly mixed up with waste boulders and the back is sufficiently solid to be safe.

*Incline Shrinkage and Fill.*—When ore is developed it is silled out on the level until the boundaries of the orebody are reached. In the case of the 624 stope this body was oval in shape, being 160 by 120 ft. (48.76 by 36.57 m.) for the longer and shorter diameters, respectively. After

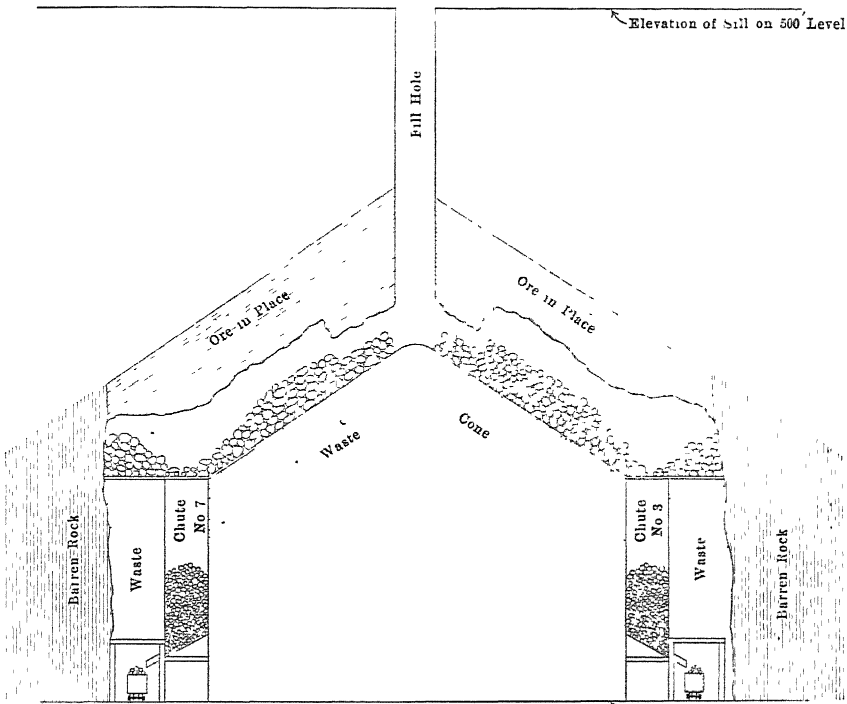


FIG. 10.—INCLINE SHRINKAGE WITH INCLINE FILL STOPE.

working out the sill, a cut 7 ft. high was taken over the back and two fill-hole raises, located near the center of the stope, were run to the level above. The object of using two raises instead of one was to locate each raise at the foci of the ellipse, so that the waste would spread itself naturally into an elliptical, instead of a circular cone, and thereby more nearly assume the shape of the orebody. A timber line was then stood, so arranged that the chutes came around the outer walls of the stope. These chutes (fourteen in number) were about 25 ft. (7.6 m.) on centers. The timber line was made of sawed timber 10 in. (25.4 cm.) square, and lagged with 2-in. plank. The posts were 8 ft. long and the caps 7 ft., giving 5 by 8 ft. in the clear. Waste was poured down the two central fill holes

and spread level around the timber line. After leveling, the ore was broken around the two fill holes, taking off 7-ft. cuts, slicing upward to make a cone. The men stood on the broken ore and mined up each time until the cone was the full size of the stope and ran up at a 35° angle, after which the ore was mucked into the chutes. These chutes were the standard mine chutes, being 5 ft. (1.5 m.) square and covered with grizzly bars made of 8 by 8 in. sawed timber, spaced with 12 in. (30.4 cm.) open-

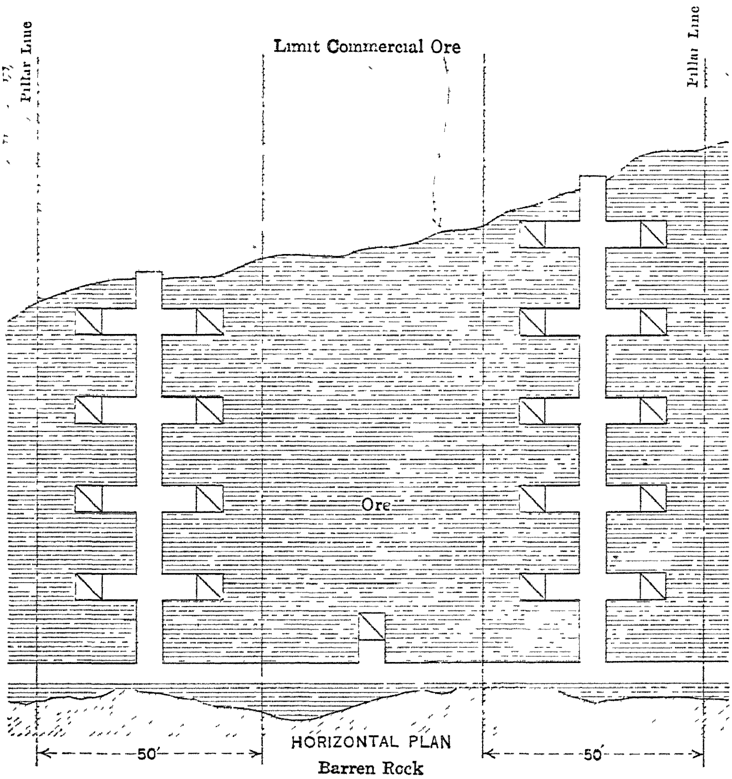


FIG. 11.—SHRINKAGE STOPE SHOWING SHOVELWAYS.

ings. Fill was now dropped through the fill holes until it spread out in a natural, elliptical shaped cone and its edge reached to the chutes. A short floor of 2-in. plank was laid next the chutes and extended 6 ft. up the cone to assist in mucking the ore on the next cut. Mining now started up all around the bottom of the cone, using stoping machines. The ore was rapidly broken up the slope to the two central fill holes, using drill holes 8 ft. long. Just enough ore was now mucked into the chutes to allow another cut to go up the cone. This was repeated until four cuts had been taken. Then the stope was mucked clean, the chutes raised 20 ft. (6.09 m.) and enough fill put in to again reach the back. There

was a slight amount of fill that had to be hand spread between the chutes and the outer walls of the orebody. The accompanying illustrations show two stages of operating this stope.

The advantages of this method are: several cuts are taken before filling; 20 ft. of ore is broken all over the stope, so that no floor is needed over the waste fill, the bottom of the ore only being raked down over the waste cone once. If fill were put in after each cut, too much waste would get mixed with the ore; if the cone were floored with plank, the extreme hardness of the Pilares ore, and the large pieces weighing 3 or 5 tons into which it breaks, would make it practically impossible to maintain

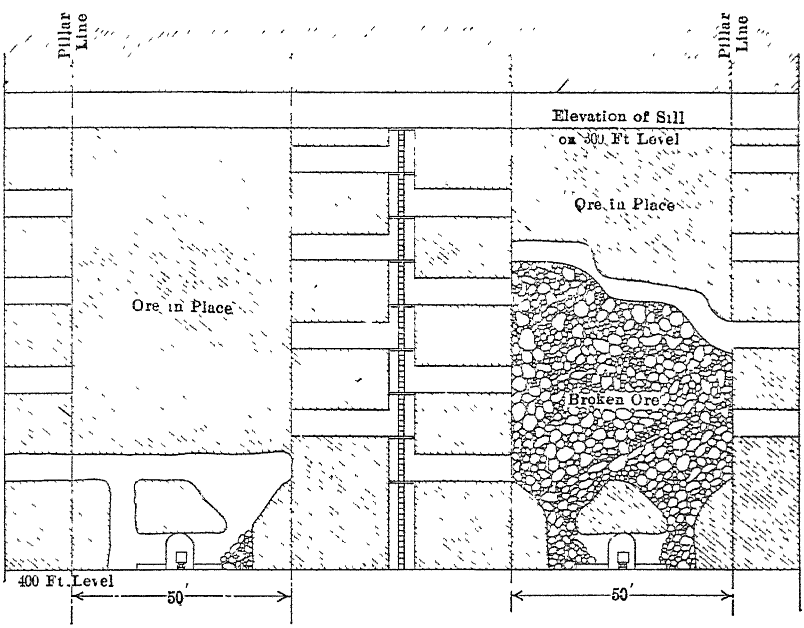


FIG. 12.—SECTION THROUGH SHRINKAGE STOPE.

the floor without great breakage. The expense of installing a floor over a large cone for every 7-ft. cut would make quite an addition in the stoping cost. We figure that the amount of fines lost on top of the waste fill is negligible because the waste cone is scraped down very carefully, and 1 or 2 in. of waste filling mixed in with 20 ft. of broken ore has no appreciable effect in lowering the grade. As the cone of such a stope is only cleaned down five times during 100 ft. of stoping, it is possible to do this work carefully without adding too much expense to the cost per ton of ore distributed over the life of the stope. A stope of this size run on this method in the extremely hard ore of the Pilares mine will produce 9000 tons per month.

*Shrinkage Stopes.*—During the earlier days, the ore was mined in

large shrinkage stopes. In some of the larger orebodies they were laid out across the orebody, generally 50 ft. (15.24 m.) wide and 100 to 200 ft. (30.48 to 60.96 m.) in length. A 50-ft. pillar was then left and another stope run up and so on. Raises were put up in the pillars for manways and crosscuts were run to the stopes at 20-ft. intervals.

Due to the exceeding hardness of the ore and the fact that it broke into big boulders, it was impossible to draw it through ordinary chutes; therefore, an arch was left over the drift and shovelways put in. These consisted of crosscuts about 12 ft. long and a short raise with a funnel-shaped top opening into the bottom of the stope No chute was used. The ore was loaded into the cars by shoveling. Large boulders were drilled and blasted as they appeared in the shovelway. Steel sheets were used on the bottom of the shovelway and the floor of the shovelway was a couple of feet higher than the track, so that the shoveler would not have to raise the ore so high in loading a car. One man would load 25 tons per shift from a shovelway. Sometimes instead of short, funnel-shaped raises, the stope was silled out at the back of the shovelway and a triangular pillar left over the drift. These shrinkage stopes were drawn after all the ore was broken and then filled with waste from the surface. In the earlier history of the mine, these stopes could be drawn very successfully; but in late years, due to the extensive stoping, the ground has been so weakened that such stopes cannot be drawn without caving the areas around and above them. This caving causes a serious mixture of the wall rocks with the ore. Some of these old shrinkage stopes were of immense length, as in the Martinez, Evans, Dominguez, Sol, and Luna, which were practically one continuous stope 2000 ft. long, with widths varying up to 60 feet.

*Top Slicing Flat.*—In the case of several of these shrinkage stopes, the ore cannot be successfully drawn and it has become necessary to top slice these broken reserves. This is done by putting up raises in the barren walls around the old shrinkage stope. These raises are made in two compartments: one side being used as an ore chute and the other as a manway. If there is a floor pillar left above the old shrinkage stope, this is taken out, either cut-and-fill or, if the ground is too heavy, square setted. Top slicing is then started under a plank floor, which is laid on top of the broken reserves of the old shrinkage stope, before cut and filling or square setting out the floor pillar above.

In the case of the 310 stope, we had the following problems: This stope had come up from the 400-ft. to the 300-ft. level as a shrinkage stope. The back was virgin ore through which two raises had been put up to the 200-ft. level. On the 200-ft. level, a shrinkage stope had been run up to the surface and drawn, after which it was partly filled with surface waste. The shovelway arches on the 200-ft. level constituted a considerable tonnage of good-grade ore.

When an attempt was made to draw the broken ore that filled this stope from the 400-ft. level to the 300-ft. level, the back began to cave in big blocks, weighing up to 2000 or 3000 tons, and the level above began to crack. The old shrinkage stope was 140 ft. long by 80 ft. wide.

We could not fill the big hole between the top of the broken ore, which had by this time been drawn to a point about 20 ft. below the 300-ft. level, and the back with waste, as this would have mixed with the broken ore, so we filled the hole with ore mined elsewhere on the 200-ft. level, dropping it down the two raises already mentioned, until we had enough filling to reach the back. This was then shot down until solid

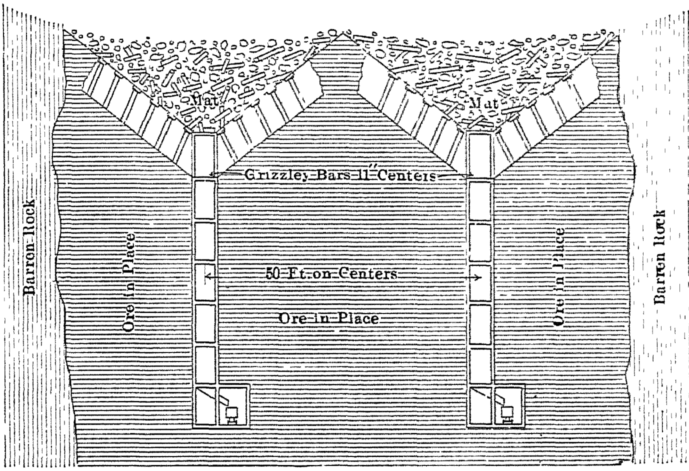


FIG. 13.—INCLINE TOP SLICE WITH SQUARE-SET TIMBER IN SLOT USED IN 273 STOPE

and caught up on bulkheads. We then sunk two-compartment cribbed winzes through the broken ore to serve as ore chutes and manways to the 400-ft. level, laid a double floor on the broken ore, and stood a timber line on it. We are now taking out the ore between the 300-ft. and the 200-ft. levels by the cut-and-fill method and will square-set the old shovelway arches when this stope gets to the 200-ft. level, after which we will top slice the broken reserves under the wooden floor between the 300-ft. and 400-ft. levels.

We have top sliced out a couple of large blocks of broken reserves and do not experience any more difficulty than in top slicing new ground. We carried these slices 9 ft. high in the past, but are now experimenting with 11-ft. slices.

*Inclined Top Slice.*—We are opening up one inclined top slice on a block of soft ore, that comes up to within 35 ft. of the surface. This inclined top slice is operated on the same plan as that described by

W. G. Scott,[7] except that instead of running a shrinkage stope for a chute, we run up a single line of square sets and lag four sets for chutes and use the fifth as a manway. The square-set posts are 9 ft. 2 in. long, so as to give a 10-ft. slice.

After dropping a floor, we draw enough ore from the row of square sets to start opening the next floor below; this gives a self-mucking top slice and a very rapid producer. Moreover, the first set of posts to catch up the mat are already in place. We recover the plank used in lagging the square sets and use the same for flooring in the next slice.

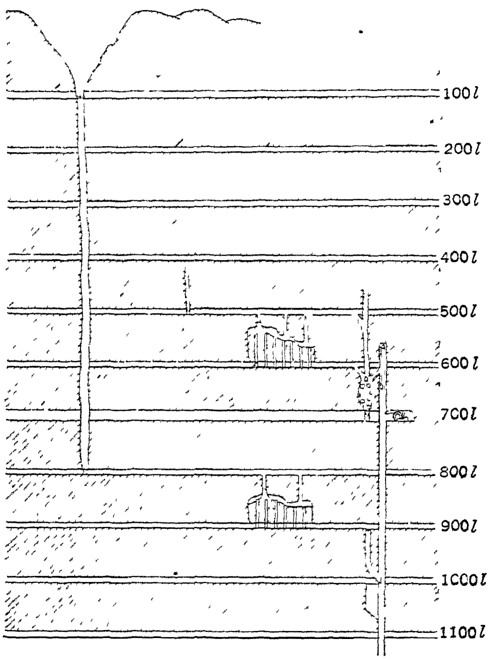


FIG. 14.—DIAGRAMMATIC REPRESENTATION OF GENERAL SYSTEM USED AT PILARES MINE.

*Square-set Stopes.*—Most of the ore stands so well that square setting is not necessary; however, when a cut-and-fill stope approaches the bottom of a mined-out block above, the stope has to be changed over to square sets. The chutes in the cut-and-fill stope are not always so arranged as to be readily joined with the chutes for the square sets, although cut-and-fill stopes that have been started recently have their chutes placed with this in view. A good many pillars between old stopes are mined by square setting so that, all told, over 20 per cent. of the mine tonnage comes from stopes operated by this method.

[7] *Trans.* (1918) **59**, 305.

The standard square set at Pilares is 5 ft. square and 8 ft. high, and because the ore breaks in large rough pieces and because of the weight in some of our square-set stopes, the standard post is either 10 by 10-in. sawed timber, or round timber 10 to 12 in. in diameter; the latter is preferred as it is stronger and cheaper.

The practice has been adopted lately of making the sill floor of a square-set stope, when it starts on the level, 9 ft. 2 in. (2.79 m ) high under the caps. This enables us, when crushing of the sill floor timbers takes place, to double up the posts and caps and still leave room for the cars to pass under the chute mouths. I might add here that the Pilares hand-trammed ore cars are exceptionally high from the rail to the top of the body.

We contract all square-set stoping and use two systems of settlement: one by the cars of ore pulled from the chutes and the other by the cubic contents mined, that is, by the square set put in place. We find one square set averages 16.5 tons of ore. Paying a contractor by the set mined has proved the better method, as there is less chance for error than by counting the cars from the chutes.

*Filling Stopes.*—The underground fill for our stopes is obtained from the surface. Raises are run up from the lower levels of the mine to the top of the Pilares Mountain. About 300 ft. below the top of the mountain these raises have an offset which is covered by a grizzly. The raises are made 15 by 15 ft. (4.57 by 4.57 m.) below the grizzly and 20 by 20 ft. (6.09 by 6.09 ft.) above the grizzly. Some of these fill holes have more than one grizzly, as in the case of fill hole No. 14, which has three grizzlies placed around the three sides of the raise and the lower raise below the grizzly has three branches run up to connect with these three grizzlies. The grizzlies are made with 18-in. (45.7-cm.) openings and are built out of 10 by 20 in. (25.4 by 50.8 cm.) timber, covered with 0.5-in. (12.7-cm.) steel boiler plate and crossbars of 60-lb. (27.21 kg.) rails laid across these timbers at right angles.

The tops of the fill holes are opened to great funnels, sometimes 100 to 225 ft. (30.4 to 68.5 m.) in diameter. Around the sides of these funnels 30-ft. to 35-ft. holes are drilled and sprung several times, then loaded with black powder; 20,000 to 30,000 tons of rock are thus broken very cheaply at one blast and descend down the 20 by 20-ft. raises to the grizzlies, where the larger pieces are blockholed. One of these fill holes can furnish about 600 tons of waste per 24 hr. Chutes are placed in the raise below the grizzly on each level in the mine and the waste is drawn out and transferred to the stopes as needed. All development waste is also used in the mine to fill stopes, but development waste itself constitutes only a very small percentage of the total fill necessary.

# Tunnel Driving at Copper Mountain, B. C.

BY OSCAR LACHMUND, E M., SPOKANE, WASH.

(Chicago Meeting, September, 1919)

DURING the driving of the main haulage level at the Copper Mountain mines of the Canada Copper Corpn., Ltd., near Princeton, B. C., some very rapid driving was done, though no claim for a world's record is made. Conditions, however, were unfavorable for economical operations. The cost of power was high, for the fuel was of poor grade; besides, during the time the work was in progress, very little other power was needed so that most of the power cost was charged against the footage. The

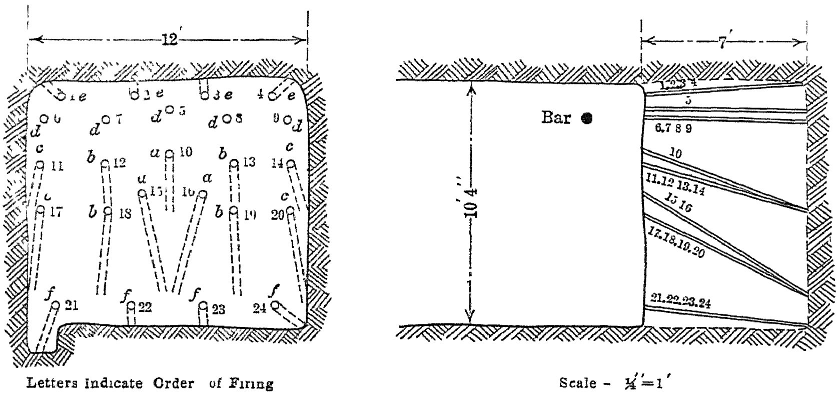


FIG. 1.—DRIFT ROUND USED IN MAIN HAULAGE LEVEL, COPPER MOUNTAIN, B. C.

transmission line consisted of No. 4 galvanized iron wire with the result that the line loss was considerable. The voltage transmitted was about 30,000. The plant was operated under a lease, which was due to expire about the same time this work was supposed to be completed; an extension was refused, therefore speed was most important.

The plans called for a straight adit 2900 ft. (884 m.) in length. At a point 2800 ft. (853 m.) from the portal, two raises were to be put up to the next nearest workings, a difference in elevation of about 800 ft. (243 m.). One of these was to be a two-compartment hoistway and the other a zigzag ore pass, or muck run. A location for these raises had been determined by a number of diamond-drill holes, but the material to be penetrated by the adit was not known. It seemed imperative to get the tunnel work completed as rapidly as possible, in

order to allow for delays in the raising program, which were certain to occur. To show how closely these operations were timed, it is interesting to note that the date of expiration of the power-plant lease was Sept. 1, 1918, and the last round, making the connection between the upper and lower workings, "broke through" in the night of Sept. 3, 1918

The plans called for a tunnel 9 ft. (2.7 m ) high by 11 ft. (3.3 m.) wide; but owing to the "blocky" nature of some of the rock a considerable "over break" occurred. This enlarged the tunnel cross-section to 10.4 ft. by 12 ft. (3.2 by 3.6 m.) indicated by measurements taken at 200-ft. (60-m.) intervals after the work was finished and slowed up the work on account of the extra waste handled, besides increasing the cost per foot of driving. Several regions of geological disturbance were crossed and the heavy ground encountered called for timber supports More than 350 ft. (106 m.) of heavy timbering was necessary at various points along the course of the tunnel; this also retarded the work to the extent of about 6 ft. (1.8 m.) per day for each set of timber placed. Once the working force was organized and the work well under way, three shifts were put on working 8 hr. each.

The drills used were the dreadnaught No. 60. They were mounted four on a horizontal bar, from which position all but the four bottom holes, or "lifters," were drilled, the miners working on the muck pile. Upon completion of the upper part of the round, most of the muck had been removed; that which was left was rapidly thrown back from the face, all hands helping on this work. The horizontal bar was then torn down and dropped to the lower position, from which the lifters were drilled. The change of the bar from the upper to the lower position, together with drilling the lifters, loading, and firing the entire round, was frequently made in 50 minutes.

The holes were pointed to pull a 7-ft. (2-m.) round and averaged about 9 ft. (2.7 m.) in depth. The center, or "cut holes," were fired first, after which followed the side holes, then the back holes, and finally the lifters. The drift round commonly used in this work is illustrated in Fig. 1, which also indicates the firing of the holes in groups. The blasting was done by hand, the fuses being "spit." The timing of the shots was regulated by cutting the fuse in different lengths; the shortest for the center holes, the next longest for the side holes, and so on. The lifters were loaded with extra heavy charges of powder, so as to throw the muck back from the face as much as possible. This was sometimes helped by placing charges of explosive outside and beneath the lifters; these were called muckers, and were set to go off after the rest of the round had been fired.

The powder used was a non-freezing kind, varying in strength from 40 to 60 per cent. nitroglycerine, depending on the hardness of the rock at the face.

The rock was handled in small, V-shaped, hand-dump cars of about 1000 lb. (453 kg.) capacity. Tramming was done by hand until the distance from heading to dump became too great, when horse haulage was substituted; later this was replaced by an electric installation. Steel plates were laid on the bottom for a distance of 30 to 40 ft. (9 to 12 m.) from the face, to facilitate shoveling, also to permit shunting empty cars past the loaded trains and thereby eliminating the need for double track.

The cars, being light, were easily pulled from the track and, with bodies tilted, were passed on the steel plates, alongside of the loaded cars and then pushed back on to the track at the muck pile and loaded. Temporary track was laid close up to the face before firing a round. The T-rails were laid on their side, allowing the flanges of the car wheels to run on the grooves thus formed.

The foul air and gases were removed, after each round was fired, by a Connersville rotary blower, of 10 cu. ft. (0.28 cu. m.) capacity, stationed at the portal of the tunnel. Later, a similar machine was placed about halfway in the adit and worked in tandem with it. The blowers were set to exhaust toward the surface through a 12-in. (30-cm.) wire-wound, wooden stave pipe. The men were able to return to the heading within 15 min. after firing.

The mucking crew was divided into three gangs, on each shift, averaging 11 men per shift. The work was divided so that one gang was shoveling muck, another was picking down from the muck pile, while the third was bringing up empty cars and forming them into trains after they were loaded. This latter work did not take up the entire time, so that this gang had an opportunity to rest. As soon as a train was loaded the gangs changed jobs; that is, the pickers went at shoveling, the car handlers took the picks, and the shovelers took the easy work, and so on. Greater efficiency was maintained in this manner, as the change of work tended to rest the men and they were able to work continuously.

A bonus system was also a large factor in keeping the men up to the mark. This was based on a daily advance of 9 ft. (2.7 m.), upon which the then "going" wages were guaranteed; for all advance over 9 ft., $6 per foot was added as bonus. For each set of timber placed, an allowance of 3 ft. (0.9 m.) was made, which applied on the bonus. Current wages at the time were $4.50 for miners, $4 for helpers, and $3.50 for common labor. The bonus distribution brought these amounts up to $5.91 for miners, $5.25 for helpers, and $4.59 for muckers. The foreman and the shift bosses also shared in the bonus, the distribution being made by pro-rating the bonus in the same ratio as the amount of regular wage received by each man. Everybody seemed satisfied and no difficulties were experienced as far as the labor situation was concerned.

The work was begun on Oct. 9, 1917, and the tunnel was finished

Mar. 11, 1918, a total of 154 days. The actual working time was 150 days, four days being lost on account of a break in the power line.

The length of the adit is 2903 ft. (884.8 m.) and the daily average progress was 19.3 ft. (5.8 m.) for each working day. The greatest advance in any one month was in December, 1917, when a total of 645 ft. (196 m.) was driven. The amount of rock handled is estimated at 185 tons per day. The material penetrated was granodiorite, for the greater part of the distance The total cost of driving the tunnel was $103,242.15, which brings the cost per foot of tunnel to $35.56. Certain equipment and supplies were charged against the work that should have been carried in a suspense account, as most of these had a certain salvage value because it was intended to use them in the future operation of the mines. For reasons already mentioned, such as expensive power, the cost given does not really represent the actual expense of driving. Had speed not been so important, no doubt the work could have been done more cheaply. A few of the cost items are as follows:

| | Totals | Unit Cost |
|---|---|---|
| Total Driving Cost | | |
| Labor | $25,517.81 | $8.80 |
| Explosives | 16,616 75 | 5.72 |
| Drills, parts and repairs | 2,767 82 | 0.95 |
| Steel, sharpening and replacement | 3,979.78 | 1.37 |
| Miscellaneous supplies | 1,908.27 | 0.66 |
| Power | 8,410 24 | 2.90 |
| | $59,200 67 | $20.40 |
| Rock Disposal | | |
| Labor | $21,843 40 | $7.52 |
| Supplies | 1,685.24 | 0 57 |
| Power | 558.64 | 0 20 |
| | $24,087.28 | $8.29 |
| Timbering | | |
| Labor | $1,335.52 | $0.46 |
| Timber and supplies | 2,513.18 | 0 86 |
| | $3,848.70 | $1 32 |
| Indirect Expense | | |
| Air and water lines | $5,439.09 | $1 88 |
| Electric lighting | 1,018.81 | 0 35 |
| Ventilation | 3,198.38 | 1.10 |
| Dump, tracks, and trestles | 503.67 | 0 17 |
| Depreciation on drills | 970.76 | 0.33 |
| Depreciation on cars | 301.82 | 0 11 |
| Surface hoisting and hauling | 3,464.58 | 1.19 |
| Miscellaneous supplies | 1,208 39 | 0.42 |
| | $16,105.50 | $5 55 |
| Total cost | $103,242.15 | $35.56 |

| | |
|---|---|
| Timbering details | |
| 53 sets timber installed, cost per set | $72 62 |
| 354 ft of tunnel timbered, cost per foot | 10 83 |
| Drilling Details | |
| Actual drilling hours. | 8022 |
| Actual working days | 149 $^{1}\!/\!_{2}$ |
| Average drilling hours per day | 53 50 |
| Cost of upkeep per drilling hour, in cents | 22 53 |

Before planning any of the work, the engineer in charge visited large mining properties in the West and Southwest, where all the latest methods were observed. From notes taken upon these trips, the layout for future work was developed. The method of driving the main haulage level was derived from the operation of the Pioneer Tunnel at Glacier, B. C., a few years ago, by the Canadian Pacific Railway.

The credit of working out the plans and details of this work belongs to Mr. F. S. Norcross, Jr., at that time superintendent of mines of the Canada Copper Corpn., now a captain in the 27th Regiment of Engineers, the mining regiment, at present in France. Captain Norcross was unable to complete the job, as he enlisted in December, 1917, but the work was carried out by his successor and former assistant engineer, Mr. P. E. Crane.

# Wedging Diamond-drill Holes

BY O. HALL,* M. SC., AND V. P. ROW,† CONISTON, ONT.

(Chicago Meeting, September, 1919)

DIAMOND drilling has become a very important factor in mining. It is the most satisfactory method of obtaining proof of the existence of an orebody and of determining the character and extent of the body in measures where a core can be made. Large investments are based on the information obtained. The method of development, the mine and reduction plants, and subsidiary essentials established are often without other information. The Mond Nickel Co. has drilled about 50,000 ft. (15,240 m.) per yr. for several years and has given considerable study to the question of improving the methods and of overcoming the limitations of this means of securing information

One of the main limitations of the diamond drill has been its inability to drill a straight hole. To depths of 1000 ft. (304 m.) the deflection is often not large but all holes tend to curve and take somewhat erratic courses. A hole running at a small angle to a fissure plane tends to run along the fissure. If the hole is at a large angle, the tendency is to penetrate the fissure plane at right angles. The face of the bit having one side in solid formation beyond the fissure, the other in the selvage of the fissure, tends to turn, the rods curving to the extent of their clearance in the hole. Usually rock measures have somewhat regular fissuring or strata and holes take somewhat similar courses, usually at right angles to the fissuring or strata. Excessive pressure or short or worn core barrels increase curvature. Slow speeds, correct feeds, sharp bits, and core barrels the full size of the bit, tend to decrease curvature but do not prevent it Curvature once started increases rapidly. The disappointments from crooked holes have been numerous. Discoveries have later been found to be on the properties of others. One Canadian company had a hole, supposedly discovering ore 300 ft. (91 m.) from old workings, unexpectedly turn in 8 ft. (2.4 m.) of wooden core from a timber in the old workings.

In many cases the information secured by a curved hole is satisfactory, if it can be accurately surveyed. The inclination of the hole at any point

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† Exploration Engineer, Mond Nickel Co.

can be determined by lowering a glass tube containing hydrofluoric acid in a special short length of rod bored out to the diameter of the test tube. By taking etchings at intervals of 100 ft. (30 m ) or less, the curvature of the hole can be plotted. It is necessary to use the same class of tubes in all cases, to have the clinometer and test-tube walls parallel and to prepare a curve of capillarity correction for the class of glass tube used. All holes should be surveyed as the work proceeds. The acid strength can be diluted to allow two or three clinometers to be lowered in a string of rods, shortening the survey time. A special instrument can be purchased for reading the glass tubes but more accurate readings can be made by clamping the tubes to the extended arm of a steel protractor, setting the protractor in a vertical position facing and about 15 ft. away from a transit, using the transit crosshair to determine that the base of the protractor is horizontal and rotating the extended arm with the tube clamped to it, until the etched line coincides with the horizontal crosshair. The angle can then be read off the protractor.

The determination of the course of the hole is not quite so easy and not as satisfactory. We have trammed vertical holes satisfactorily to a depth of about 600 ft. and think deeper tramming is possible. Two sharp nails or steel points are fixed, one at the floor touching the rods, one exactly 2 or 3 rod lengths vertically above. The rods are punch-marked opposite the tram points every two or three lengths and are lowered keeping the punch marks touching tram points and are raised, as an added precaution, in the same way. The direction fixed by the tram point and the center of the rod having been marked on the glass tube in the clinometer lowered, both direction and dip can be calculated from the etching. Deep holes cannot be satisfactorily trammed. A great deal of study has been given to the question of surveying deep holes. A recent Swedish method adopts the principle that a hole is a series of straight lines and angles and places instruments at various points with a fine wire between the instruments. The instruments record the angles turned at the points surveyed. Apparatus has been used that photographs a compass and a small plummet. The simplest method is to divide the etching tube, putting acid in one part, warm gelatine and a small compass in the other. In deep holes it is necessary to have the gelatine in a thermos bottle, as in the Maas instrument, otherwise the gelatine chills too quickly. When a compass is used, a brass clinometer is necessary and two 10-ft. brass rods are placed between the clinometer and the steel rods. Where local attraction is present, the compass is unsatisfactory but tramming and compass will usually give some consistent readings and give a fair idea of the course of the hole. Dividing the gelatine and placing a plummet in one part gives a second way of determining inclination and a check on the etching.

The more accurate survey methods have made it possible to interpret

curved holes, but there are many cases where highly valuable information cannot be obtained if the curvature cannot be controlled. Large orebodies and orebodies with a flat dip are usually defined and sampled by vertical holes at the corners of squares. The computations are very unsatisfactory if the deflection is large. In some cases no information is given by a curved hole. This is the case where the hole runs parallel to or away from the orebody, or again where the deflection is greater than the extent of the orebody. In some measures, holes started vertically deflect so rapidly that they are running horizontally at shallow depths and deep information cannot be obtained.

The question of controlling the direction of drill holes first arose on the Rand and holes, which would never have reached the reef otherwise were turned by wedges and obtained the information sought. Wedges were also used to obtain a second sample. Credit for the greatest enterprise in drilling should be given the Rand, and J. I. Hoffmann's paper[1] and Hugh Mariott's[2] are classics on what Mr. Hoffmann calls the "art of diamond drilling." No doubt Mr. Hoffmann would have considered the carrying of wedging sufficiently far to bore a vertical drill hole a still further improvement of the art.

The Mond Nickel Co. found on two properties that holes, started vertically, were out as much as 400 ft. (121 m.) at depths of 1200 to 1800 ft. (365 to 548 m.). Having several holes to go depths of 2500 to 3000 ft., it was necessary to find a method of controlling curvature. After considerable expensive experimentation, the company has adopted a standard practice of wedging that appears to overcome one

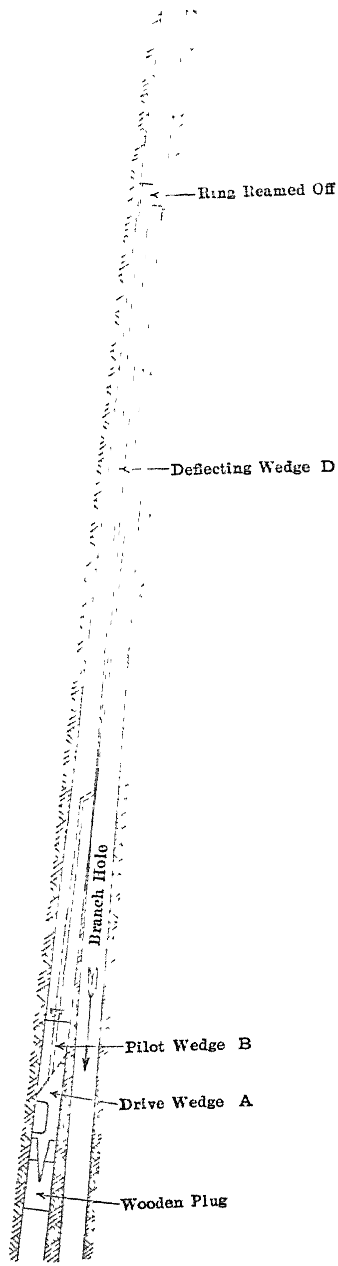


FIG. 1.—DIAGRAM OF DIVERTING WEDGES.

[1] *Trans.* Inst. Min. and Met. (1912) **21,** 481.
[2] *Trans.* Inst. Min. and Met. (1905) **14,** 255.

of the large limitations of the diamond drill. Holes that warrant the expense are wedged back to vertical or back to a straight line as soon as they show deflection of over 3°. A diagram of diverting wedges is shown in Fig. 1.

Each wedging requires the use of a wooden plug, a drive wedge *A*, a pilot wedge *B*, a deflecting wedge *D*, a special clinometer *C*, Fig. 2, and a special reaming bit *E*. Wedging is possible in any hole and no change of size is made, that is an "E" hole remains "E" size, an "A" hole "A" size, and a "B" hole "B" size. The dimensions and description given are for "A" holes.

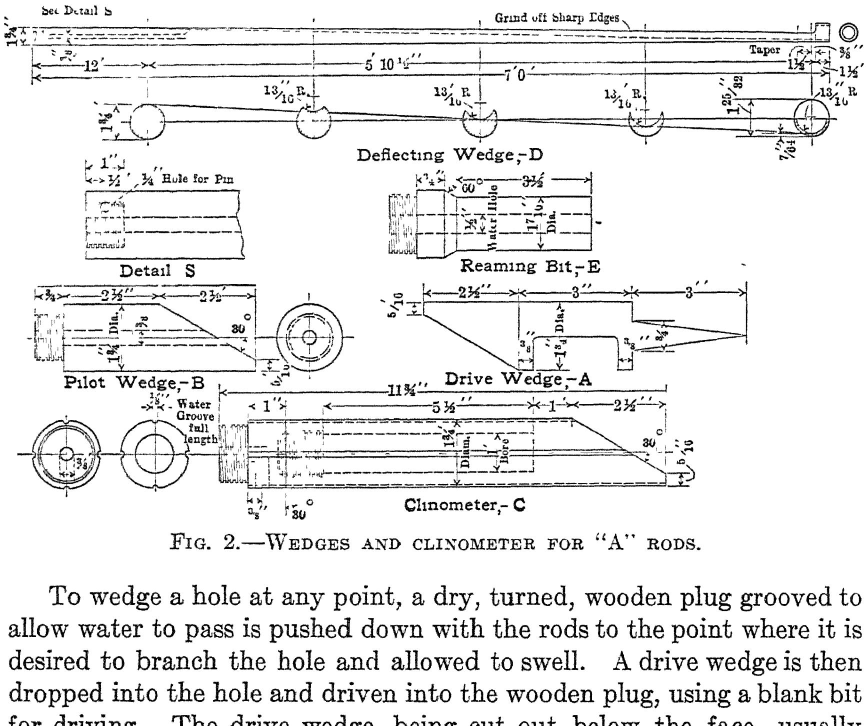


FIG. 2.—WEDGES AND CLINOMETER FOR "A" RODS.

To wedge a hole at any point, a dry, turned, wooden plug grooved to allow water to pass is pushed down with the rods to the point where it is desired to branch the hole and allowed to swell. A drive wedge is then dropped into the hole and driven into the wooden plug, using a blank bit for driving. The drive wedge, being cut out below the face, usually falls with the face of the wedge in the direction of dip but in every case its position is surveyed carefully by using the special clinometer *C*, Fig. 2. Lines are cut on the inside of the clinometer parallel to and in the plane of the long axis of the 30° beveled part of the clinometer. Lines indicating this plane are marked on the sides of a glass test tube with a small carbon, the low point of the bevel face being indicated. The glass tube is filled to a height of about 2 in. with hydrofluoric acid diluted with two parts of water, a cork put in, gummed paper placed over cork and fastened to the sides of the tube, and the plane marked across the paper so that it fits the clinometer snugly and when lowered into the

clinometer the marks coincide. The clinometer is lowered into the hole and allowed to set 30 min. to take the etching; it is then pulled up, the tube cleaned and dried, and the low and high points of the etching marked when the tube is held vertical. The relative direction of the face of the drive wedge and the dip of the hole are indicated. If the two low points coincide, they are the same and it is only necessary to set the face of the pilot wedge and the face of the deflecting wedge in the same direction and lower them. If the two low marks do not coincide, it is necessary to determine the angle between them and to rotate the pilot wedge with regard to the deflecting wedge, to bring the deflecting wedge, when in place, in a direction opposite to the dip of the hole. Strips of paper wound around the test tube and wedges allow marking and measuring the arc determining the angle. If the dip is small, a standard dip protractor or the transit and protractor may be necessary to determine the low point of the etching.

A ring is left at the top of the deflecting wedge to lower it. The pilot wedge and deflecting wedge properly orientated are lowered by riveting a special lowering plug threaded into a blank "A" bit to the ring with a copper rivet. Neither the 30° face of the drive wedge nor the face of the pilot wedge is brought to a thin point but an end ¼ in. thick is left. This provides a surface for driving the drive wedge; also, in one position, the point of the pilot wedge will rest on the point of the drive wedge and on being rotated 180° the pilot wedge will drop 2 in. (5.08 cm.) into its place, indicating when the two are in their proper relation. Shearing of the copper rivet gives a further drop of 3 in. (7.6 cm.). The amount of stretch in the rods must also be taken into account in working at depth; 1500 ft. (457 m.) of "A" rods have about 2.5 in. (6.35 cm.) of slack.

When the deflecting wedge is in place, an "E" bit and core barrel are used and an "E" hole drilled to a point 3 or 4 ft. (0.9 or 1.2 m.) below the wedge. The curved face of the deflecting wedge is "E" size, so the "E" bit follows the wedge without cutting into it. After the "E" drilling, the wedge and deflected hole are reamed out with the special reaming bit "E." The small part of this is "E" size and acts as a pilot. Diamonds are set in the beveled part to ream the wedge and hole out to "A" size. After doing this, the regular "A" rods and "A" bit are used, but reaming with the "A" bit is started at the top of the wedge to make sure there is ample clearance for passing up and down. The "A" hole is continued as an "A" hole below the wedge.

Wedging was first tried in a vertical hole that had been abandoned as useless at 1100 ft. (335 m.). The hole was branched by a wedge at 436 ft. (132 m.) where the deflection was 5° and, by using seventeen wedges, was guided to a depth of over 2400 ft. (731 m.) with a deflection under 24° The correction per wedge was less than expected and the work

indicated that it would have paid to start a new hole and wedge whenever the deflection exceeded 3° The upper part of the hole was a fissured granite or gneiss; the lower, fissured quartzite. The first wedgings were expensive and only partly satisfactory but a skilful setter and crew corrected the difficulties, making accurate wedgings without difficulty after the third wedging. The average correction per wedge was 2°. Two subsequent holes in norite, greenstone, and granite were drilled to depths of over 2500 ft. (762 m.), keeping the deflection within 5° by using three wedges in each. One of these holes had deflected to 5° at a depth of 362 ft. (110 m.). It was wedged at 328, 472, and 580 ft., bringing it back to 1° 10′. The other was out 2° 25′ at a depth of 700 ft. A wedge brought this back to 0° 35′. Though new to the work the setters and crews on the latter holes had no difficulties. Thin core shells being lowered to recover lost cores should be lowered slowly past the top of a wedge. The first deflecting wedges were made without the extra foot of the base and one gave trouble by loosening.

A set of wedges, comprising a drive, a pilot, and a deflecting wedge, cost about $25 in 1918. The average cost of a wedging was about $250, labor $100, fuel $50, carbon $50, wedges $25, and miscellaneous $25. Experienced crews took five shifts to complete a wedging, a shift placing a wedge, one shift drilling with "E" bit, two shifts reaming "E" hole to "A" size, one shift reaming through the wedge with the "A" bit.

Credit for the many suggestions is due the members of the firm of Smith & Travers, contractors. A successful wedging method, in addition to overcoming curvature, can be used to branch a hole for any purpose, for securing additional records of strata or additional samples of the vein or deposit. The application of wedging to the correction of curvature appears to overcome one of the large limitations of the diamond drill.

## DISCUSSION

Hugh M. Roberts,* Minneapolis, Minn. (written discussion†).—The paper by Messrs. Hall and Row marks a distinct advance in the art of diamond drilling, because it records a systematic application of a method for directing the path of the diamond bit. Their work is an admirable example of engineering development. They had a definite object to attain; *i.e.*, the drilling of many vertical diamond-drill holes to depths of 2500 or 3000 ft. (762 or 914 m.). They availed themselves of a method of wedging seldom used, improved the manner of making the wedges, developed a technique of survey, and put the process into practical, continuous operation. So effectively has this been done that they determine the average cost of the work. The publication of

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* E. J. Longyear Co.

† Received Sept. 22, 1919.

the results, in the form of detailed diagrams, makes the application general to whoever has a similar problem.

The wedging of diamond-drill holes has been practised in the Lake Superior region for the purpose of making branch holes, particularly where the original hole has penetrated a great thickness of glacial drift. The uneven nature of the iron formations has caused the results to be somewhat uncertain. One operation, in 1907, near the American mine on the Marquette Range, by the firm of Longyear & Hodge, predecessors of the E. J. Longyear Co., resulted in the drilling of a hole 1828 ft. (557 m.) deep, with two branches at depths 100 ft. (30 m.) and 770 ft. (234 m.), respectively. A second hole from the same set-up went to a depth of 2497 ft. (761 m.) with a branch at 1850 ft. (563 m.). In this instance, the wedges were forged to the desired shapes and angles with respect to each other. A drill hole, sunk in 1915, in the township of Bates in the Iron River district of Northern Michigan, which penetrated 385 ft. (117 m.) of boulders and gravel, was also similarly deflected for the purpose of taking a second sample across the iron formation. The wedging of diamond-drill holes has been accomplished in the copper country, also for the making of branch holes. However, it has remained for Messrs. Hall and Row to develop a precise method that can be generally applied.

In their paper, the authors touch upon methods of surveying diamond-drill holes. These are generally unsatisfactory because of the limited size of the hole, expensive by reason of the time consumed, and the results of the surveys usually leave a feeling of uncertainty in their wake. It has occurred to me that if a method of survey could be devised that would be entirely independent of the drill hole itself, it would be desirable.

The grinding of a diamond bit may be heard distinctly for long distances through solid rock, even through a hundred feet of glacial drift. Listening instruments of great accuracy have been developed in connection with underground warfare in Europe. One instrument known as the American geophone, a simplified seismograph, has been developed by the United States Army Engineers, and has lately been applied by the U. S. Bureau of Mines to purposes of mine rescue. Officers who have served in Canadian tunneling regiments state that the position of German headings could be determined by means of listening instruments, from the sounds of tools at work, with accuracy through chalk at distances of 1000 ft. (304 m.) or more, not only as to position in azimuth but as to elevation. In a solid rock, like the norite of the Sudbury district, sound waves are transmitted distinctly. By applying the geophone to the survey of diamond-drill holes, it may be possible to fix the position of the bit while at work at various points in the hole, say every 100 ft. (30 m.) in depth, and thus determine quickly and with a fair degree of precision

the path of the drill hole in three dimensions. The method would be particularly applicable to the survey of angle holes. These instruments are as yet in the hands of the Corps of Engineers and the Bureau of Mines The Bureau of Mines has indicated a willingness to make tests on the application of the geophone to the survey of diamond-drill holes. The results will be watched with great interest.

The paper by Messrs Hall and Row brings forcibly to mind the growing effectiveness of the diamond drill as an exploring agent. The prime necessity in any mining enterprise is an orebody to work. Improvements in the methods used for discovering orebodies and determining their nature are of great importance to the whole mining industry.

# Unwatering the Tiro General Mine by Air-lift

BY S F SHAW, E M,* CHARCAS S L P, MEXICO

(New York Meeting, February, 1920)

In 1913, the Tiro General mine, at Charcas, S.L.P., Mexico, which had been making from 125 to 150 gal. of water per min., was allowed to become flooded, after all the pumps had been removed, and in 1918 the problem of unwatering it came up for solution. There are two vertical shafts, the Tiro General, 1390 ft., and the San Fernando, 1225 ft. in depth, with five connecting levels between them. The San Fernando shaft had been partly retimbered, while the Tiro General had been newly timbered from the collar to 1090 ft. in depth. It had been possible to place guides in only one compartment of the Tiro General, from the collar to the seventh or 850-ft. level, to which it was necessary to lower the water before station pumps could be installed. Soundings indicated that compartments in the Tiro General were open to 1082 and 428 ft., and in the San Fernando shaft to 471, 488, and 586 ft. The water had risen to a point 140 ft. below the collar of the Tiro General shaft.

## Pumping Equipment

The steam plant at the Tiro General shaft includes one 300-hp., one 150-hp. and one 100-hp. Erie City water-tube boiler, and two 160-hp. Heine water-tube boilers. All but one of these boilers were supposed to be in good shape, but it was soon discovered that six years of idleness in a climate that is foggy for several months of the year had left them in poor condition. The compressed-air plant comprises one Nordberg cross-compound compressor with piston displacement of approximately 2500 cu. ft. of free air compressed to 96 lb. per sq. in.; and another Nordberg cross-compound with piston displacement of 1235 cu. ft. of free air, to the same pressure.

The main pump station is on the seventh level where had been installed a Prescott pot-valve plunger pump with capacity of 200 gal. per min.; also a Gould triplex plunger pump with capacity of 135 gal. per min., driven by a 6½ by 10 by 12 in. steam engine. From the lower levels, the water had been raised to the seventh level by small station pumps and sinking pumps.

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* Superintendent, American Smelting & Refining Co.

Although the submergence of an air-lift pump at the final stage of unwatering to the seventh level would be limited to about 225 ft., we were assured by pump manufacturers that this point could be reached by a single-stage air-lift, and we therefore decided to adopt this method. The equipment to be purchased consisted only of piping, footpieces, etc., since the compressor plant was considered adequate for the purpose. Not only did we save the first cost of such equipment as sinkers, or centrifugal pumps and motors, but we also found that delivery of pipe could be effected in much less time than pumps and motors.

At the beginning of operations, two air-lifts were installed, one in the Tiro General and one in the San Fernando shaft. These were operated until it became evident that they were not suitable for the work to be required of them at greater depth; they were then removed and replaced by air-lifts of better design, designated Nos. 3 and 4. Since the lift involved in lowering the water to the seventh level, where steam pumps could be installed, was much greater than had ever been attempted before where a large quantity of water was to be handled, it was necessary to make many careful experiments as the work progressed, in order to anticipate the probable performance of the lifts that were installed. In order to show the line of development, the successive installations will be described.

## Air-lift No 1

*Delivery Pipe.*—The first installation consisted of 300 ft. of 6-in. spiral-riveted pipe connected to the footpiece, then a 6 to 5-in. reducer, followed by 5-in. spiral-riveted pipe to the surface. It was soon discovered that the reduction in diameter was a mistake, and therefore the 5-in. pipe was replaced by 6-in. pipe to a height of 600 ft., at which point the 5-in. pipe had to be retained, as all the available 6-in. pipe had been used. The completed line, therefore, consisted of 600 ft. of 6-in. spiral-riveted pipe, from the footpiece, a 6 to 5-in. reducer, 23 in. long, 100 ft. of 5-in. spiral-riveted pipe, a 6-in. nipple 7 ft. 9 in. long attached to the headpiece, to permit rapid connection of the spiral pipe, a 6-in. nipple 26 in. long within the headpiece, and at the footpiece a 6-in. nipple 4 ft. 1 in. long which allowed the riveted pipe to be conveniently connected with the footpiece. The total length of the delivery line was therefore 716 feet.

*Air Line.*—The air line from the footpiece to the shaft collar consisted of 4¼-in. casing; this was connected flexibly to the 6-in. air main from the compressor plant.

*Footpiece.*—A Talbot 6-in. footpiece was used. The body, being of wood, was wrapped in sheet iron and wire to protect it from falling rocks.

*Headpiece.*—This was a sheet-iron tank, 36-in. in diameter, and 69 in. high inside measurements. In the center of the bottom was a 6-in. hole, to which, both above and below, flanges were bolted into which

6-in. nipples were screwed. The inside nipple was 26 in. long while the lower one was 7 ft. 9 in. long, and had a flange screwed on its lower end, by which connection could be made easily and quickly with the 6-in. spiral-riveted pipe. A sheet-iron conical deflector was hung from the top

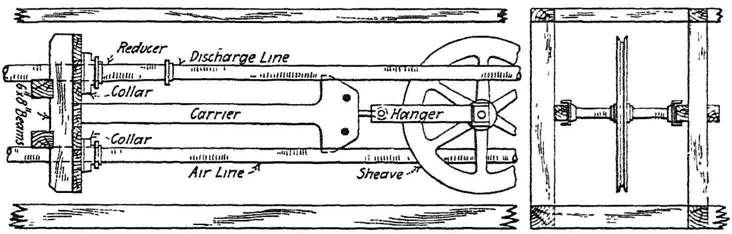


FIG. 1—SKETCH OF AIR-LIFT CARRIER.

of the headpiece, to which it was rigidly attached at a distance of about 50 in. above the bottom of the tank. The water was thereby diverted to the sides of the tank, while the air escaped through an opening in the top of the headpiece. A 12-in. opening was cut in the side of the headpiece, close to the bottom, from which the water was conducted through a 12-in. pipe to the measuring box.

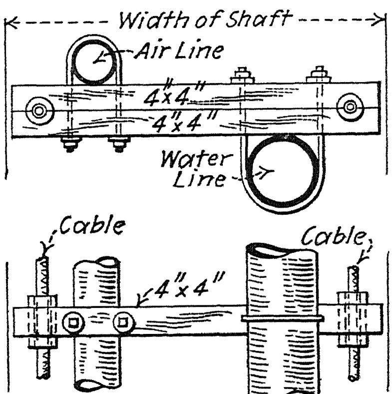


FIG. 2.—WOOD CLAMPS SUPPORTING PIPE COLUMNS ABOVE CARRIER.

*Carrier.*—The lower 300 ft. of the water column and air pipe were suspended below a carrier, the frame of a 4-ton skip from which the box had been removed. The pipes were supported on the floor of the carrier by heavy iron-clamps which also carried the weight of the columns above the carrier. The carrier was hung from a sheave wheel to the axle of which were attached shoes which fitted the shaft guides. A 1-in. steel-wire cable from the hoisting engine and headframe sheave passed under the

carrier sheave, and then up to the headframe, where its end was securely attached. This arrangement diminished the danger of breaking the cable and permitted the entire weight to be raised by the hoisting engine. It also made easier the adding of lengths to the water and air lines, Fig. 1. Above the carrier, the air and water pipes were supported by wood clamps, through the ends of which the two segments of hoisting cable were free to pass, as shown in Fig. 2.

*Measuring Box.*—To obtain accurate measurements of water, a box was built containing a V-notch weir. This box was 36 in. wide, 37 in. deep, and 10 ft. 4 in. long, and was provided with a Sanborn automatic water-recording instrument, which permitted approximate water measurements to be taken from the chart at any time, and also enabled us to estimate the number of hours during the day that the air-lift was in operation.

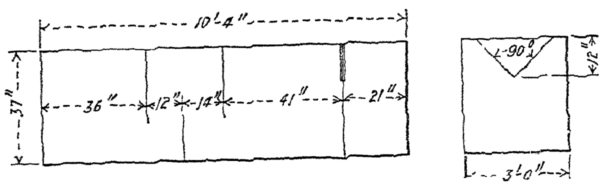


FIG 3.—MEASURING BOX USED AT TIRO GENERAL SHAFT

*Operation.*—At first, when the lift consisted of 300 ft. of 6-in. pipe followed by 60 ft. of 5-in. pipe reaching to the shaft collar, a 5-in. pipe 350 ft. long and containing four 90° elbows was connected directly to the top of the water column with the intention of conducting the water to a reservoir from which it could be measured and drawn off as desired. With this arrangement, the discharge was intermittent and the mixed water and air issued in powerful blasts. The maximum capacity, when lifting the water only 150 ft., was 275 gal. per min. The headpiece described above was then attached to the water column, allowing the water and air to issue freely, and immediately increasing the capacity to 400 gal. per min. On substituting 6-in. pipe for the 60 ft. of 5-in. pipe, a further increase of 125 gal. resulted, making the total discharge 525 gal. per min. This equipment was in operation from Feb. 10 to May 19, during which time the lift ranged from 155 to 485 ft. and the capacity decreased from 525 to 270 gal. per min. Table 1 gives various data relating to this period. As the lift increased, the capacity of the 6-in. water line constantly diminished. A series of tests was made to find the relation between lift, amount of water raised, and cubic feet of free air consumed, the results being shown in Tables 2 and 3 and Fig. 4. A fairly constant decrease in pumping capacity, for a given diameter of water column, as the lift increases will be noticed, provided

TABLE 1.—*Air-lift No.* 1

| Lift, Feet | Submergence, Feet | Total Length of Pipe, Feet | Per Cent. Submergence | Operating Capacity, Gallons | Cubic Feet Air per Gallon |
|---|---|---|---|---|---|
| 160 | 187 | 347 | 53.8 | 275 | 1.42 |
| 171 | 197 | 368 | 53.6 | 275 | 1.51 |
| 182 | 190 | 372 | 51.2 | 435 | 0.98 |
| 192 | 179 | 371 | 48.3 | 525 | 0.91 |
| 201 | 189 | 390 | 48.5 | 525 | 0.98 |
| 212 | 178 | 390 | 45.7 | 490 | 1.05 |
| 220 | 212 | 430 | 48.9 | 520 | 1.01 |
| 229 | 201 | 430 | 46.7 | 540 | [a] |
| 241 | 198 | 439 | 45.3 | 530 | |
| 252 | 198 | 450 | 44.0 | 485 | |
| 259 | 211 | 470 | 44.7 | 500 | |
| 271 | 200 | 471 | 42.4 | 420 | |
| 282 | 208 | 490 | 42.5 | 460 | |
| 295 | 197 | 492 | 40.0 | 430 | |
| 303 | 202 | 505 | 40.0 | 430 | |
| 310 | 204 | 514 | 39.8 | 412 | |
| 325 | 200 | 525 | 38.3 | 396 | |
| 330 | 205 | 535 | 38.3 | 390 | |
| 343 | 232 | 575 | 40.4 | 415 | |
| 350 | 225 | 575 | 39.2 | 412 | |
| 373 | 232 | 605 | 38.3 | 402 | |
| 382 | 233 | 615 | 37.9 | 390 | |
| 398 | 227 | 625 | 36.4 | 350 | |
| 410 | 234 | 644 | 36.4 | 355 | |
| 423 | 232 | 655 | 35.6 | 350 | |
| 433 | 242 | 675 | 35.8 | 315 | |
| 440 | 249 | 689 | 35.8 | 320 | |
| 454 | 241 | 695 | 34.8 | 275 | |
| 461 | 234 | 695 | 33.6 | 280 | 2.92 |
| 471 | 244 | 715 | 34.2 | 285 | 3.04 |
| 481 | 234 | 715 | 32.7 | 275 | |

[a] Tiro General No. 1 air-lift and San Fernando air-lift were supplied with air from the same air-main simultaneously, therefore individual measurements of air consumed could not be made under regular operating conditions.

the depth of submergence remains constant, and in the case of the 6-in. installation the inference can be drawn that at about 900 ft. lift there would be little or no water issuing from the discharge pipe.

It was observed that beyond a certain amount of air, further additions did not increase the quantity of water raised; this point is designated as the "maximum capacity." Another point was observed on all curves at which the pumping efficiency was greater than for either a smaller or larger amount of water; this is designated as the point of "maximum efficiency." The capacity at which the air-lift was operated during

TABLE 2.—*Tests on Air-lift No. 1*

| No. | Lift, Feet | Sub-mergence, Feet | Total Length Pipe, Feet | Per Cent Sub-mergence | Capacity at Maxi-mum Effi-ciency | Cubic Feet Air at Maxi-mum Effi-ciency | Maxi-mum Capacity | Cubic Feet Air at Maxi-mum Capacity | Pump-ing Effi-ciency at Maxi-mum Effi-ciency | Pump-ing Effi-ciency at Maxi-mum Capacity |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 181 | 190 | 370 | 51.0 | 340 | 0.66 | 510 | 1.60 | 57.6 | 23.1 |
| 2 | 192 | 178 | 370 | 52.0 | 392 | 0.74 | 580 | 1.30 | 54.0 | 32.3 |
| 3 | 214 | 216 | 430 | 50.0 | 450 | 0.67 | 588+ | 1.26 | 62.7 | 33.3 |
| 4 | 300 | 195 | 495 | 39.5 | 307 | 1.52 | 464 | 2.36 | 41.3 | 26.3 |
| 5 | 354 | 221 | 575 | 38.5 | 295 | 1.67 | 440 | 2.70 | 41.3 | 25.6 |
| 6 | 396 | 219 | 615 | 35.7 | 270 | 1.84 | 406 | 3.17 | 42.9 | 25.0 |
| 7 | 428 | 247 | 675 | 36.6 | 255 | 2.00 | 370 | 3.18 | 41.0 | 21.0 |
| 8 | 466 | 249 | 715 | 34.8 | 242 | 2.47 | 307 | 3.57 | 35.2 | 24.4 |

TABLE 3.—*Tests on Air-lift No. 1*

| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 |
|---|---|---|---|---|---|---|---|---|
| Submergence (ft.) | 190 | 178 | 216 | 195 | 221 | 219 | 247 | 249 |
| Lift (ft.) | 181 | 192 | 214 | 300 | 354 | 396 | 428 | 466 |
| Ratio $\frac{s}{l}$ | 105.0 | 93.0 | 101.0 | 65.0 | 62.0 | 56.0 | 58.0 | 53.4 |
| Gal. water per min. | 340 | 392 | 450 | 307 | 295 | 270 | 255 | 242 |
| Total cu. ft. free air (7250 ft.) | 283 | 366 | 384 | 594 | 627 | 633 | 648 | 760 |
| Total cu. ft. free air (sea level) | 223 | 289 | 302 | 468 | 492 | 498 | 510 | 598 |
| Cu. ft. per gal. (sea level) | 0.66 | 0.74 | 0.67 | 1.52 | 1.67 | 1.84 | 2.00 | 2.47 |
| Per cent. pumping efficiency. | 57.6 | 54.0 | 62.7 | 41.3 | 41.3 | 42.9 | 41.0 | 35.2 |
| Velocity per second, water footpiece | 4.0 | 4.6 | 5.3 | 3.6 | 3.5 | 3.2 | 3.0 | 2.8 |
| Velocity per second, water headpiece | 4.0 | 4.6 | 5.3 | 3.6 | 3.5 | 4.4 | 4.1 | 4.0 |
| Velocity per second, air footpiece | 2.9 | 3.9 | 2.4 | 5.9 | 5.6 | 5.6 | 5.2 | 6.1 |
| Velocity per second, air headpiece | 25 | 32 | 34 | 52 | 55 | 78 | 79 | 93 |
| Velocity per second, mixture footpiece | 6.9 | 8.5 | 7.7 | 9.5 | 9.1 | 8.8 | 8.2 | 8.9 |
| Velocity per second, mixture headpiece | 29 | 37 | 39 | 56 | 59 | 82 | 83 | 97 |
| Air pressure (submergence) | 82 | 77 | 94 | 85 | 96 | 97 | 107 | 108 |
| Per cent submergence | 51.0 | 52.0 | 50.0 | 39.5 | 38.5 | 35.7 | 36.6 | 34.8 |
| Maximum gal. per min. | 510 | 580 | 588 | 464 | 440 | 406 | 370 | 307 |
| Cu. ft. per gal., max. capacity | 1.60 | 1.30 | 1.26 | 2.36 | 2.70 | 3.17 | 3.18 | 3.57 |
| Sq. ft. inside pipe surface. | 590 | 590 | 682 | 786 | 913 | 976 | 1056 | 1109 |
| Pumping effic. at max. cap. | 23.1 | 32.3 | 33.3 | 26.3 | 25.6 | 25.0 | 21.0 | 24.4 |

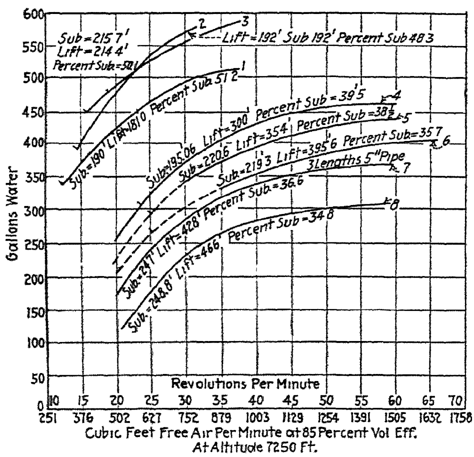


FIG. 4.—CURVES BASED ON DATA OF TABLE 3.

any day is designated the "operating capacity." The capacity at maximum efficiency is indicated in Fig. 4, as well as the maximum capacity of the pipe. In Fig. 5 it will be seen that curves showing the most efficient capacity, the operating capacity and the maximum capacity are

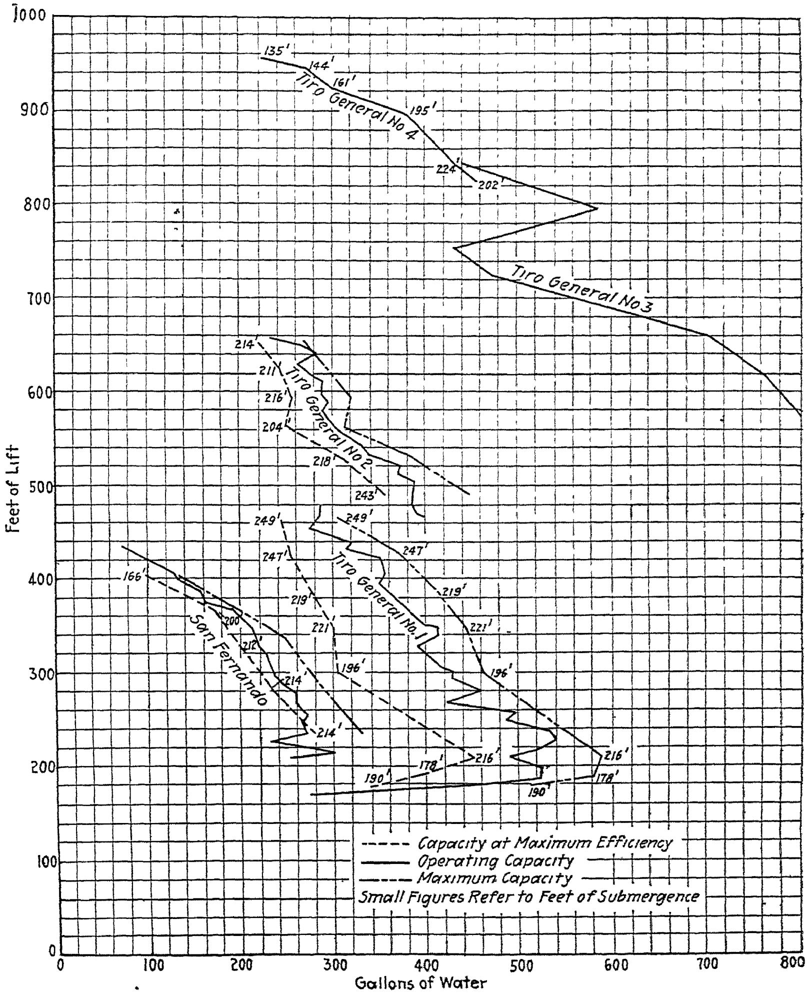


Fig. 5.—Influence of lift on capacity.

roughly parallel, indicating that all diminish as the lift is increased provided the depth of submergence remains approximately constant.

## Air-lift No. 2

*Delivery Pipe.*—The water column ultimately consisted of 865 ft. of wood pipe, of 6-in. diameter at the footpiece and 6⅜ in. at the top. This was tightly bound by wire to the air pipe from the footpiece to the

carrier, a distance of 300 ft., in such manner that its entire weight was carried by the air pipe. The joints were wrapped with light canvas, painted with heavy fuel oil and wrapped with No. 16 annealed iron wire, in spite of which there were many leaks in the line The staves of the pipe were ½ in. thick.

*Air Pipe.*—This was 3-in. casing, from the footpiece to the shaft collar, where it was attached through flexible connections to the 6-in. air main.

TABLE 4.—*Air-lift No. 2*

| Lift, Feet | Submergence, Feet | Total Length of Pipe, Feet | Per Cent Submergence | Operating Capacity, Gallons | Cubic Feet Air per Gallon |
|---|---|---|---|---|---|
| 465 | 239 | 705 | 34.0 | 400 | |
| 471 | 234 | 705 | 33.2 | 390 | |
| 482 | 237 | 719 | 33.0 | 385 | |
| 496 | 239 | 735 | 32.5 | 386 | 2.60 |
| 503 | 232 | 735 | 31.7 | 388 | 2.60 |
| 514 | 229 | 743 | 30.8 | 370 | 2.72 |
| 521 | 229 | 750 | 30.5 | 375 | 2.64 |
| 532 | 218 | 750 | 29.0 | 340 | 2.90 |
| 541 | 209 | 750 | 27.9 | 332 | 2.88 |
| 550 | 216 | 766 | 28.2 | 322 | 3.36 |
| 561 | 205 | 766 | 26.8 | 304 | 3.53 |
| 570 | 226 | 796 | 28.4 | 295 | 3.11 |
| 580 | 216 | 796 | 27.2 | 290 | 3.31 |
| 590 | 222 | 812 | 27.3 | 295 | 3.40 |
| 599 | 227 | 826 | 27.5 | 290 | 3.35 |
| 611 | 223 | 834 | 26.7 | 290 | 3.46 |
| 622 | 220 | 842 | 26.1 | 270 | 3.47 |
| 630 | 212 | 842 | 25.2 | 260 | 3.72 |
| 639 | 231 | 870 | 26.5 | 285 | 3.65 |
| 649 | 221 | 870 | 25.4 | 265 | 3.90 |
| 657 | 213 | 870 | 24.5 | 233 | 4.07 |

TABLE 5.—*Tests on Air-lift No. 2*

| No. | Lift, Feet | Submergence, Feet | Total Length Pipe, Feet | Per Cent. Submergence | Capacity at Maximum Efficiency | Cubic Feet Air at Maximum Efficiency | Maximum Capacity | Cubic Feet Air at Maximum Capacity | Pumping Efficiency at Maximum Efficiency | Pumping Efficiency at Maximum Capacity |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 492 | 243 | 735 | 33.0 | 358 | 2.40 | 447+ | 2.80 | 38.7 | 33.2 |
| 2 | 532 | 218 | 750 | 29.0 | 308 | 2.73 | 384 | 3.60 | 38.1 | 29.0 |
| 3 | 562 | 204 | 766 | 26.6 | 250 | 3.54 | 314+ | 4.10 | 31.9 | 27.8 |
| 4 | 595 | 216 | 811 | 26.8 | 255 | 3.04 | 320 | 4.0 | 38.8 | 29.2 |
| 5 | 630 | 211 | 841 | 25.1 | 238 | 3.31 | 290 | 4.43 | 38.1 | 28.5 |
| 6 | 657 | 214 | 871 | 24.6 | 219 | 3.90 | 268 | 5.00 | 33.1 | 26.2 |

TABLE 6—*Tests on Air-lift No. 2*

| | 1 | 2 | 3 | 4 | 5 | 6 |
|---|---|---|---|---|---|---|
| Submergence (ft ) | 243 | 218 | 204 | 216 | 211 | 214 |
| Lift (ft.) . . . . | 492 | 532 | 562 | 595 | 630 | 654 |
| Ratio $\frac{s}{l}$.. .. . | 50 0 | 41 0 | 36 0 | 36 3 | 33 5 | 32.5 |
| Gal water per min | 358 | 308 | 250 | 255 | 238 | 219 |
| Total cu. ft free air (7250 ft.) | 1093 | 1070 | 1123 | 987 | 1003 | 1086 |
| Total cu ft free air (sea level) | 860 | 842 | 884 | 770 | 790 | 855 |
| Cu. ft. per gal. (sea level).. | 2 40 | 2 73 | 3 54 | 3 04 | 3 31 | 3 90 |
| Per cent pumping efficiency | 38 7 | 38 1 | 31 9 | 38 8 | 38 1 | 33 1 |
| Velocity per second, water foot-piece . . . . . . . . . | 4 2 | 3 6 | 2.9 | 3 0 | 2 8 | 2 6 |
| Velocity per second, water head-piece . . . . . . . . | 3 6 | 3 1 | 2 5 | 2 6 | 2 4 | 2 2 |
| Velocity per second, air footpiece | 9 0 | 9 5 | 10 8 | 9 1 | 9 3 | 10 1 |
| Velocity per second, air headpiece | 82 | 80 | 84 | 74 | 78 | 82 |
| Velocity per second, mixture foot-piece.... ........ .. . | 13 2 | 13 1 | 13.7 | 12 1 | 12 1 | 12 7 |
| Velocity per second, mixture head-piece........... ........ | 86 | 83 | 87 | 77 | 80 | 84 |
| Air pressure (submergence) . | 105 | 97 | 88 5 | 94 | 92 | 93 |
| Per cent submergence . . | 33 0 | 29 0 | 26 6 | 26 8 | 25 1 | 24.6 |
| Maximum gal per min . | 447 | 384 | 314 | 320 | 290 | 268 |
| Cu. ft. per gal , max. cap . . | 2 88 | 3 60 | 4 10 | 4 0 | 4 43 | 5 0 |
| Pumping effic., max. cap. . | 33 2 | 29 0 | 27 8 | 29 2 | 28 5 | 26 2 |
| Sq. ft. inside pipe surface . | 1170 | 1178 | 1203 | 1274 | 1321 | 1368 |

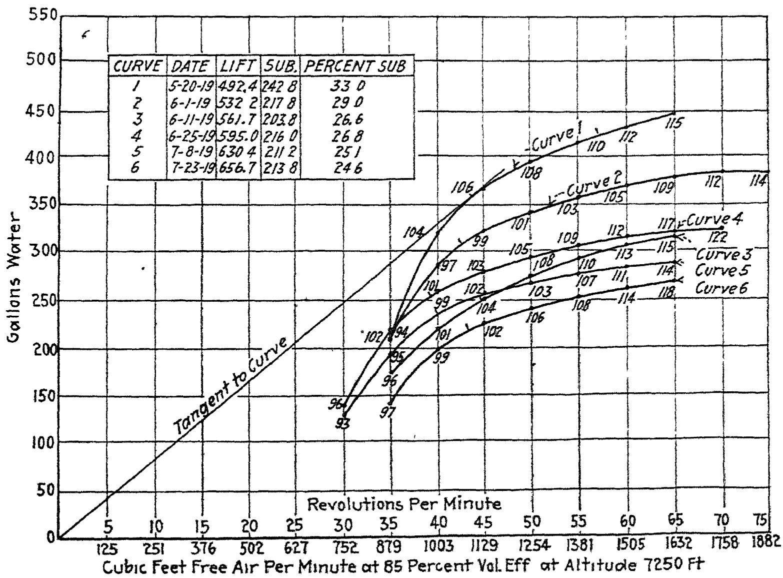


FIG. 6.—CURVE BASED ON DATA OF TABLE 6.

*Footpiece.*—This was a Talbot 6-in. design, like the one used on No. 1 lift.

*Headpiece.*—This was a wooden tank of 42 in. inside diameter and 36 in. depth. A 9-in. flanged nipple 22 in. long was bolted to the bottom on the inside, through which the wood pipe passed. A deflector was held by long bolts at a distance of 32 in. above the bottom. The water escaped through an 8-in. pipe to the measuring box.

*Operation.*—This installation was in operation between May 17 and July 23, the lift ranging from 465 to 657 ft. The points of maximum capacity, operating capacity and efficient capacity are noted on Fig. 5. Table 4 gives the operating capacity at lifts increasing by approximately 10-ft. intervals. Tables 5 and 6 record the results of special tests, which are also plotted in Fig. 6.

The slightly larger diameter, together with the reduced friction on the sides of the wood pipe, increases the capacity to some extent over that of the 6-in. spiral-riveted pipe. The consumption of air per gallon of water was also diminished as compared with that which would have obtained with the 6-in. spiral-riveted pipe. There were several leaks in the pipe which reduced the capacity and efficiency by probably not less than 5 per cent.

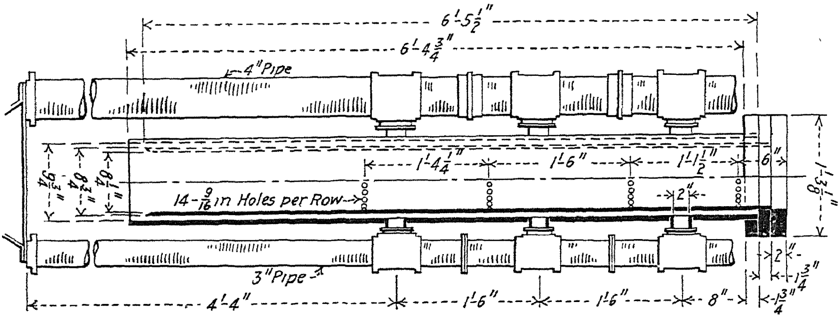


FIG. 7.—CHARCAS FOOTPIECE FOR AIR-LIFT NO. 3.

## AIR-LIFT No. 3

*Delivery Pipe.*—The delivery line ultimately consisted of 1050 ft. of wood pipe of 8¼-in. diameter at the footpiece, increasing gradually to the discharge, where it was 12½-in. diameter. The staves were 1 in. thick and the joints 2½ in. long. The outside of each joint was first coated with tar, then wrapped with about 10 ft. of canvas, 5 in. wide, and then close wrapped with No. 16 annealed iron wire, covering the entire canvas. At the beginning, the length of pipe was 765 ft. and required 5½ days for installation.

*Air Pipe.*—This consisted of two parallel lines, 400 ft. long, of 3-in. and 4¼-in. casing, extending from the footpiece to the carrier; above this point these pipes were connected to a 6-in. spiral-riveted pipe, 520 ft.

long, and then reduced to 5-in. spiral-riveted pipe which reached to the collar of the shaft, a further distance of 100 feet.

*Footpiece.*—The footpiece was made at Charcas, and consisted of an inner casing pipe, 8-in. diameter, in which were bored four rows of 14 holes each, the diameter of the holes being 9/16 in. Air was introduced into the footpiece by suitable connections through a 10-in. extra-heavy pipe fitted outside the 8-in. pipe, as shown in Fig. 7. A Talbot 8-in. footpiece collapsed at the beginning of operations.

*Headpiece.*—This was practically identical with the one used for No. 2 lift.

TABLE 7.—*Air-lift No.* 3

| Lift, Feet | Submergence, Feet | Total Length of Pipe, Feet | Per Cent Submergence | Operating Capacity, Gallons | Cubic Feet Air per Gallon |
|---|---|---|---|---|---|
| 548 | 230 | 778 | 29 5 | 900 | |
| 556 | 223 | 779 | 28 6 | 800 | 2.32 |
| 568 | 225 | 793 | 28 4 | 750 | 1 90 |
| 579 | 244 | 823 | 29 7 | 800 | 1.83 |
| 590 | 255 | 845 | 30 2 | 750 | 1 97 |
| 602 | 250 | 852 | 29 3 | 800 | 1 83 |
| 613 | 247 | 860 | 28 5 | 800 | 1.80 |
| 623 | 248 | 871 | 28 2 | 750 | 1 91 |
| 632 | 250 | 882 | 28 4 | 750 | 1 98 |
| 646 | 251 | 897 | 27.9 | 700 | 2 07 |
| 657 | 241 | 898 | 26 9 | 700 | 2 04 |
| 669 | 258 | 927 | 27 9 | 700 | 2.05 |
| 679 | 248 | 927 | 26 7 | 700 | 2 05 |
| 690 | 237 | 927 | 25.6 | 700 | 2.03 |
| 700 | 257 | 957 | 26 9 | 480 | 2.92 |
| 710 | 247 | 957 | 25 8 | 470 | 2.90 |
| 718 | 253 | 971 | 26.1 | 475 | 2.90 |
| 729 | 258 | 987 | 26.1 | 475 | 2.96 |
| 740 | 247 | 987 | 25.1 | 475 | 2.99 |
| 751 | 266 | 1017 | 26.1 | 375 | 3.09 |
| 761 | 256 | 1017 | 25.2 | 455 | 3.08 |
| 764 | 253 | 1017 | 24.9 | 500 | 3.09 |
| 774 | 242 | 1016 | 23.9 | 630 | 3.42 |
| 784 | 233 | 1017 | 22.9 | 640 | 3.40 |
| 795 | 222 | 1017 | 21.9 | 600 | 3.40 |
| 804 | 220 | 1024 | 21.5 | 590 | 3 41 |
| 813 | 218 | 1031 | 21.1 | 630 | 3.42 |
| 824 | 207 | 1031 | 20 1 | 445 | 4.86 |
| 832 | 199 | 1031 | 19.2 | 447 | 4.87 |
| 843 | 188 | 1031 | 18 3 | 443 | 4.87 |

*Operation.*—This installation operated from Aug. 19 until Oct. 6, with lifts ranging from 548 to 843 ft., and capacities of from 900 gal. at 548 ft. to 443 gal. at 843 ft. A test was made at the beginning, before

one of the compressors broke down, allowing sufficient air to enter the footpiece to produce a continuous flow. During this test the lift was 548 ft., submergence 230 ft., air 3325 cu. ft. per min. (altitude of 7250 ft.), and flow of water 1100 gal. per minute. After one of the compressors broke down, there was insufficient air to produce a continuous discharge, and the lift operated in pulsations of 50 to 60 sec.

Table 7 gives the operating capacity of No. 3 air-lift, at approximately 10-ft. intervals. Owing to the irregular operation, it was impossible to obtain close measurements of the flow of water, and therefore tests were not made as for the previous installation. Moreover, there was insufficient air to carry the tests to a proper conclusion. The second compressor was repaired and placed in operation when the lift reached 774 ft., thus accounting for the increased discharge at that point. A test, using both compressors, is plotted in Fig. 8.

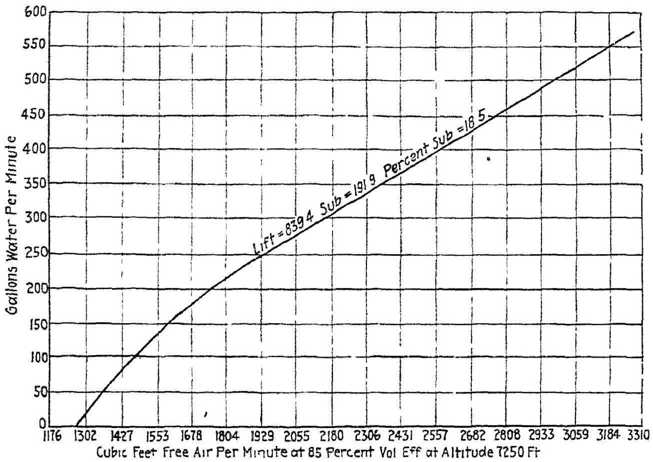


FIG. 8.—RECORD OF A TEST OF AIR-LIFT NO. 3.

## AIR-LIFT NO. 4

*Delivery Pipe.*—The completed line consisted of 270 ft. each of 7-in., 8-in., 9-in. and 10-in. spiral-riveted pipe, with appropriate reducers, each 23½ in. long. Inside the headpiece was a 10-in. nipple, 26 in. long, making the total length of the delivery line 1090 ft. Installing the pipeline, with an original length of 1030 ft., required 2 days.

*Air Line.*—This consisted of parallel 3-in. and 4¼-in. casing from the footpiece to the carrier, 365 ft., and 500 ft. of 6-in. spiral-riveted pipe above the carrier, followed by 200 ft. of 5-in. spiral-riveted pipe to the shaft collar.

*Footpiece.*—The footpiece was of the same design as used with the No. 3 installation. The headpiece, also, was practically identical with that used for No. 2 installation.

*Operation.*—This air-lift was in operation from Oct. 10 to Jan. 6, the lift ranging from 812 to 1072 ft., and the operating capacity from 465 gal. at the former to 65 gal. per min. at the latter distance. The latter submergence was only 18 ft. A steady flow was obtained from the first and continued throughout.

Table 8 shows the operating capacity of this air-lift at intervals of approximately 10 ft. Since it was operated close to the point where pulsations would begin, it is probable that the operating capacity was close to the efficient capacity.

TABLE 8.—*Air-lift No.* 4

| Lift, Feet | Submergence, Feet | Total Length Pipe, Feet | Per Cent Submergence | Operating Cap., Gal | Cu. Ft. Air Per Gal. | Refining Efficiency, Per Cent |
|---|---|---|---|---|---|---|
| 812 | 218 | 1030 | 21.2 | 465 | 4.38 | 36.3 |
| 820 | 210 | 1030 | 20.4 | 475 | 4.67 | 35.4 |
| 830 | 215 | 1045 | 20.6 | 462 | 4.43 | 36.8 |
| 839 | 221 | 1060 | 20.8 | 445 | 4.56 | 38.4 |
| 849 | 226 | 1075 | 21.0 | 335 | 4.25 | 38.8 |
| 859 | 231 | 1090 | 21.2 | 455 | 4.59 | 36.0 |
| 872 | 218 | 1090 | 19.9 | 465 | 4.70 | 36.4 |
| 884 | 206 | 1090 | 18.9 | 440 | 4.85 | 38.8 |
| 891 | 199 | 1090 | 18.2 | 320 | 5.55 | 33.0 |
| 902 | 188 | 1090 | 17.3 | 380 | 5.70 | 33.3 |
| 909 | 181 | 1090 | 16.6 | 350 | 6.13 | 31.3 |
| 919 | 171 | 1090 | 15.6 | 325 | 6.66 | 30.2 |
| 929 | 161 | 1090 | 14.8 | 300 | 6.74 | 31.2 |
| 939 | 151 | 1090 | 13.8 | 255 | 7.15 | 30.8 |
| 949 | 141 | 1090 | 12.9 | 270 | 7.90 | 29.1 |
| 959 | 131 | 1090 | 12.0 | 222 | 8.75 | 27.3 |
| 969 | 121 | 1090 | 11.1 | 206 | 10.12 | 26.6 |
| 980 | 110 | 1090 | 10.1 | 152 | 13.09 | 20.5 |
| 992 | 98 | 1090 | 9.0 | 135 | 14.65 | 19.5 |
| 1001 | 89 | 1090 | 8.1 | 130 | 15.65 | 19.4 |
| 1009 | 81 | 1090 | 7.4 | 125 | 15.92 | 20.50 |
| 1021 | 69 | 1090 | 6.3 | 125 | 16.45 | 22.25 |
| 1031 | 59 | 1090 | 5.4 | .......... | .......... | 23.6 |
| 1041 | 49 | 1090 | 4.5 | 102 | 20.24 | 26.8 |
| 1050 | 40 | 1090 | 3.7 | 110 | 19.80 | 29.0 |
| 1060 | 30 | 1090 | 2.8 | 123 | 22.00 | 30.7 |
| 1070 | 20 | 1099 | 1.8 | 67 | 29.68 | |

TABLE 9.—*Tests on Air-lift No.* 4

| No. | Lift, Feet | Submergence, Feet | Total Length Pipe, Feet | Per Cent. Submergence | Cap. at Max. Effi. | Cu. Ft Air at Max Effi. | Maximum Capacity | Cu. Ft. Air at Max. Cap. | Pump, Effi. at Max. Effi. | Pump. Effi. at Max. Cap. |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 846 | 214 | 1060 | 19.9 | 425 | 4.47 | .......... | .......... | 40.0 | |
| 2 | 949 | 141 | 1090 | 12.9 | 246 | 7.76 | .......... | .......... | 29.5 | |
| 3 | 1001 | 89 | 1090 | 7.9 | 153 | 14.90 | .......... | .......... | 20.0 | |

Tables 9 and 10 and Fig. 9 give the results of two tests with this installation. With test No. 1, there was insufficient air to carry the test to the maximum capacity of the discharge pipe.

TABLE 10.—*Tests on Air-lift No. 4*

| | 1 | 2 | 3 |
|---|---|---|---|
| Submergence ($s$) | 214 | 141 | 89 |
| Lift ($l$) | 846 | 949 | 1001 |
| Ratio $\frac{s}{l}$ | 25 3 | 14 9 | 8 9 |
| Gal. water per min | 425 | 246 | 153 |
| Total cu ft. free air at 7250 ft | 2409 | 2431 | 2896 |
| Total cu. ft. free air at sea level | 1900 | 1910 | 2280 |
| Cu. ft. air per gal. (sea level) | 4 47 | 7.76 | 14.90 |
| Per cent. pumping efficiency | 40 0 | 29 5 | 20 00 |
| Velocity water at footpiece | 3 5 | 2 0 | |
| Velocity water at discharge | 1.7 | 1 0 | |
| Velocity air at footpiece | 16.2 | 23 6 | |
| Velocity air at discharge | 73 8 | 74 5 | |
| Pressure of air at submergence | 93 | 61 | |
| Ratio $\frac{s - \text{subpressure}}{l}$ | 22.2 | 12.1 | 6 7 |
| Per cent. submergence | 19.9 | 12.9 | 7 9 |
| Velocity mixture at footpiece | 19 7 | 25 6 | |
| Velocity mixture at discharge | 75 5 | 75 5 | |
| Maximum capacity | | | |
| Cu. ft. air per gal. at max. cap. | | | |
| Sq ft. of pipe, inside surface | 2427 | 2427 | 2427 |
| Pumping effic. at max. cap. | | | |

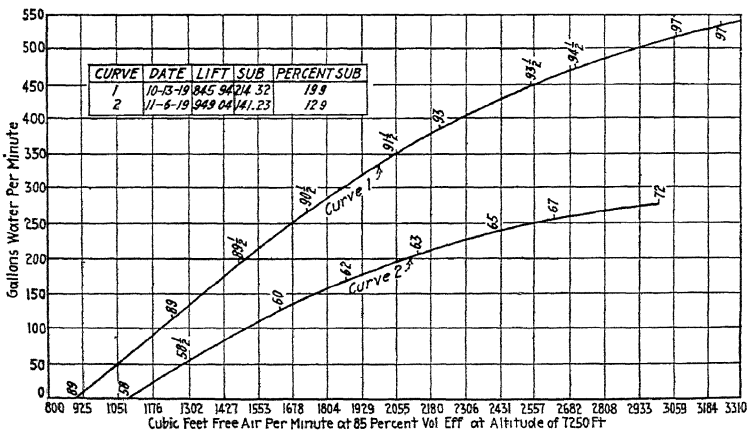


FIG. 9.—CURVES BASED ON DATA OF TABLE 10.

## San Fernando Air-lift

*Delivery Pipe.*—The water delivery pipe consisted ultimately of a 6-in. nipple 3 ft. long at the footpiece, a reducer, 6 to 5-in., 23 in. long, 560 ft. of 5-in. spiral-riveted pipe, a 5-in. nipple 14 in. long below the headpiece, and a 5-in. nipple 24 in. long inside the headpiece, making a total length of 568 ft.

*Air Pipe.*—This was 3-in standard pipe, connected through flexible joints to a 3-in. line 1500 ft in length.

*Footpiece*—The footpiece was an Ingersoll-Rand 6-in. "Imperial," with air admission for a 1½-in. pipe. It was 7 ft. long, the inside pipe being 6½-in. diameter to the throat, where it was 6-in. diameter; the outside pipe was 8⅝-in. diameter.

The headpiece and the carrier were similar to those used with Tiro General No. 1.

TABLE 11.—*San Fernando Air-lift*

| Lift, Feet | Submergence, Feet | Total Length Pipe, Feet | Per Cent Submergence | Operating Cap., Gal |
|---|---|---|---|---|
| 213 | 197 | 410 | 48.0 | 250 |
| 221 | 195 | 416 | 46.8 | 300 |
| 230 | 204 | 434 | 47.1 | 229 |
| 239 | 205 | 444 | 46.4 | 270 |
| 250 | 204 | 454 | 44.8 | 265 |
| 259 | 214 | 473 | 45.1 | 270 |
| 271 | 203 | 474 | 43.2 | 260 |
| 282 | 202 | 484 | 41.7 | 257 |
| 290 | 204 | 494 | 41.3 | 245 |
| 304 | 199 | 503 | 39.7 | 233 |
| 322 | 211 | 533 | 39.6 | 225 |
| 329 | 203 | 532 | 38.3 | 220 |
| 351 | 211 | 562 | 37.6 | 211 |
| 368 | 204 | 572 | 35.8 | 190 |
| 379 | 186 | 565 | 33.7 | 160 |
| 392 | 180 | 572 | 31.4 | 155 |
| 402 | 170 | 572 | 29.8 | 135 |
| 409 | 163 | 572 | 28.6 | 125 |
| 419 | 153 | 572 | 26.6 | 105 |
| 432 | 141 | 573 | 24.6 | 80 |
| 438 | 134 | 572 | 23.4 | 70 |

*Measuring Box.*—This was a steel tank 8 ft. 6 in. long, 3 ft. 2 in. wide, and 3 ft. 8 in. deep, provided with a 90° V-notch weir and with baffles to insure a quiet overflow. Below the discharge, a cylindrical tank 31 in. diameter and 8 ft. 6 in. long was placed in order to check the weir measurements. A Sanborn automatic recording water gage was connected with the measuring box and continuous records were kept

TABLE 12.—*Tests on San Fernando Lift*

| No | Lift, Feet | Submergence, Feet | Total Length Pipe | Per Cent Submergence | Cap at Max Effi | Cu Ft Air at Max Effi | Maximum Capacity | Cu Ft Air at Max Cap | Pump. Effi at Max Effi | Pump Effi at Max Cap |
|---|---|---|---|---|---|---|---|---|---|---|
| 1 | 239 | 214 | 453 | 47 | 278 | 1 14 | 330 | 1 50 | 41 7 | 32 0 |
| 2 | 280 | 214 | 494 | 43 | 237 | 1 60 | 294 | 2 35 | 34 7 | 23 4 |
| 3 | 340 | 212 | 552 | 38 | 194 | 1 89 | 246 | 2 33 | 35 7 | 29 2 |
| 4 | 372 | 200 | 572 | 35 | 166 | 2 43 | 192 | 2 75 | 30 9 | 27 6 |
| 5 | 406 | 166 | 572 | 28 | 96 | 2 7 | 125 | 3 93 | 33 6 | 23 2 |

TABLE 13.—*Tests on San Fernando Air-lift*

| | 1 | 2 | 3 | 4 | 5 |
|---|---|---|---|---|---|
| Submergence (ft.) $s$ | 214 | 214 | 212 | 200 | 166 |
| Lift (ft.) $l$. | 239 | 280 | 341 | 372 | 406 |
| Ratio $\frac{s}{l}$ | 90 0 | 84 0 | 62 0 | 54 0 | 45 0 |
| Gal. of water per min | 278 | 237 | 194 | 166 | 96 |
| Total cu. ft. free air (7250 ft.) | 403 | 481 | 466 | 512 | 331 |
| Total cu ft free air (sea level) | 317 | 379 | 366 | 403 | 260 |
| Cu. ft. per gal. (sea level)... | 1.14 | 1.60 | 1 89 | 2.43 | 2.71 |
| Per cent. pumping efficiency | 41 7 | 34.7 | 35 7 | 30 9 | 33 6 |
| Velocity per second water footpiece | 4 5 | 3 9 | 3.2 | 2 7 | 1 6 |
| Velocity per second water headpiece | 4.5 | 3 9 | 3 2 | 2 7 | 1 6 |
| Velocity per second air footpiece.. ... | 5 3 | 6.3 | 6.2 | 7.1 | 5 4 |
| Velocity per second air headpiece. | 49 | 59 | 57 | 63 | 40 |
| Velocity per second mixture footpiece | 9 8 | 10.2 | 9.4 | 9 8 | 7.0 |
| Velocity per second mixture headpiece | 54 | 63 | 60 | 66 | 42 |
| Air pressure (submergence) | 93 | 93 | 92 | 87 | 72 |
| Per cent. submergence..... | 47 0 | 43.0 | 38.0 | 35 0 | 24 7 |
| Maximum gal per min... | 330 | 294 | 246 | 192 | 125 |
| Cu. ft. per gal., max. cap... | 1.50 | 2 35 | 2.33 | 2.75 | 3.93 |
| Pumping effic. at max cap. | 32.0 | 23.4 | 29.2 | 27.6 | 23.2 |
| Sq. ft. pipe surface (inside) | 604 | 660 | 736 | 764 | 764 |

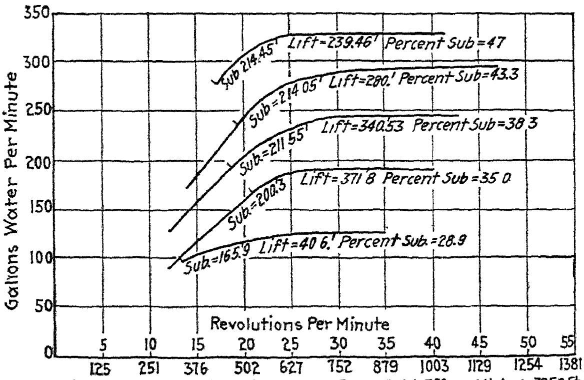


FIG. 10.—CURVES BASED ON DATA OF TABLE 13.

*Operation.*—This air-lift was in operation from March 19 to May 9, during which period the lift ranged from 205 to 438 ft., and operating capacity decreased from 300 to 70 gal. per min. With a lift of 439 ft. and submergence of 134 ft. this air-lift would operate only in pulsations, and it was therefore taken out of commission. The submergence could not be increased, owing to an obstruction in the shaft.

In practice it was found that if an excessive amount of air were admitted to the footpiece, a large proportion would escape below the bottom of the footpiece, due to the fact that the air-admission holes were too small and too few in number. This caused the curves shown in Fig. 10 to flatten very quickly at the upper end.

Table 11 shows the operating capacity of this air-lift at approximately 10-ft. intervals. Tables 12 and 13 and Fig. 10 show the results of tests at various stages of lift. The results confirmed those obtained with Tiro General No. 1 and 2, and indicated that this equipment was unsuited for lifts of over 400 ft. unless a submergence of more than 200 to 250 ft. could be obtained.

## Capacity of Delivery Pipe

Our work at the Tiro General mine has led toward certain tentative conclusions as regards the capacity of water-delivery pipes used in air-lifts. The capacity of pipes will vary with four factors: The diameter (or area) of the pipe; the submergence; the lift; and the quantity of air admitted to the footpiece.

*Diameter.*—The capacities of water columns at zero lift and given submergence vary directly as the area of the pipe. As the lift increases, the difference in capacities between pipes of different diameters remains constant. From our experiments it would appear that the most efficient capacity of a pipe with submergence of 215 ft. is approximately 25 gal. per sq. in. of area per min. Fig. 11 gives graphically the most efficient capacity of pipes of various diameters, assuming a constant submergence of 215 feet.

Since the No. 3 and No. 4 air-lifts used water columns of varying diameter, it was not possible to make close comparisons of these with the 5-in., 6-in. and 6⅝-in. air-lifts. However, rough comparisons appear to indicate that the capacity of a water column of varying diameter is equal approximately to the capacity of a pipe having an area of 85 per cent. of the average area of the varying column.

*Submergence.*—The capacity of a pipe of given diameter, at constant lift, will vary with the submergence, within certain limits. By referring to the curves in Fig. 5, it will be seen that where there is a considerable increase in submergence, the capacity of the pipe increases. Experiments tended to indicate that if equal lengths be added to the submergence and

to the lift, the capacity would remain practically constant; however, the range covered by these observations was too limited to draw any positive conclusions.

*Lift.*—The capacities of the water columns employed, corresponding to variations in lift of approximately 10 ft., are given in Tables 1, 4, 7, 8 and 11. With a water column of given diameter and submergence, it is seen that the capacity, whether it be the efficient capacity, operating capacity, or maximum capacity, varies inversely with the lift. The curves in Figs. 4 and 5 illustrate this point very clearly

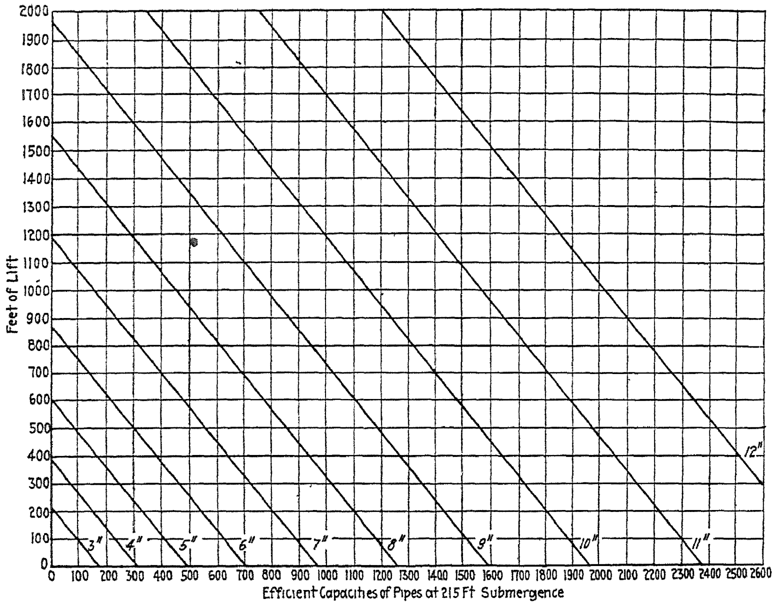


FIG. 11.—EFFICIENT CAPACITIES OF PIPES AT 215 FT. SUBMERGENCE.

*Quantity of Air.*—The quantity of air admitted to the footpiece affects the capacity of a water column of given diameter, lift, and submergence; within certain limits there is a point where no discharge of water will result. If the air is further increased, the quantity of water will increase until a limit is reached, beyond which no increase in water will result, however great the quantity of air; this we have termed the point of maximum capacity. This is shown in Figs. 4, 6 and 10.

Experiment was made with No. 3 air-lift, at a time when the lift was 840 ft. and the submergence 192 ft., by admitting air up to 1250 cu. ft. of free air per min., up to which point there was no flow of water. Admission of a greater quantity of air caused water to discharge, which increased as indicated in Fig. 8. This lift continued to operate in pulsations, and

it was impossible to secure a steady flow with the greatest quantity of air admitted, that is, up to 3000 cu. ft. per minute.

A similar experiment was made with No. 4, having submergence of 200 ft. and lift of 840 ft.; no flow of water resulted with air up to 950 cu. ft. free air. Beyond that point, the water issued in pulsations until 2700 cu. ft. of air was admitted, when it became steady. These results are plotted in Fig. 9. The curves in Figs. 4, 6 and 10 indicate clearly the points at which the maximum capacities were reached with 5-in., 6-in. and 6⅝-in. water delivery pipes.

It would be impracticable to keep compressors operating at all times at such speed as to maintain a flow of water at the exact point of maximum efficiency, especially where the lift and submergence are constantly changing, as in unwatering a mine. Moreover, the rapid lowering of the water level is usually of more importance than decreased efficiency. However, if the installation is well designed, it is probable that the capacity can be maintained within a reasonable distance of the most efficient capacity.

## Quantity of Air Consumed

The quantity of air, on sea-level basis, actually consumed per gallon of water raised at the Tiro General mine at the various stages of lift is noted in Tables 1, 2, 4, 5, 7, 8, and 12. It will be seen that the quantity of free air consumed per gallon of water raised is considerably greater during daily operation than when pumping at highest efficiency, as shown in the various tests. It was desired to unwater the mine as quickly as possible; hence Nos. 1 and 2 and the San Fernando air-lift were run at a capacity greater than the efficient capacity of the water delivery line.

Conditions vary so widely between lifts of 140 and 900 ft. that it would be difficult to design an air-lift that would operate at highest efficiency at all stages of the lift. However, for the conditions of lift noted, where the capacity of the compressors was limited to 3000 cu. ft. of free air, No. 4 air-lift was probably as nearly the most efficient design as could be secured. Fig. 12 shows the relations between the lift and the quantity of free air required per gallon of water for air-lifts Nos. 1, 2 and 4.

## Theoretical Quantity of Air Required

The work done by air in expanding follows the equation $W = PdV$, which, when integrated becomes:

$$W = P_2V_2 \text{ Napierian log } \frac{P_1}{P_2}$$

in which $P_1$ = greatest pressure; $P_2$ = least pressure; $V_1$ = least volume; and $V_2$ = greatest volume.

Let $P_1$ = absolute pressure at which air enters footpiece;
$P_2$ = absolute pressure at which air leaves headpiece or water discharge;

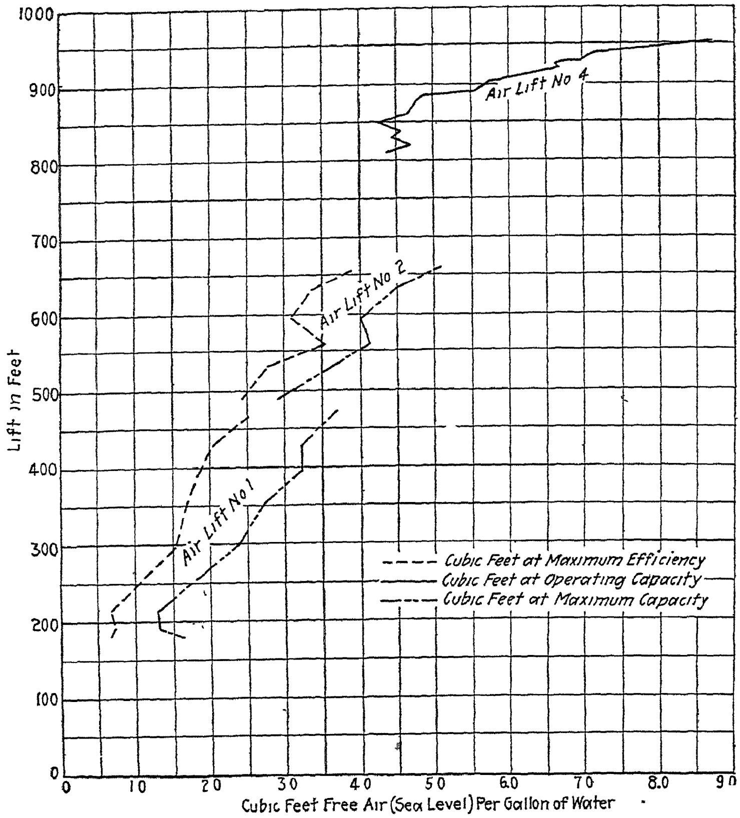


FIG. 12.—AIR CONSUMPTION PER GALLON OF WATER.

$V_1$ = volume of 1 cu. ft. free air at footpiece, when compressed to pressure of $P_1$ lb.;
$V_2$ = volume of 1 cu. ft. free air;
$L$ = lift, in feet, to which water is raised;
$S$ = submergence, in feet.

Since the weight of water is so great in comparison with that of the air consumed, the expansion can be considered as isothermal. It will be also assumed that $P_2$ = air at atmospheric pressure.

The work, in foot-pounds, performed by 1 cu. ft. of free air at a pressure of $P_1$ lb., in expanding from volume $V_1$ to $V_2$, is, therefore,

$$W = 144\, P_2 V_2 \log_e \frac{P_1}{P_2}$$

Disregarding friction and losses, the work performed in raising 1 gal. of water $L$ ft. is $8.35 \times L$ ft.-lb. If $X$ = cubic feet of free air, at sea level, required to raise 1 gal. of water $L$ ft., where the submergence is $S$ ft., and where $P_1$ is the pressure required to balance $S$ ft. of water,

$$X\left(144\, P_2 V_2 \log_e \frac{P_1}{P_2}\right) = 8.35 \times L, \text{ or}$$

$$X = \frac{8.35 \times L}{144\, P_2 V_2 \log_e \frac{P_1}{P_2}} = \frac{0.058 \times L}{P_2 V_2 \log_e \frac{P_1}{P_2}} = \frac{L}{254 \log_e \frac{P_1}{P_2}} \qquad (1)$$

From this formula, Table 14 has been calculated, giving the number of cubic feet of free air, at sea level, theoretically necessary to raise 1 gal. of water through a lift of $L$ ft., with a submergence of $S$ ft., where all losses are neglected. The two lines of figures at the head of the table are respectively the submergence, in feet, and the corresponding hydrostatic pressure, in pounds per square inch. If the lift, the submergence, and the cubic feet of free air consumed by a given installation are known, its efficiency can readily be computed by comparing with the theoretical quantity of free air required, as given in Table 14. In order to develop a formula that will include all the known factors that enter into the work that is being done, let:

$W$ = energy, in foot-pounds, in 1 cu. ft. free air compressed to pressure necessary to overcome submergence, assuming isothermal expansion;
$X$ = cubic feet of free air required to raise 1 gal. of water $L$ ft., with submergence of $S$ ft.;
$W_w$ = work in raising 1 gal. of water $L$ ft., or $8.35 \times L$;
$L_w$ = loss due to friction of water in water pipe;
$L_a$ = loss due to friction of air in water pipe;
$L_s$ = loss due to slippage of water in water pipe, also other unknown losses;
$W_a$ = loss of energy in air at discharge.

$$\text{Then } XW = (W_w + L_w + L_a + L_s + W_a) \text{ or}$$

$$X = \left[\frac{(8.35 \times L + L_w + L_a + L_s + W_a)}{W}\right] \qquad (2)$$

In this formula, the factors $L$, $W$, and $W_a$ are determinable by observation or by computation. No data exist for determining $L_w$, $L_a$, or $(L_w + L_a)$, but we will tentatively assume these, as being the sum of the

**TABLE 14.—*Quantity of Free Air, Sea-level Basis, at Given Pressure, Theoretically Required to Raise 1 Gal. of Water to a Given Height***

| Lift in Feet | 115′ | 127′ | 138′ | 150′ | 161′ | 173′ | 184′ | 196′ | 207′ | 219′ | 230′ | 242′ | 253′ | 265′ | 276′ | 288′ | 300′ | 311′ | 323′ | 334′ | 346′ | 357′ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 50 | 55 | 60 | 65 | 70 | 75 | 80 | 85 | 90 | 95 | 100 | 105 | 110 | 115 | 120 | 125 | 130 | 135 | 140 | 145 | 150 | 155 |
| 10 | 0.03 | 0.03 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 |
| 20 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.04 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 | 0.03 |
| 30 | 0.08 | 0.08 | 0.07 | 0.07 | 0.07 | 0.06 | 0.06 | 0.06 | 0.06 | 0.06 | 0.06 | 0.06 | 0.06 | 0.06 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 |
| 40 | 0.11 | 0.10 | 0.10 | 0.09 | 0.09 | 0.09 | 0.09 | 0.08 | 0.08 | 0.08 | 0.08 | 0.08 | 0.07 | 0.07 | 0.07 | 0.07 | 0.07 | 0.07 | 0.07 | 0.07 | 0.07 | 0.06 |
| 50 | 0.13 | 0.13 | 0.12 | 0.12 | 0.11 | 0.11 | 0.11 | 0.10 | 0.10 | 0.10 | 0.10 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.09 | 0.08 | 0.08 | 0.08 | 0.08 |
| 60 | 0.16 | 0.15 | 0.14 | 0.14 | 0.14 | 0.13 | 0.13 | 0.12 | 0.12 | 0.12 | 0.12 | 0.11 | 0.11 | 0.11 | 0.11 | 0.11 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 | 0.10 |
| 80 | 0.21 | 0.20 | 0.19 | 0.19 | 0.18 | 0.17 | 0.17 | 0.16 | 0.16 | 0.16 | 0.15 | 0.15 | 0.15 | 0.15 | 0.14 | 0.14 | 0.14 | 0.14 | 0.13 | 0.13 | 0.13 | 0.13 |
| 100 | 0.27 | 0.25 | 0.24 | 0.23 | 0.23 | 0.22 | 0.21 | 0.21 | 0.20 | 0.20 | 0.19 | 0.19 | 0.18 | 0.18 | 0.18 | 0.18 | 0.17 | 0.17 | 0.17 | 0.17 | 0.16 | 0.16 |
| 120 | 0.32 | 0.30 | 0.29 | 0.28 | 0.27 | 0.26 | 0.25 | 0.25 | 0.24 | 0.24 | 0.23 | 0.23 | 0.22 | 0.22 | 0.21 | 0.21 | 0.21 | 0.20 | 0.20 | 0.20 | 0.20 | 0.19 |
| 140 | 0.37 | 0.35 | 0.34 | 0.33 | 0.32 | 0.31 | 0.30 | 0.29 | 0.28 | 0.27 | 0.27 | 0.26 | 0.26 | 0.25 | 0.25 | 0.25 | 0.24 | 0.24 | 0.23 | 0.23 | 0.23 | 0.23 |
| 160 | 0.43 | 0.41 | 0.39 | 0.38 | 0.36 | 0.35 | 0.34 | 0.33 | 0.32 | 0.31 | 0.31 | 0.30 | 0.30 | 0.29 | 0.29 | 0.28 | 0.28 | 0.27 | 0.27 | 0.26 | 0.26 | 0.26 |
| 180 | 0.48 | 0.46 | 0.44 | 0.42 | 0.41 | 0.39 | 0.38 | 0.37 | 0.36 | 0.35 | 0.35 | 0.34 | 0.33 | 0.33 | 0.32 | 0.32 | 0.31 | 0.31 | 0.30 | 0.30 | 0.30 | 0.29 |
| 200 | 0.53 | 0.51 | 0.49 | 0.47 | 0.45 | 0.44 | 0.42 | 0.41 | 0.40 | 0.39 | 0.38 | 0.38 | 0.37 | 0.36 | 0.36 | 0.35 | 0.35 | 0.34 | 0.34 | 0.33 | 0.33 | 0.32 |
| 250 | 0.67 | 0.63 | 0.61 | 0.59 | 0.57 | 0.55 | 0.53 | 0.51 | 0.50 | 0.49 | 0.48 | 0.47 | 0.46 | 0.45 | 0.45 | 0.44 | 0.43 | 0.43 | 0.42 | 0.41 | 0.41 | 0.40 |
| 300 | 0.80 | 0.76 | 0.73 | 0.70 | 0.68 | 0.66 | 0.63 | 0.62 | 0.60 | 0.59 | 0.58 | 0.56 | 0.55 | 0.54 | 0.54 | 0.53 | 0.52 | 0.51 | 0.50 | 0.50 | 0.49 | 0.48 |
| 350 | 0.93 | 0.87 | 0.85 | 0.82 | 0.79 | 0.77 | 0.74 | 0.72 | 0.70 | 0.69 | 0.68 | 0.66 | 0.65 | 0.64 | 0.63 | 0.61 | 0.60 | 0.60 | 0.59 | 0.58 | 0.57 | 0.56 |
| 400 | 1.07 | 1.01 | 0.97 | 0.94 | 0.90 | 0.87 | 0.84 | 0.82 | 0.80 | 0.79 | 0.77 | 0.76 | 0.74 | 0.73 | 0.71 | 0.70 | 0.69 | 0.68 | 0.67 | 0.66 | 0.65 | 0.65 |
| 450 | 1.20 | 1.14 | 1.09 | 1.06 | 1.02 | 0.98 | 0.95 | 0.93 | 0.90 | 0.88 | 0.86 | 0.85 | 0.83 | 0.82 | 0.80 | 0.79 | 0.78 | 0.77 | 0.75 | 0.74 | 0.73 | 0.73 |
| 500 | 1.33 | 1.27 | 1.21 | 1.17 | 1.13 | 1.09 | 1.06 | 1.03 | 1.01 | 0.98 | 0.96 | 0.95 | 0.92 | 0.91 | 0.89 | 0.88 | 0.86 | 0.85 | 0.84 | 0.83 | 0.82 | 0.81 |
| 550 | 1.47 | 1.39 | 1.34 | 1.29 | 1.24 | 1.20 | 1.16 | 1.13 | 1.11 | 1.08 | 1.05 | 1.04 | 1.02 | 1.00 | 0.98 | 0.96 | 0.94 | 0.94 | 0.92 | 0.91 | 0.90 | 0.89 |
| 600 | 1.60 | 1.52 | 1.46 | 1.41 | 1.36 | 1.31 | 1.27 | 1.24 | 1.21 | 1.18 | 1.15 | 1.13 | 1.11 | 1.09 | 1.07 | 1.05 | 1.03 | 1.03 | 1.01 | 0.99 | 0.98 | 0.97 |
| 650 | 1.73 | 1.65 | 1.58 | 1.52 | 1.47 | 1.42 | 1.37 | 1.34 | 1.31 | 1.29 | 1.24 | 1.23 | 1.20 | 1.18 | 1.16 | 1.14 | 1.12 | 1.10 | 1.09 | 1.08 | 1.06 | 1.05 |
| 700 | 1.87 | 1.77 | 1.70 | 1.64 | 1.58 | 1.53 | 1.48 | 1.44 | 1.41 | 1.37 | 1.34 | 1.32 | 1.29 | 1.27 | 1.25 | 1.23 | 1.21 | 1.20 | 1.17 | 1.16 | 1.14 | 1.13 |
| 750 | 2.00 | 1.90 | 1.82 | 1.76 | 1.70 | 1.64 | 1.58 | 1.54 | 1.51 | 1.47 | 1.44 | 1.42 | 1.39 | 1.36 | 1.34 | 1.31 | 1.29 | 1.28 | 1.26 | 1.24 | 1.22 | 1.21 |
| 800 | 2.13 | 2.03 | 1.94 | 1.88 | 1.81 | 1.75 | 1.69 | 1.65 | 1.61 | 1.57 | 1.53 | 1.51 | 1.48 | 1.45 | 1.43 | 1.40 | 1.38 | 1.36 | 1.34 | 1.32 | 1.30 | 1.29 |
| 850 | 2.26 | 2.15 | 2.08 | 1.99 | 1.92 | 1.86 | 1.79 | 1.75 | 1.71 | 1.67 | 1.63 | 1.61 | 1.57 | 1.54 | 1.52 | 1.49 | 1.47 | 1.45 | 1.42 | 1.41 | 1.39 | 1.37 |
| 900 | 2.39 | 2.28 | 2.19 | 2.11 | 2.04 | 1.97 | 1.90 | 1.85 | 1.81 | 1.77 | 1.73 | 1.69 | 1.66 | 1.63 | 1.61 | 1.58 | 1.55 | 1.53 | 1.51 | 1.49 | 1.47 | 1.45 |
| 950 | 2.52 | 2.41 | 2.31 | 2.23 | 2.15 | 2.07 | 2.01 | 1.96 | 1.91 | 1.87 | 1.82 | 1.79 | 1.76 | 1.72 | 1.69 | 1.67 | 1.64 | 1.62 | 1.59 | 1.57 | 1.55 | 1.53 |
| 1000 | 2.66 | 2.53 | 2.40 | 2.34 | 2.26 | 2.18 | 2.11 | 2.06 | 2.01 | 1.96 | 1.92 | 1.88 | 1.85 | 1.82 | 1.78 | 1.75 | 1.73 | 1.70 | 1.68 | 1.65 | 1.63 | 1.61 |

combined losses from the two sources when computed separately. No data exist for determining $L_s$, and until more is known of this factor it will be assumed that this accounts for all losses not previously included, which losses may be taken as a percentage of the work done under various conditions.

A gage, connected for several weeks close to the water discharge of an air-lift, showed a pressure occasionally rising to 5 lb. but normally zero. For all practical purposes, this factor can therefore be neglected.

A formula that has been employed as a basis for air consumption is developed as follows:

Let $S$ = number of feet of submergence;
$L$ = number of feet of lift;
$P_2$ = pressure of air at discharge, that is, atmospheric pressure;
$v$ = number of cubic feet of air necessary to be admitted to foot-piece, under absolute pressure equivalent to $S$ feet of submergence plus atmospheric pressure, necessary to discharge 1 cu. ft. of water;
$V$ = number of cubic feet of free air equivalent to $v$.

If the weight of $v$ cu. ft. of air is neglected, the specific gravity of 1 cu. ft. of water and $v$ cu. ft. of air will be: $1 \div (1 + v)$.

The pressure of the mixture in the delivery pipe corresponds to $S$ ft. of water + $P_2$ at the footpiece, and $P_2$ at the discharge. Between the footpiece and the discharge, the pressure per square foot, at any point $x$, will be equal to the weight of a column of the mixture having a cross section of 1 sq. ft. and a height extending from $x$ to the discharge, plus atmospheric pressure.

Let $x$ = lift, in feet, between the point $x$ and the discharge;
$wL_x$ = weight of column of mixture, measured in feet of water.

Then
$$wL_x = \int_0^x \frac{wdx}{1 + v_x}.$$

Differentiating for $wL_x$, we have,
$$wdL_x = \frac{wdx}{1 + v_x}, \text{ or } dL_x = \frac{dx}{1 + v_x}, \text{ or } dx = dL_x\,(1 + v_x)$$

Since $v_x : V = (x + P_2) : (L_x + P_2)$
$$v_x = \frac{v(S + P_2)}{(L_x + P_2)}$$

Substituting for $v_x$, we have
$$dx = dL_x\left[1 + \frac{v(S + P_2)}{(L_x + P_2)}\right] = dL_x + dL_x\left[\frac{v(S + P_2)}{(L_x + P_2)}\right]$$

Therefore $x = L_x + v(S + P_2)\log_e (L_x + P_2) + c$

and $c = x - L_x - v(S + P_2)\log_e (L_x + P_2)$.

When $x = c$, $L_x = 0$; and when $x = S + L$, $L_x = S$

and

$$S + L = S + v(S + P_2) \log_e \left(\frac{S + P_2}{P_2}\right)$$

therefore

$$L = v(S + P_2) \log_e \left(\frac{S + P_2}{P_2}\right)$$

and

$$v = \frac{L}{(S + P_2) \log_e \left(\frac{S + P}{P_2}\right)}.$$

But

$$V = \frac{v(S + P_2)}{P_2} = \frac{L}{P_2 \log_e \left(\frac{S + P_2}{P_2}\right)}$$

Assuming $P_2$ at sea level,

$$V = \frac{L}{34 \log_e \left(\frac{S + 34}{34}\right)}$$

Since 1 cu. ft. contains 7.48 gal..

$$X = \frac{V}{7.48} = \frac{L}{255 \log_e \left(\frac{S + 34}{34}\right)} \tag{3}$$

This equation gives the theoretical quantity of free air required at sea level, to raise 1 gal. of water $L$ ft., the results being the same as those from equation 1.

A formula employed for actual operations by the Ingersoll-Rand Co., suggested in part by E. A. Rix, is as follows (using symbols as previously):

$$X = 0.8 \left[\frac{L}{C \times \log \left(\frac{S + 34}{34}\right)}\right] \tag{4}$$

where $C$ is a factor depending on the lift, as follows:

| Lift in Feet | $C$ |
|---|---|
| 10 to 60 | 245 |
| 61 to 200 | 233 |
| 201 to 500 | 216 |
| 501 to 650 | 185 |
| 651 to 750 | 156 |

The following formula, used by the Sullivan Machinery Co., is taken from Peele's "Mining Engineers' Handbook."

$$X = \frac{L}{292.5 \log \left(\frac{S + 34}{34}\right)} \tag{5}$$

It will be seen that equation 5 is probably derived from equation 3. A table is given in Peele's Handbook, based on equation 5, which assumes

an efficiency of 50 per cent. in actual operation. This relation will be evident when comparing with equation 3.

A formula employed by Frank Richards, in which a factor is introduced, which varies with the lift, according to observed results, is given as follows:

$$X = \frac{L}{C\left(\frac{\log S + 34}{34}\right)} \tag{6}$$

where $C$ is as follows:

| LIFT, FEET | C | LIFT, FEET | C |
|---|---|---|---|
| 100 | 245 | 550 | 194 5 |
| 150 | 240 | 600 | 189 |
| 200 | 235 | 650 | 184 5 |
| 250 | 228 5 | 700 | 178 |
| 300 | 222 | 750 | 172 5 |
| 350 | 216 5 | 800 | 167 |
| 400 | 211 | 850 | 161.5 |
| 450 | 205.5 | 900 | 156 |
| 500 | 200 | 1000 | 143 |

TABLE 15.—*Cubic Feet of Free Air, at Sea Level, Required to Raise 1 Gal. of Water (Frank Richards)*

| Lift, Feet | Per Cent. Submergence | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | 25 | 30 | 35 | 40 | 45 | 50 | 55 | 60 | 65 | 70 |
| ·20 | | | | | | 0 39 | 0.33 | 0 29 | 0 25 | 0.22 |
| 30 | | | | | 0.50 | 0 43 | 0.37 | 0 32 | 0.28 | 0 25 |
| 40 | | | | | 0 54 | 0 47 | 0 41 | 0 36 | 0.32 | 0 28 |
| 50 | | | | | 0.58 | 0.51 | 0 45 | 0 40 | 0.34 | 0.30 |
| 60 | | | | 0 73 | 0.62 | 0.55 | 0.48 | 0 43 | 0.38 | 0.34 |
| 80 | | | | 0 78 | 0 69 | 0.62 | 0.55 | 0 49 | 0.44 | 0.38 |
| 100 | | | | 0 86 | 0.76 | 0 68 | 0.62 | 0.56 | 0 50 | |
| 125 | | | | 0 95 | 0 86 | 0.77 | 0.70 | 0 63 | 0.58 | |
| 150 | | | | 1.05 | 0.93 | 0.85 | 0 77 | 0 71 | | |
| 175 | | | 1.28 | 1.14 | 1.03 | 0.93 | 0.85 | 0.78 | | |
| 200 | | | 1 37 | 1 23 | 1.11 | 1.02 | 0.93 | | | |
| 250 | | | 1.57 | 1.42 | 1.30 | 1.18 | 1.09 | | | |
| 300 | | | 1.78 | 1.60 | 1.47 | 1.36 | | | | |
| 350 | | | 1.97 | 1.81 | 1.65 | 1.53 | | | | |
| 400 | | | 2.19 | 2.00 | 1.84 | 1.70 | | | | |
| 450 | | 2 66 | 2.40 | 2 21 | 2.06 | 1.90 | | | | |
| 500 | 3 25 | 2 86 | 2.66 | 2.42 | 2.24 | | | | | |
| 550 | 3.52 | 3 15 | 2 88 | 2.64 | 2.45 | | | | | |
| 600 | 3.76 | 3.36 | 3.10 | 2.88 | 2.68 | | | | | |
| 650 | 4.08 | 3.67 | 3 36 | 3.12 | 2.90 | | | | | |
| 700 | 4.39 | 3.95 | 3.62 | 3.37 | 3.14 | | | | | |
| 750 | 4.74 | 4 28 | 3.96 | 3 64 | | | | | | |

Equation 5 approximated the results obtained with the first three air-lifts employed at the Tiro General mine more closely than any of the other formulas given; the observed air consumptions were from 5 to 10 per cent. lower than those given by equation 5.

Table 15, calculated from equation 5, gives the number of cubic feet of free air required to raise 1 gal. of water with given lifts and percentages of submergence.

### Submergence

In a shaft of given depth, the submergence of any air-lift installation is limited, on the one hand, to the pressure obtainable at the compressor and, on the other hand, to the depth to which it is desired to lower the water. At most mines, the compressors are designed for about 100 lb. gage pressure, corresponding to approximately 230 ft. of submergence, disregarding the friction loss in the air pipe. This loss may become important if a large volume of air has to be transmitted to the footpiece through a pipe of small diameter.

Ivens[1] shows by test that, for a water-delivery pipe of 10 in. diameter and lift of 37 ft., the most efficient submergence is 70 per cent.

Our experience indicates that in unwatering a mine, where the lift is constantly increasing and submergence remaining practically constant, the curve showing the relation between gallons raised per minute and feet of lift will be practically a straight line, at approximately 50° to the abscissa, see Fig. 5. If the submergence is increased or decreased, the line will be raised or depressed. If, with every increase in the lift there is a corresponding increase in depth of submergence, it will be found that this line will be vertical or nearly so. Further experiments would be necessary to determine the percentage of increase in submergence necessary to maintain the most efficient pumping capacity where the lift is being increased.

### Efficiency

If there were no losses due to friction, slippage, and other unknown factors, the quantity of air, at sea level, required to raise 1 gal. of water would be those shown in Table 14, for given lift and submergence. The velocity of the air at the footpiece, in the tests made, varied from 2.4 to 10.8 ft. per sec., and on reaching the headpiece the velocity varied from 25 ft. to 93 ft. per sec.; these velocities do not consider the air and water as mixed. The velocity of the mixture of air and water at the footpiece varied from 6.9 to 13.7 ft. per sec., and on reaching the headpiece it varied from 29 to 97 ft. per sec. There is a certain friction loss depending upon the velocity of the mixture and the inside area of the pipe, the exact influence of which is not known at present. It is probable that this loss

[1] E M Ivens "Pumping by Compressed Air, 120." N. Y., 1914. Wiley.

is that of neither air nor water alone, and also that it increases as the square of the velocity.

It is suggested by Prof. Elmo G. Harris that the loss due to slippage of the water below the ascending bubbles of air is a considerable loss; this is also an undetermined factor.

There will be an apparent gain or loss in efficiency depending upon whether the water is at a higher or lower temperature than the air admitted to the footpiece; but this is probably a small factor since the air is probably cooled, while passing through the submerged air pipe, to a temperature closely approximating that of the water.

Reference to Table 2 shows the pumping efficiencies of the 5-in. air-lift to range from 42 to 30 per cent., the efficiency apparently decreasing somewhat as the length of pipe line increases; the efficiency of the 6-in. air-lift (Table 5) ranges from 62 to 35 per cent., also decreasing as the length of pipe line increases; the efficiency of the 6⅝-in. air-lift (Table 12) ranges from 38.8 to 33 per cent., the decrease being too small and uncertain to warrant any deductions. The water measurements of No. 3 air-lift were too uncertain to permit definite calculation of efficiency, owing to the lift operating in pulsations, but, in general, the efficiency probably never fell below 35 per cent. At the beginning of operations with the No. 3 installation, the lift being 548 ft. and submergence 230 ft., a continuous flow was obtained, with pumping efficiency of 44.2 per cent. This would indicate that the efficiency of a pipe of large and gradually expanding diameter is greater than that of a pipe of small and uniform diameter.

Our experience led to the conclusion that it is better to use efficiency percentages in direct connection with Table 14, rather than tables in which are introduced certain factors as in equations 3, 4, 5 and 6. The percentage factors can be employed to cover all classes of air-lifts, since the efficiencies vary with the design, whereas equations 3, 4, 5 and 6 do not take into consideration any differences in design or in sizes of pipe.

Our work has led us to adopt the following generalizations:

1. The pumping efficiency of a given installation may decrease somewhat as the lift increases while submergence remains constant, but should not decrease very greatly.

2. If the submergence is decreased, the pumping efficiency will decrease.

3. The pumping efficiency is somewhat higher in pipes of large diameter than in those of small diameter and is probably somewhat higher in pipes expanding upwards.

In order to ascertain the point of maximum efficiency, tests were made beginning with admission of air just sufficient to start the water flowing, then enlarging this amount gradually until no further increase in flow of water could be obtained. Curves of these tests were then plotted, as

shown in Figs. 4, 6 and 10. If a straight line is drawn through the origin of coördinates, tangent to the curve, the point of maximum efficiency for the set of conditions involved in this curve will be located. It has been our experience that at this point the gage pressure, allowing for friction in the air line, is practically that necessary to balance the submergence.

If the water level is gradually lowered while the depth of submergence remains practically constant, a periodical plotting of these curves will reveal the fact that the quantity of water that can be raised, whether at maximum efficient capacity, operating capacity, or maximum capacity, gradually diminishes. It is therefore important, in designing a lift for unwatering a mine, to make the pipe line of sufficient size to handle the desired quantity of water when approaching the final stage of unwatering.

The efficiencies attained in air-lift pumping are not so high as those reached by centrifugal pumps, especially when the latter pumps are in good condition and working under proper head and at proper capacity; nevertheless we believe that the over-all efficiency of an air-lift will, in many cases, be found approximately equal to that of any form of pumping apparatus. Labor costs are reduced to a minimum; with the air-lift once properly installed, no labor is required except the engineer in the compressor plant. One very great advantage over all other forms of pumping apparatus is that the air-lift is in no danger of being drowned by a suddenly increased flow of water.

As air-lift pumping becomes better understood, it is quite likely that installations will be made in which 18 to 24-in. boreholes will be sunk at suitable points along the workings, with such depth as to provide submergence that will result in the highest working efficiency. Connections with all the levels in which there is a flow of water would then enable the air-lift to do the whole pumping from the mine. By locating such a hole close to where a shaft is being sunk in wet ground, the cost of shaft-sinking can be much reduced. The air-lift will show a much higher efficiency than a pump driven by compressed air; it is often found that considerable pumping is done, in mines, by compressed air where it is not practicable to carry steam lines.

In case of such a permanent installation, a bore hole could be driven which would suffice for all future requirements. One of 24-in. diameter to a depth of 1500 ft. would probably suffice to admit pipe sufficient to discharge up to 2000 gal. per min. with a lift of 1000 to 1200 ft. In making such an installation, in order to be certain of having a proper balance to insure the highest efficiency, careful study should be made to determine the proper compressor capacity, proper sizes of air and water pipes, and correct design of the water pipe. If the water is heavily charged with acid, the water pipe should be of wood wrapped with copper wire, and the air pipe should be of wood, copper or brass.

## TABLE 16.—*Daily Air-lift Data*

| Date, 1919 | Level Below Collar | Lowered, Feet | Total Gallons | Cubic Feet Air per Gallon | Hours Operation | Number Revolutions | Fuel Oil, Gallons | Average Air Pressure | Average Steam Pressure | Lift, Feet | Submergence, Feet | Gallons per Minute | Per Cent Submergence | Pump Efficiency |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Feb. 10 | 146.0 | 1.7 | 328,500 | 2.58 | 18¼ | 43,800 | 1,730 | 87 | 99 | 154.4 | 192.3 | 300 | 55.5 | |
| 15 | 149.13 | 0.30 | 224,000 | 1.43 | 13⅛ | 16,130 | 2,205 | 80 | 95 | 157.5 | 189.7 | 280 | 54.6 | |
| Mar. 1 | 168.36 | 2.93 | 626,400 | 0.98 | 24 | 31,208 | 1,700 | 81 | 102 | 181.6 | 190.4 | 435 | 51.2 | |
| 8 | 185.13 | 1.94 | 598,000 | 0.84 | 21⅜ | 25,773 | 2,565 | 82 | 94 | 198.4 | 191.7 | 460 | 49.1 | |
| 15 | 198.52 | 1.80 | 705,600 | 1.05 | 24 | 37,469 | 1,615 | 75 | 97 | 211.8 | 178.4 | 490 | 45.7 | |
| Apr. 1 | 263.52 | 2.53 | 832,160 | 1.16 | 18⅜ | 48,869 | 3,078 | 91½ | 95 | 269.0 | 215.5 | 743 | 44.5 | |
| 12 | 328.00 | 7.12 | 921,600 | 1.63 | 24 | 75,900 | 3,420 | 96 | 98 | 335.0 | 224.5 | 640 | 40.2 | |
| May 1 | 431.11 | 4.16 | 568,800 | 2.90 | 24 | 83,604 | 2,650 | 103 | 100 | 440.0 | 223.0 | 395 | 33.0 | |
| 15 | 451.15 | 2.23 | 416,100 | 2.93 | 23¾ | 61,794 | 3,591 | 104 | 92 | 468.43 | 246.43 | 292 | 34.5 | |
| Jun. 1 | 515.86 | 4.76 | 489,600 | 2.90 | 24 | 71,660 | 2,000 | 98 | 98 | 537.00 | 213.00 | 340 | 28.4 | 37.0 |
| 15 | 544.78 | 0.95 | 340,200 | 3.06 | 21 | 52,638 | 3,847 | 108 | 96 | 568.0 | 213.0 | 270 | 27.3 | 36.6 |
| Jul 1 | 591.43 | 3.06 | 408,980 | 3.30 | 23⅚ | 68,408 | 2,050 | 108 | 102 | 613.4 | 228.3 | 286 | 27.1 | 35.5 |
| 15 | 622.75 | 2.03 | 381,600 | 3.85 | 24 | 74,387 | 1,537 | 110 | 102 | 645.2 | 228.3 | 265 | 25.9 | |
| | Operations discontinued July 24 to install 8¼ in. lift | | | | | | | | | | | | | |
| Aug. 19 | 526.45 | 3.65 | 573,000 | 1.08 | 9½ | 48,031 | 1,539 | 98 | 120 | 548.3 | 229.9 | 900 | 29.5 | |
| Sept. 1 | 638.13 | 1.47 | 882,000 | 2.01 | 21 | 89,900 | 2,632 | 106 | 117 | 659.14 | 253.06 | 700 | 27.7 | |
| 15 | 698.48 | 3.80 | 598,500 | 2.90 | 21 | 87,760 | 1,881 | 108 | 120 | 718.3 | 253.4 | 475 | 26.1 | |
| Oct. 1 | 785.82 | 7.08 | 769,950 | 3.41 | 21¾ | 130,348 | 2,760 | 97 | 118 | 804.0 | 219.9 | 590 | 21.5 | |
| 7 | Operations suspended 7.30 A. M. Oct. 6 to change No. 3 to No. 4 air-lift | | | | | | | | | | | | | |
| 15 | 836.26 | 5.77 | 597,600 | 4.38 | 24 | 131,806 | 3,078 | 102 | 123 | 852.9 | 237.3 | 415 | 21.7 | 37.2 |
| Nov. 1 | 921.47 | 0.74 | 337,005 | 7.42 | 19 | 124,465 | 2,736 | 71 | 122 | 940.7 | 149.6 | 292 | 13.7 | 29.8 |
| 15 | 940.92 | 1.21 | 309,600 | 9.38 | 24 | 143,996 | 3,591 | 63 | 124 | 958.9 | 131.4 | 215 | 12.0 | |
| Dec. 21 | 1012.80 | 0.00 | 151,840 | 18.8 | 23⅜ | 143,894 | 4,160 | — | | 1030.94 | 59.3 | — | 5.4 | — |
| Jan 1 | 1039.29 | 5.64 | 115,020 | 21.00 | 20¼ | 120,750 | 6,276 | 22 | 114 | 1058.26 | 32.0 | 102 | 2.7 | |
| 5 | 1048.23 | 7.50 | 96,480 | 29.68 | 24 | 142,145 | 6,763 | 27 | 120 | 1070.02 | 20.00 | 67 | 1.9 | |

## Operating Data at Tiro General Mine

Table 16 gives the principal data that were summed up daily, covering the operation of the air-lifts at the Tiro General mine.

The first column gives the data covered by the set of observations. The second gives the water level in the shaft at 7 A.M. The third notes the distance that the water was lowered in the shaft; in a few instances the water arose above the point reached on the previous day, in which case the minus sign is used. The fourth column gives the total gallons pumped during the 24 hr. The fifth gives the cubic feet of air consumed (sea-level basis) per gallon of water raised; the volumetric efficiency of the compressors was assumed at 85 per cent., and the resulting volume was divided by 1.27 to reduce the volume from 7250-ft. altitude to sea-level basis. The seventh column gives the combined number of revolutions of the two compressors. The eighth gives the gallons of fuel oil consumed during the 24 hr., but this whole quantity was not used for pumping; hoists, shop machinery, etc. were driven by steam from the same boilers. The ninth column gives the average air pressure at the compressor, and the tenth gives the average steam pressure for the 24 hr. The eleventh column gives the average feet of lift, and the twelfth the average feet of submergence for the 24 hr. Average number of gallons for the 24 hr. is given in column 13; average percentage of submergence in column 14; and pumping efficiency in column 15.

Air-lift No. 1 was in operation alone between Feb. 10 and Feb. 28. Between Feb. 28 and May 9, the data refer to the Tiro General No. 1 and the San Fernando air-lifts combined. From May 19 to July 23, No. 2 air-lift was operating alone; from Aug. 19 to Oct. 6, No. 3 alone; and from Oct. 9 to Jan. 6, No. 4 alone.

## Data on Other Air-lifts

Table 17 gives data compiled from various sources regarding the performance of several other air-lifts, and Fig. 13 indicates their design. They range from 2½-in. to 13-in. diameter. In the light of the data obtained at the Tiro General mine, the following comments on these other air-lifts may be interesting.

*A, 2½-in.*—This lift is composed of three sizes of pipe, beginning with 2½ in. at the footpiece and increasing to 3½ in. at the discharge, the effect of which has been to secure a pumping efficiency of from 25 to 40 per cent., which is quite high considering that a small pipe was used for a lift of 600 ft.

*B, 3-in.*—An efficiency of 31 per cent. for this air-lift, where the lift is only 75 ft. and submergence is 53 ft., is rather low. Possibly an elbow was placed at the top of the pipe, thus preventing a free discharge.

*C, 3-in.*—The efficiency of 47.1 per cent. is high, and is accounted

TABLE 17.—*Data Pertaining to Air-lifts*

| | 2½" to 3½" No. 1 | 3" No 1 | 3" to 4" No. 2 | 3" to 4½" No. 3 | 3½" No 1 Ell | 3½" No 3 Ell | 3½" No 4 Ell |
|---|---|---|---|---|---|---|---|
| Submergence, $s$, feet | 478 | 53 | 337 | 478 | 248 | 205 | 215 |
| Lift, $l$, feet | 600 | 75 | 245 | 600 | 129 | 115 | 121 |
| Ratio $\frac{s}{l}$ | 80 % | 71 % | 138 % | 80 % | 194 % | 178 % | 178 % |
| Gallons water per minute | 75 | 119 | 120 | 117 to 160 | 82 5 | 120 | 135 |
| Total cubic feet free air | 166 to 295 | 93 | 102 | 354 to 565 | 82 | 74 4 | 85 |
| Cubic feet per gallon | 2 21 to 3 53 | 0 78 | 0 85 | 2 21 to 3.53 | 0 99 | 0 62 | 0 63 |
| Pumping efficiency, per cent. | 39 7 to 25 | 31 | 47 1 | 39.7 to 25 | 24 7 | 37 | 38 |
| Velocity water at footpiece | | 5 3 | 5 3 | 5 3 to 7 3 | 2 7 | 4 0 | 4 5 |
| Velocity water at headpiece | 5 0 to 2 5 | 5 3 | 3 1 | 2 4 to 3 2 | 2 7 | 4 0 | 4 5 |
| Velocity air at footpiece | | 12 3 | 3 1 | | 2.4 | 2 6 | 2 9 |
| Velocity air at headpiece | | 31 6 | 19 5 | | 20 6 | 30 0 | 33.7 |
| Pressure air at submergence | 207 | 23 | 146 | 207 | 108 | 89 | 93 |
| Ratio $\frac{s - \text{sub pressure}}{s}$ | 74 | 25 | 124 | 74 | | | |
| Submergence, per cent | 44 3 | 41 4 | 57 9 | 44 3 | 65 8 | 64 0 | 64.0 |
| Velocity mixture at footpiece | | 17 6 | 8 4 | | 5 1 | 6.6 | 7.4 |
| Velocity mixture at headpiece | | 36 9 | 22 6 | | | | |

| 4" No 1 No 1 Ell | 4" No 2 No 2 | 4½" to 6" No 1 | 4½" to 6" No 2 | 6" No. 1 No. 1 | 7¼" to 9½" No. 1 | 8" to 12" No. 1 Ell | 10" No. 1 | 10" to 12" No. 2 | 10" to 12" No. 3 | 10" No. 4 | 10" No. 5 | 10" No 6 Ell | 13" No 1 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 377 5 | 595 | 330 | 343 | 123 | 203 | 219 | 652 | 177 | 188 | 230 | 140 | 177 5 | 168 2 |
| 1135 5 | 895 | 243 | 193 | 133 | 250 | 161 | 73 9 | 200 | 431 | 360 | 50 | 47 5 | 120 6 |
| 31 | 66 | 136 | 171 | 92 | 81 | 136 | 88 | 86 | 44 | 64 | 280 | 380 | 140 |
| 32 | 45 2 | 350 | 290 | 207 | 700 | 441 | 974 | 1679 | 1233 | 750 | 1750 | 1496 | 1410 |
| 573 | 326 | 287 | 200 | 329 | 700 | 316 | 336 | 1809 | 3306 | 1200 | 756 | 329 | 513 |
| 17 92 | 7 21 | 0 82 | 0 69 | 1 59 | 1 00 | 0 72 | 0 345 | 1 08 | 2 68 | 1 60 | 4 32 | 0 22 | 0 364 |
| 11 2 | 17 5 | 48 8 | 46 4 | 19 5 | 50 | 43 | 50 | 38 9 | 34 | 44 | 28 | 45 | 72 3 |
| 0 8 | 1 2 | 7 0 | 5 8 | 2 4 | 5 5 | 2 95 | 4 0 | 6 9 | 5 0 | 3 1 | 7 1 | 6 1 | 3 4 |
| 0 8 | 1 2 | 4.0 | 3 3 | 2 4 | 3 2 | 1 3 | 4 0 | 4 8 | 3 5 | 3 1 | 7 1 | 6 1 | 3.4 |
| | | 4 0 | 2 7 | 8 9 | 5 8 | 2 0 | 3 6 | 8 9 | 15 3 | 4 7 | 4 4 | 1 6 | 1.56 |
| 110 2 | 62 7 | 24 6 | 17 1 | 28 3 | 23 5 | 6.7 | 10 3 | 46 | 84 | 36 7 | 23 2 | 10 1 | 9 3 |
| 164 | 258 | 143 | 149 | 53 | 88 | 95 | 27 | 77 | 82 | 100 | 61 | 77 | 73 |
| | | | | | | 115 | | | | | | | |
| 24 9 | 39 9 | 57 6 | 64 | 48 | 45 | 57 6 | 46 7 | 47 | 30 4 | 39 | 73 6 | 78 6 | 57 6 |
| | | 11 0 | 8 5 | 11 3 | 11 3 | 50 | 7 6 | 15 8 | 20 3 | 7 8 | 11 5 | 7.7 | 4 96 |
| | | 28 5 | | | 26 7 | 8 0 | 14 3 | | | | | | 12 7 |

for by the deep submergence and by the expansion of the pipe toward the point of discharge.

*D*, 3-*in.*—An efficiency of 25 to 40 per cent. is very good, considering the height to which the water is raised through a 3-in. pipe. The expansion of the pipe from 3 in. through 3½ in., 4 in., to 4½ in., has permitted a fair capacity to be obtained.


FIG. 13.

A—Lift, 600 ft., submergence, 478 ft; per cent. submergence, 44 3; cu. ft free air, 2.21–3.53; gal water, 75; per cent. efficiency, 39 7–25 0.

B—Lift, 75 ft.; subm., 53 ft; per cent. subm., 41.4; cu. ft free air, 0.78; gal. water, 119; per cent. eff., 31.

C—Lift, 245 ft.; subm., 337 ft.; per cent. subm, 57.9, cu. ft free air, 0.85; gal. water, 120; per cent. eff., 47.1.

D—Lift, 600 ft.; subm., 478 ft.; per cent. subm, 44 3, cu. ft. free air, 2 21–3 53; gal. water, 117–160; per cent eff., 39.7–25.0; pressure, 207.

E—Lift, 115 ft.; subm., 205 ft., per cent subm., 64; cu ft. free air, 0 62; gal. water, 120, per cent eff., 37.

F—Lift, 129 ft.; subm, 248 ft.; per cent. subm., 65.8; cu. ft. free air, 0.99; gal. water, 82.5; per cent. eff., 24.7.

G—Lift, 1135 5; subm., 377.5; per cent. subm., 24.9; cu. ft. free air, 17.92; gal. water, 32; per cent. eff., 11.2.

H—Lift, 895; subm., 595; per cent. subm., 39.9; cu. ft. free air, 7 21; gal. water, 45.2, per cent. eff, 17.5.

I—Lift, 243; subm., 330; per cent. subm., 57.6; cu. ft. free air, 0.82; gal water, 350; per cent. eff., 48.8; pressure, 143.

J—Lift, 193; subm., 343; per cent. subm., 64; cu. ft. free air, 0.69; gal. water, 290; per cent. eff, 46.4; pressure, 149.

K—Lift, 133; subm, 123; per cent. subm., 48; cu. ft. free air, 1.59; gal water, 207; per cent. eff., 19.5.

L—Lift, 250; subm., 203; per cent. subm., 45; cu. ft. free air, 1.00; gal water, 700; per cent. eff., 50.

M—Lift, 161; subm., 219; per cent. subm., 57.6; cu. ft. free air, 0.72; gal water, 441; per cent. eff., 43.

N—Lift, 73.9; subm., 65 2; per cent. subm, 46.7; cu. ft. free air, 0.345; gal. water, 974; per cent. eff., 50.

O—Lift, 200; subm., 177; per cent. subm., 47; cu. ft. free air, 1.08; gal. water, 1679; per cent. eff., 38.9; air pressure, 80.

P—Lift, 431; subm., 188; per cent. subm., 30.4; cu. ft. free air, 2.68; gal. water, 1233; per cent. eff., 34; air pressure, 90.

Q—Lift, 360; subm., 230; per cent. subm., 39; cu. ft. free air, 1.60; gal. water, 750; per cent. eff., 44.

R—Lift, subm., 140; per cent. subm., 73.6; cu. ft. free air, 0.423; gal. water, 1750; per cent. eff., 28.

S—Lift, 47.5; subm., 177.5; per cent. subm., 78.6; cu. ft. free air, 0.22; gal. water, 1496; per cent. eff., 45.

T—Lift, 120.6; subm., 168.2; per cent. subm., 57.6; cu. ft. free air, 0.364; gal. water, 1410; per cent. eff., 72.3.

*E*, 3½-*in.*—The efficiency of 24.7 per cent. is low, considering such deep submergence. An elbow at the top of the delivery pipe probably decreased considerably the capacity and efficiency. This arrangement of pipe should raise at least 150 gal. per min. to the height noted.

*F*, 3½-*in.*—Efficiency of 37 per cent. is fair for this installation, especially as an elbow was placed at the top of the delivery pipe.

*G*, 4-*in.*—The pumping efficiency is very low in this lift, and is explained by the fact that the diameter of the pipe was uniform throughout the great length of 1513 ft. This should have been increased probably to 8 in. by the time the top was reached, to secure greater capacity and efficiency.

*H*, 4-*in.*—The efficiency of 17.5 per cent. in this installation is low, for the same reason given for the preceding lift.

*I*, 4½-*in.*—Efficiency of this air-lift is high—48.8 per cent.—and is explained by the increased diameter of the pipe toward the top. The submergence is probably somewhat greater than is necessary.

*J*, 4½-*in.*—This installation, with an efficiency of 46.4 per cent., indicates a good design. Here again the submergence was probably greater than necessary.

*K*, 6-*in.*—The efficiency is low at 19.5 per cent., and is difficult to explain unless it was operated greatly below its efficient capacity, or unless there was an elbow at the top to which was connected a considerable length of pipe.

*L*, 7¼-*in.*—This installation consisted of a gradually tapering wooden pipe, and it will be seen that excellent results were secured, although the capacity could have been raised to about 1200 gal. per minute with efficient results. The 50 per cent. efficiency, however, is very good

*M*, 8-*in.*—The efficiency of this air-lift is fair, but the sudden increase in diameter from 8 in. to 12 in. is too great to secure best results. An air-lift of 8 in. uniform diameter, with this low lift, should secure an efficiency of over 60 per cent. It is doubtful whether any purpose is served in tapering a pipe where the lift is only 161 ft. This air-lift, with conditions as noted, should have a capacity of about 2000 gal. per minute.

*N*, 10-*in.*—The efficiency of 50 per cent. for this air-lift is fair, although it is no more than to be expected where the lift is only 74 feet.

*O*, 10-*in.*—The 38.9 per cent. efficiency of this lift, with conditions given, is only fair. A better design would be to insert a section of 11-in. pipe between the 10-in. and 12-in. sections. The efficient capacity under the conditions noted would be about 2100 gal. per minute.

*P*, 10-*in.*—The efficiency of 34 per cent. for this air-lift is fair, but better results would be obtained if arranged as suggested for the preceding case. The efficiency capacity of this arrangement should be nearly 1900 gal. per minute.

*Q*, 10-*in.*—Efficiency of 44 per cent. is good with this pipe of uniform

diameter, under the conditions noted. Apparently it was operating at about one-half its efficient capacity.

*R*, 10-*in.*—Efficiency of 28 per cent. is exceedingly low for this air-lift with its great submergence. It is possible that the lift was not operated at its efficient capacity, and possibly an elbow was connected at the top of the pipe, with long horizontal pipe attached.

*S*, 10-*in.*—Efficiency of 45 per cent. for this air-lift, at only 48 ft., is only fair. Better results could be secured by increasing the capacity 200 to 300 gal. per minute.

*T*, 13-*in.*—The efficiency of 72 per cent. for this air-lift is high, and in view of the fact that it was operating at only one-half its efficient capacity, it would appear as if there were some error in the data given.

Acknowledgment is due to Mr. W. M. Drury, general manager of the Mexican Mining Department of the American Smelting & Refining Co. for support in carrying on this work and for permission to publish this paper; to Mr. M. B. Stewart for the design of carrier arrangements; and to Mr. W. W. Shelby, Jr., for assistance in compiling these data and making tests on the air-lifts employed.

## DISCUSSION

Robert Peele,* New York, N. Y. (written discussion†).—The air-lift was originally employed almost exclusively for raising water from wells or other sources of water supply; in recent years, it has been increasingly used for mine service. It is especially well adapted to the unwatering of flooded mines, as the apparatus is readily lowered to follow the changing water level; but it is suitable also for permanent installations, instead of steam or electric-driven pumps. Perhaps the most important part of the paper is comprised in the descriptions of the operation of air-lifts Nos. 3 and 4 and the results obtained therefrom, followed by the discussion of the different factors of air-lift work, leading to certain tentative conclusions respecting percentage of submergence, capacity of delivery pipes of different sizes, air and consumption.

Heretofore it has been customary to make the submergence from, say, 50 to 60 per cent. of the lift; occasionally, even 70 per cent. or more. The impression has therefore sometimes been received that these high percentages of submergence are essential to the operation of an air-lift. It must be remembered that there are several interrelated factors. The percentage of submergence is influenced by the height of lift, the air pressure at the foot-piece, and the air consumption per gallon of water raised. Upon a proper adjustment of these variables depends the efficiency, which may be put roughly at 30 per cent.—ranging from say 20 to 38 per cent.

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† Received Feb. 24, 1920.

E. M. Ivens[2] states that "it is impracticable to pump at all under 20 per cent. submergence." He adds, p. 121, that "no set rules can be given as to what is proper submergence, and no formulas can be derived that will be even an approximate guide." These, I think, are unwarranted assertions.

The data obtained by Mr. Shaw from air-lifts Nos. 3 and 4 (Tables 7 and 8), and his conclusions therefrom, show submergences ranging down to even 1.8 per cent.; and, at 2 8 per cent., the pumping efficiency was 30.7 per cent., though at a high air consumption. These results were secured with lifts of over 1000 ft. Table 8 covers a range of lift from 812 to 1070 ft., with submergences of 21.2 per cent. and less. The fifth and sixth columns of this table show the corresponding reduction in volume of water raised (465 to 67 gal.), and the increase of air consumed (4.38 to 29.68 cu. ft. per gal.).

Another interesting point brought out by Mr. Shaw's tests is that the air-lift is applicable to greater heights of lift than any heretofore recorded, except that of example G, cited on p. 453. This installation gave the very low efficiency of 11.2 per cent., but it could undoubtedly have been improved by replacing the 4-in. delivery pipe of uniform diameter by a pipe increasing in diameter toward the top, as was done in the Charcas air-lifts Nos. 3 and 4.

Since his paper was printed, Mr. Shaw has been preparing to extend the delivery line of air-lift No. 4 to a length of 1355 ft. In a letter dated Jan. 18, 1920, he states that he plans to operate this at a constant lift of 1070 ft., with 285 ft. (21 per cent.) submergence. The diameter of the delivery pipe will increase from the foot-piece to the point of discharge, being composed of five sections of about 270 ft. each, of 6-in., 7-in., 8-in., 9-in., and 10-in. pipe. The use of this type of delivery pipe is unquestionably important in securing greater capacity and efficiency. Mr. Shaw plans to run his installation for a month or so, to obtain operating data, and then to install a Prescott station-pump on the 1050-ft. level. Thus he will be able to make a direct comparison between the performances of steam and air-lift pumps. Under the conditions named, the efficiency of the air-lift would probably exceed 30 per cent., and it is doubtful if an ordinary steam pump could do better.

Regarding air pressure, Mr. Shaw points out (in a letter dated Jan. 11) that the pressure at a foot-piece of good design must always closely approximate that which is necessary to overcome the submergence, and cannot rise appreciably higher. Any rise in air pressure would increase the volume of discharge. It follows that, in the absence of working data, pumping efficiencies can be computed on the basis of the expansion of the air between the foot-piece and the discharge point. Each cubic

[2] "Pumping by Compressed Air," 118.

foot of free air admitted to the footpiece, under the pressure necessary to overcome the submergence, gives up an amount of energy in foot-pounds, in expanding to the discharge pressure, equal to

$$W = P_2V_2 \text{ Nap. log. } \frac{P_1}{P_2}$$

which is the formula for isothermal compression of air, where $W$ is the theoretical power input at the footpiece. The loss of air pressure due to pipe friction is not large; it usually ranges from 2 to 5 per cent.

The formula, at the bottom of p. 441, for computing the volume of air required, exhibits all the factors entering into the problem. There are no existing data for determining accurately the losses due to friction of water and air and to slippage in the delivery pipe. These factors can be computed, approximately, from the principles of the flow of air and water in pipes, but their actual values will have to be obtained experimentally, as recorded data on air-lift operation are accumulated.

In carrying out the extensive series of tests made on the Charcas air-lifts, Mr. Shaw has had the unusual advantage of being able to vary the operating conditions within wide limits. He has proved that very small percentages of submergence are not only feasible, but are entirely consistent with the obtaining of reasonably good efficiencies. These results are particularly valuable in connection with mine pumping, because, in installing air-lifts in shafts, it is often difficult to provide the large submergences that have usually been assumed to be necessary.

# New Angles to the Apex Law

BY JOHN A. SHELTON,* M. S., BUTTE, MONT.

(Chicago Meeting, September, 1919)

ONE of the heaviest burdens uselessly cast by our mineral land laws upon the holder of the title conveyed by a patent from the United States is due to the provision excepting known veins from land patented as placer. In many instances the expense of defending such titles from the assaults of the owners of lode locations made upon the theory that the veins so located were known to exist at the date of the placer application for patent has exceeded many times the value of the land in question. Neither the lapse of time, however long after the issuance of the patent, nor repeated adjudications by the courts serve to quiet such titles or to prevent them from being questioned.

In the passage of laws providing for the acquisition of title to the mineral lands of the United States, Congress provided that the title to known veins should not pass from the United States by a patent based upon a placer location. As construed by the courts, the law (Sec. 2333 R.S.U.S.) provides that the applicant for a patent upon a placer claim may, if any vein of sufficient value to justify the expenditure of money in its development is known by him to exist within the boundaries thereof, include in his patent application an application for patent upon the vein as a lode claim; and failure to do so is to be deemed a conclusive declaration that the applicant makes no claim to the vein nor to an area of surface ground extending for 25 ft. (7.6 m.) on each side of its middle or center. As to such vein and such area of enclosing surface ground under the circumstances stated, the title remains in the United States notwithstanding the issuance of a patent that purports to convey the entire surface of a described area and everything beneath it, excepting only the segments of veins the apices of which lie outside of such area. Such known veins with such area of enclosing surface ground are, of course, at any time, so long as they remain unlocated, subject to location and appropriation in the same manner as other veins upon the public domain.

The history of the operation of this law in one district will serve to illustrate its effect generally throughout the mining regions of the United States. In the Butte district, the mineral that first attracted the atten-

* Attorney and Counsellor at Law.

tion of the miner was placer gold. The beginning of quartz mining was a few years later. A few years after the discovery of mineral practically the whole district, embracing several square miles of territory, was covered with mineral locations upon which patents were afterward issued. Roughly speaking, the low-lying land along Silver Bow Creek and its branches extending part way up the slope in the direction of the Butte

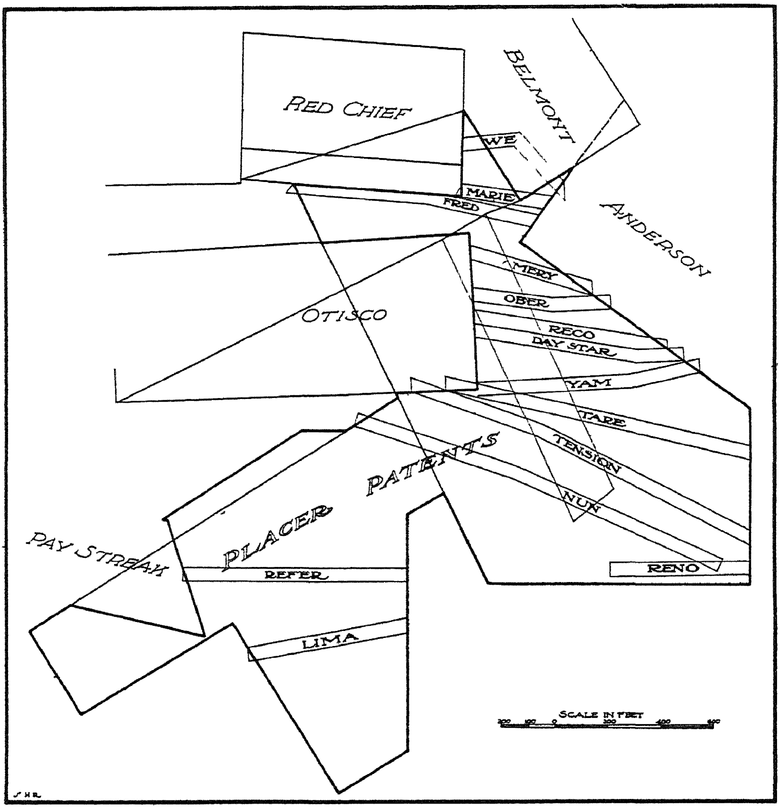


FIG. 1.

hill comprised the portion of the district within which the placer-mining operations were carried on. There the surface and subsurface for several feet generally consisted of loose earth and gravel or boulders in which was found the deposit of placer gold and beneath which was the solid formation or bed-rock. As to that land covering about one-half of the district, the patents issued were placer patents or patents based upon placer locations. While the apices of the veins showed in the surface of bed-rock generally throughout the whole district, their existence in ground covered by placer wash could not have been known until bed-rock was exposed by the removal of the placer wash in the placer-mining operation

or otherwise. When discovered in the early days they were not generally considered worthy of development. Frequently they did not contain mineral, on or near the surface, that could be profitably mined. Later development showed that many veins carrying little mineral at the surface were rich in copper at a depth, after which it became a practice in the district to sink on veins that had no showing of surface mineral, with the hope of discovering paying ore at a depth. As time went on, copper smelters were built and underground explorations were extended and it was proved that some of the veins in ground held under placer patents were of considerable value because of the facilities then existing for the treatment of ore of such character. This circumstance, coupled with the fact that the surface had in many instances been built upon and improved and had greatly increased in value, furnished the inducement to lode claimants to appropriate and obtain title to veins held under placer patents. The litigation resulting has been carried on more or less continuously ever since and now, after a lapse of 40 years from the issuance of the placer patents, still continues. Since the title to all veins known to exist at the date of the placer application for patent remained in the United States, no statute of limitations would bar such suits; and as a judgment in one suit will only bind those who are parties to it or their privies, a judgment in favor of the placer owner in one suit does not prevent a new lode location by another party, and a new suit concerning it. The judgment in such cases in every instance rests chiefly, if not solely, upon the determination of a question of fact, namely, whether or not the vein in question was a vein known to exist; that is, whether or not it was known to exist by the applicant for placer patent at the date of his application and was under the then prevalent conditions of sufficient value to justify exploitation and development. In every case it was necessary to call those as witnesses who at the date of placer application possessed such knowledge of the ground in question as qualified them to testify whether or not the vein was known to exist. Upon this point the testimony generally consisted of either affirmative or negative statements as to whether or not the vein outcropped on the natural surface of the ground or was exposed by the removal of the placer wash, so that its existence was known prior to the date of the placer application, and visible to the placer applicant, and whether if it was known, it was of such character that it would have justified exploitation and development under the conditions then existing.

As might naturally be expected, those who were called as witnesses in any particular case were in direct conflict in their statements upon such questions. It is scarcely necessary to remark that in many instances the witnesses testifying were mistaken as to the matters concerning which they testified and, such is the weakness of human nature, in other cases their testimony appeared to be knowingly false. As the litigation con-

cerning such question was carried on those who were called as witnesses became divided into hostile camps. Partisanship was not confined to those testifying as witnesses. The sentiment in the community concerning the merits of such controversies was divided. Under such circumstances it is not surprising that the outcome of such litigation was, in most cases, very uncertain. Occasionally different courts gave directly opposite decisions. To illustrate, a portion of one vein was involved in litigation in a case in the federal court and an adjacent portion of the same vein was involved in litigation in a state court. The two portions were covered by different lode locations, both included within the same placer patent. The testimony relied on in each case was substantially the same, but the decision in one case was in favor of the known existence of the vein at the date of the application for the placer patent and in the other case it was against such known existence.

The importance of the stability of titles is recognized by everyone. It is needless to point out that if the title is uncertain, development of the property is in consequence discouraged, and as a large proportion of the titles acquired under the mineral laws of the United States are held under placer patents (so-called), the effect of such uncertainty of title in the sections of the country where such titles are held is very considerable and must necessarily materially retard the development of those sections.

While what has been said about the importance of the stability of titles acquired and held under patents of the United States is everywhere recognized, it is also true, probably, that a large proportion of the people are not conscious of the extent to which the prosperity of the community may be unfavorably affected by such uncertainty of titles. We may gain a better conception of such effect by considering a similar, though extreme case. Much of the land of Mexico is more fertile than any land of the United States. The country is richer in its deposits of oil and minerals, and it has besides a great capacity for the production of rubber, for which there is a great and constantly increasing demand. In a word, Mexico is potentially a richer country than ours. But with all of such natural wealth, many of the Mexican people are in a pitiable state of abject poverty. There is no reason why they should be, except for their government or lack of government, and uncertainty of title is one of the principal consequences of their unstable government.

It is certainly true, though the confession is decidedly humiliating, that by reason of the provision which excepts veins known to exist from land patented as placer and because of the lack of a provision by means of which any such defect may be cured or an attack upon the placer title barred by the lapse of time, a portion of the territory of the United States has been, to a limited extent, Mexicanized. Congress has been asked to pass a law that would prevent an attempted lode location upon ground

patented as placer after the expiration of some fixed limit of time from the issuance of the placer patent; but for some reason no relief has been given.

The title held under a placer patent covering land within which there is a vein sufficient to support a lode location is uncertain, and cannot be rendered otherwise. The holder of such title may, however, relinquish it and procure in its stead a patent upon a lode claim, which is not subject to the same infirmities.

In procuring a lode patent to be issued covering land as to which a placer patent has already been issued two different courses have been followed. While the placer patent is outstanding, whatever land was conveyed by it has passed from the public domain and is not subject to disposal by the government, and an application for a second patent upon the same land will not be entertained by the land department of the government. There is nothing, however, to prevent the officers of the land department from accepting, on behalf of the United States, a deed from the owner of the placer title conveying back to the government whatever was conveyed by the placer patent, and the land having been in that manner restored to the public domain, is again subject to disposal by the government, and the necessary preliminary steps having been taken, a lode patent covering the same ground may be issued.

In practice the placer owner, in following such course, may first make a lode location or locations covering the ground, embracing in a single location the same extent of surface area as in the case of a lode location made upon unpatented land. Then, having in other respects complied with the law entitling him to a patent upon his lode claim or claims, application for such patent is filed and with it a deed is tendered conveying to the United States the title conveyed by the placer patent or such part of it as is covered by his lode location or locations. In the absence of any adverse claim, the land department will treat such location as though made upon unpatented land, and the deed being accepted, there is no longer an outstanding patent and no obstacle to the issuance of a second patent.

Of course all precautionary measures necessary to prevent a loss of rights are taken. The deed tendered becomes effective only when accepted by the government. Immediately upon its acceptance, the placer owner makes a second lode location covering the ground for the purpose of preventing an adverse location by some other person, upon the theory that until the acceptance of the deed the land was not subject to location. The placer owner, of course, keeps his second lode location alive by annual representation until the issuance of the lode patent.

If within the land covered by the placer patent there is a vein that was, in fact, known to exist at the date of the placer application, the titles to such vein and to an enclosing strip of surface ground 50 ft. in width is in the United States, and the placer owner, equally with any one else,

may locate it as a lode claim and, by performance of the acts required by law to enable him to procure patent, may procure the issuance of a lode patent on such location in the same manner that patent is procured upon a lode claim made outside of the limits of ground patented as placer, except that in connection with the application for patent, satisfactory proof must be furnished the land department, by affidavit or otherwise, of the known existence of the vein, and except, of course, that the extent of area that may be embraced in such location is limited to 25 ft. on each side of the middle or center of the vein. If the question of the known existence of the vein is involved in doubt, the placer owner may proceed upon the theory that it was known, provided that satisfactory evidence is procurable to the effect that it was known.

When such a lode location is made by a person other than the placer owner and judgment is recovered in his favor against the owner of the placer title that such vein was known to exist, and such lode locator then prosecutes proceedings for the purpose of procuring patent, such judgment is entitled to be given a conclusive effect in the land department, as elsewhere, upon the question of the known existence of the vein and may be relied upon as proof of such fact, dispensing with the necessity of affidavits.

If such judgment is rendered as the result of a compromise of a lawsuit or is rendered in a suit in which the placer owner does not resist the demands of the lode claimant, such facts in no way affect the question of the effect to be given such judgment. If the placer owner has a good title but allows it to be taken, the United States or any private individual is not thereby the loser and if his placer patent did not convey title to him and the title remained in the United States, the vein was subject to location and its locator upon compliance with the law is entitled to a patent to it.

Of course, the owner of the lode patent has extralateral rights, while the owner of the placer patent has not, and assuming that the vein was in fact not known to exist the lode locator who procures a lode patent on it on the theory that it was a known vein thereby may procure a greater right in the vein than was held by the placer owner. The segments of any veins lying outside the placer boundaries, the apices of which are within such boundaries, which veins were not known to exist at the date of the placer application for patent, of course do not belong to the placer owner but the title thereto is in the United States, and they are something therefore which the placer owner does not lose by allowing such judgment to be taken against him, and the lode locator does acquire them by his lode location and patent. The fact remains, however, that the recovery of the judgment affects the rights of no one except the lode locator and the placer owner, for such segments of veins are not, as something separate and distinct from the veins of which they are a part, sub-

ject to appropriation by private individuals or to disposal by the United States under existing laws.

The two methods here outlined have not in the past been frequently resorted to as a means of quieting uncertain placer titles, due, no doubt, to the risks apprehended as incident to the relinquishment of title. The proceedings are not recommended as being free from all possible danger, but such risks as there are, are of much less moment than the evils attending the uncertain title to land patented as placer and within which there are veins likely to give rise to a claim that they were in fact known at the date of the placer application.

# Mining Methods of Alaska Gastineau Mining Co.

BY G T JACKSON,* JUNEAU, ALASKA

(Chicago Meeting, September, 1919)

THE Alaska Gastineau Mining Co.'s mine is located at Perseverance, about 4 mi. east of Juneau, Alaska. Its property consists of a group of claims, the lode system traversing these claims for a distance of 11,000 ft. (3352 m.). It is shown in Fig. 1.

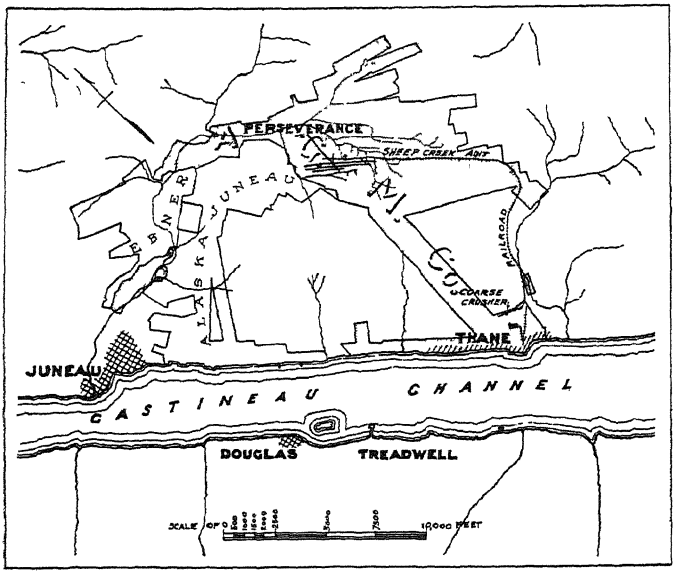


FIG. 1.—ALASKA GASTINEAU MINING CO.'S PROPERTY NEAR JUNEAU.

The ore deposits occur in what is known as the Juneau gold belt. This gold belt can be traced for about 50 mi. northwest of Juneau and 40 mi. southeast. It consists of a single band, several hundred feet wide, in which stringers and veins of quartz, carrying gold, occur in a slate formation near its contact with some altered volcanic rock known as greenstone.

The Perseverance orebody consists of stringer lodes, having a strike northwest and southeast, dipping about 60° to the northeast. These

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orebodies occupy a zone of shearing in the slate near its contact with the greenstone foot wall. This shearing and fracturing is very marked in the slate, and to a less extent in the schist, and metagabbro dykes, which intrude the slate. The quartz veins and stringers are more numerous near the foot wall, and become smaller and more scattered in the hard black slate hanging wall.

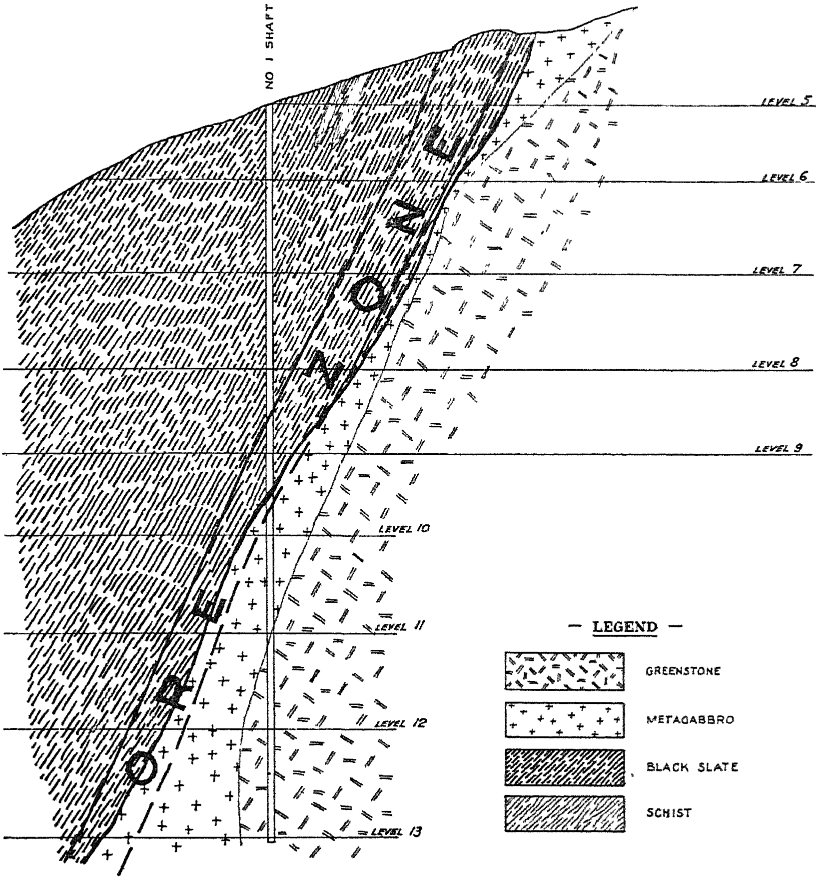


FIG. 2.—SECTION THROUGH PERSEVERANCE OREBODY AT NO. 1 SHAFT.

Fig. 2 is a cross-section through the No. 1 shaft, showing the rock formation; it shows that the orebody lies practically on the contact of the slate with the greenstone. Fig. 3 is a section through the orebody 1000 ft. (304 m.) east of the No. 1 shaft. It shows that in the lower levels the ore occurs in metagabbro and slate. From level 10 to level 6 it is wholly in the slate, and from level 6 to within 150 ft. of the surface it is

in schist, and outcrops in a metagabbro dyke. Other mineralized zones, intersected with numerous quartz stringers, are found in the slate and metagabbro dykes in the hanging wall of the main orebody. The only orebody developed so far is the one that lies on or near the main contact.

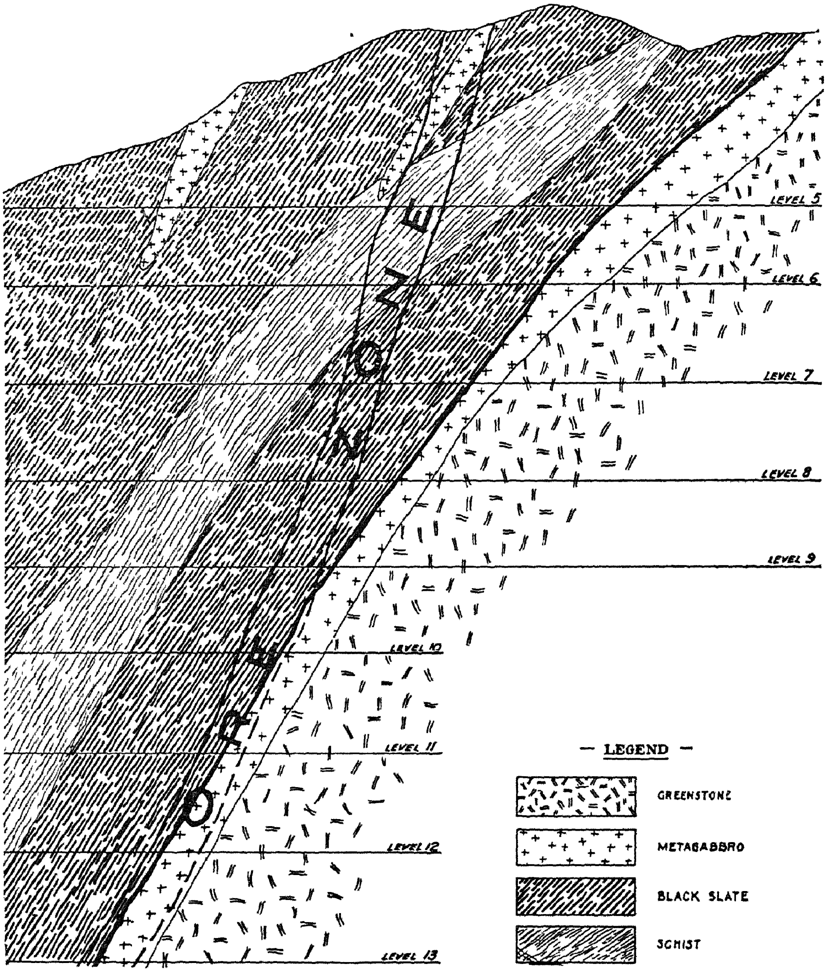


FIG. 3.—SECTION THROUGH PERSEVERANCE OREBODY 1000 FT. EAST OF NO. 1 SHAFT.

The gold occurs usually free, is fairly coarse, and is generally found near the contact of the sulfides with the quartz. The predominating minerals are zinc-blende, galena, arsenopyrite, pyrite, and pyrrhotite, named in order of their importance as gold carriers. There is, on an average, as much silver by weight as gold in the ore.

## History

Gold was discovered in Silver Bow Basin near Perseverance in 1880 and placer mining was carried on for several years, in the gulches and basins, with more or less success. Several attempts were made to work the small rich quartz stringers, transporting the ore by aerial tramways to small stamp mills, but owing to the shortness of the summer season and the small tonnage treated they were not successful.

About the year 1900, the late Mr. Joseph Gilbert bought the Perseverance claims and, with the assistance of the late Col. W. J. Sutherland, the Alaska Perseverance Mining Co. was formed. This company drove the Alexander cross-cut, the orebody being intersected at a point 2000 ft. (609 m.) from the tunnel portal. Drifting was done both east and west on the orebody, which work demonstrated an orebody of sufficient size and value to warrant the erection of a 100-stamp mill just below the portal of the tunnel. This mill commenced operations during 1907 and continued each summer, when water power was available, until 1912, when it was destroyed by fire. In 1910, the company was reorganized under the name of the Alaska Gastineau Mining Co., and has done business under that name ever since. In 1912, the Alaska Gold Mines Co was organized as a holding company to finance the Alaska Gastineau Mining Co., and development of the mine on a large scale commenced in July, 1912. At this time, the total underground development consisted of about 6200 ft. (1889 m.) of drifts, cross-cuts, and raises, also three large stopes, in which 812,444 tons of ore had been broken; 384,445 tons of this ore had been milled in the Perseverance 100-stamp mill during the years 1907–12. The three stopes, together with the pillars between them, covered practically all the extent of the orebody then developed on the Alexander level. They were opened up and worked without any attempt at selective mining or sorting and had a total length of 1150 ft. (350 m.), an average width of 90 ft. (27 m.), and were worked up to an average height of about 100 ft. (30 m.) The distance of the Alexander level to the outcrop, measured on the dip of the vein, would average 1200 feet.

## General Plan of Development

Fig. 4 is a longitudinal section of the mine, showing work done to July, 1912, and the proposed development program. This plan called for a main shaft for the handling of men, supplies, motors, and cars; opening up the mine by a series of levels, placed approximately 200 ft. (60 m.) apart; two main inclined oreways driven in the hard footwall rock for transporting the ore from the different levels to the Sheep Creek Adit; a main haulage tunnel about 2 mi. (3 km.) in length; and preparing a sufficient number of stopes to deliver 6000 tons of ore per day.

The methods adopted for driving of main drifts, cross-cuts, oreways, shaft sinking and raising do not call for any special mention, this work being usually done by contractors, the company providing all tools, machine drills, and compressed air, and the contractors providing the labor for drilling and mucking and paying for all supplies used.

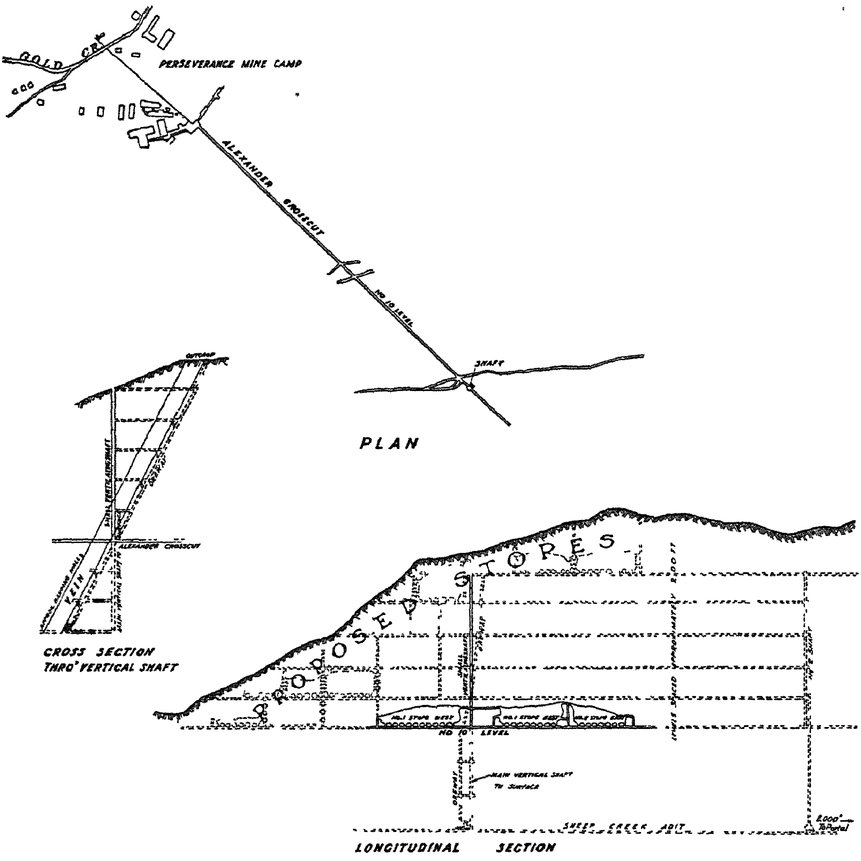


FIG. 4.—PERSEVERANCE MINE DEVELOPMENT PROGRAM, 1912. SOLID LINES = WORK DONE TO JULY, 1912. DOTTED LINES = PROPOSED WORK.

## MAIN DRIFTS

The main tramming drifts are driven in the foot wall parallel to the orebody, from 30 to 50 ft. (9 to 15 m.) distant. They are 7 ft. high, 8 ft. wide in the back and 9 ft. wide at the bottom, with a 24 by 24-in. ditch on the foot-wall side. No timber is used in the drifts, except where they cross the main contact fault. The following table gives in detail average costs per foot for driving in various formations:

| | Slate | Gabbro | Schist |
|---|---|---|---|
| Labor | $4 87 | $4 76 | $4 86 |
| Power | 0 29 | 0 36 | 0 29 |
| Superintendence and general expense | 2 05 | 2 39 | 2 20 |
| Store expense | 0 08 | 0 08 | 0 27 |
| Tramming labor | 2.64 | 2 86 | 3 35 |
| Tramming | 0.03 | 0 13 | |
| Machine-drill expense | 0.89 | 1 10 | 1 08 |
| Explosives | 2 50 | 3 26 | 2 82 |
| Candles and carbide | 0.07 | 0 07 | 0 09 |
| Timber | 0 12 | 0.10 | 0 54 |
| Other stores | 0.01 | 0 01 | 0 09 |
| Total | $13 55 | $15 12 | $15 59 |

As the drifts advance, the orebody is prospected by cross-cuts about every 400 ft., and is further prospected by diamond drills every 100 ft. between the cross-cuts.

## Stoping

The first stopes opened were those on the upper levels directly under the outcrop. With shrinkage stoping, less than 50 per cent. of the ore broken in the stopes is available for milling until the stopes are finished. It was, therefore, necessary to work as many machines as possible in the first stopes opened, so that they could be quickly finished and the broken ore remaining in them made available for the mill. After the first year's operation, about 60 per cent. of the ore delivered to the mill was drawn from finished stopes and 40 per cent. from working stopes and the preparing of new ones.

When the orebody has been thoroughly prospected, the stopes are laid off. In the slate formation, the stopes are cut out from 200 to 300 ft. (60 to 91 m.) in length, with a 40-ft. (12-m.) pillar between, while in the schist formation, stopes have been successfully worked up to 400 ft. in length, the width of the stopes varying from 40 to 120 ft.

*Charging Stoping Costs.*—In order to maintain an average cost for ore broken, stoping is divided into two accounts, preparing of stopes and stoping. The cost of preparing stopes is charged to a deferred account, which is written off at a fixed charge of 8 c. per ton against all ore reported broken in stopes each month; the actual cost of preparing stopes for the first 4 years of operation is slightly under 8 c. per ton. The following work is charged to preparing of stopes account: All manway raises and manway drifts, chute raises, building of chutes, bulldozing raises and drifts, and the connecting of the chute raises.

*Driving the Manway Raises.*—The first work undertaken in preparing stopes is driving the manway raises. These raises, measuring 5 ft. (1.5 m.) wide by 8 ft. (2.4 m.) long, are located in the pillars between two stopes, usually in the foot wall, and are driven at an angle of about 60°.

The rock through which these raises are driven is of a very firm character, no timber being required other than that used for staging. The method adopted is shown by Fig. 5; it is as follows. After the raise has been driven about 15 ft. (4.5 m.), a chute is built on one side, stulls spaced 5 ft. apart are placed up the center and lagged with 2 by 12 in. (5 by 30.5

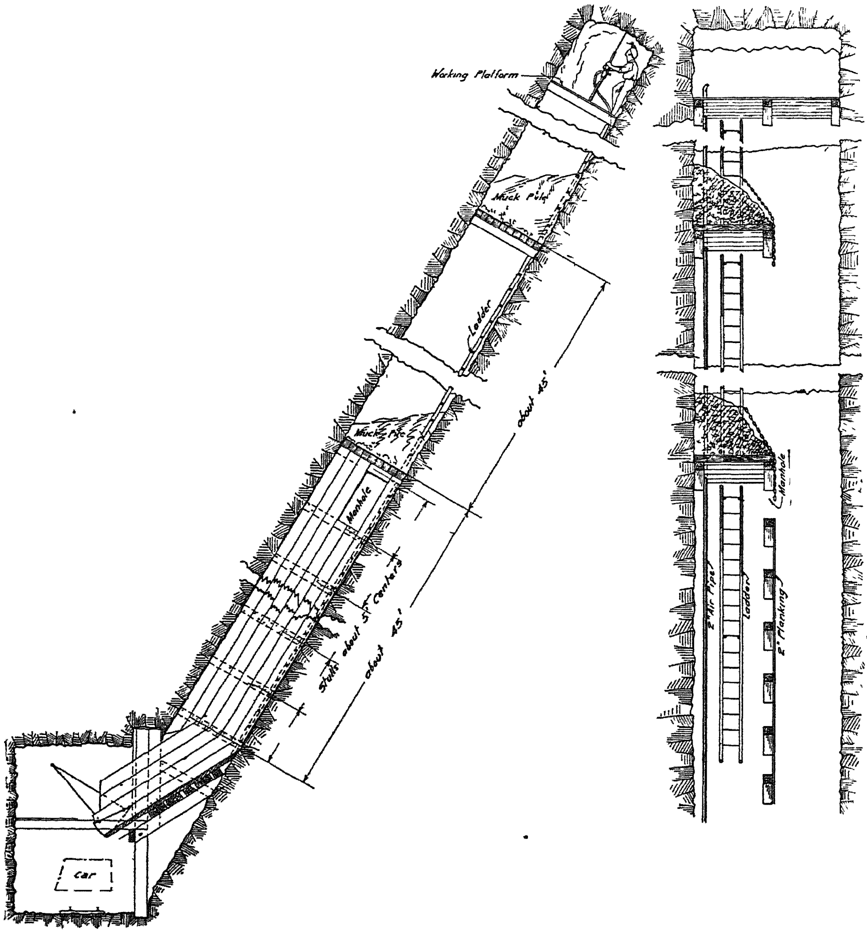


FIG. 5.—METHOD OF RAISING USED AT PERSEVERANCE MINE.

cm.) timber, thus forming a rock chute and a manway. When a height of about 45 ft. is reached, a bulkhead is placed over the manway, closing this off, a manhole being left in the side of the lagging. As the raise is advanced from this bulkhead, the ladders are placed on the foot wall and fastened by means of short drills placed in drill-holes. When the raise has reached a height of about 50 ft. (15 m.) from this bulkhead, another bulkhead is placed on the same side, covering half of the width of the

raise. The air pipe is then brought up to this point, and the raise continued. Other bulkheads are placed in a similar manner every 45 ft. (13 m.) until the raise reaches the desired height. During blasting, tools and spare steel are stowed away under the bulkheads. Stulls 4 by 6 in. (10 by 15 cm.) are generally used to hold the stage near the working face, intermediate drifts being driven from the raise as it advances, these to be used as passageways into the stopes.

When the raise reaches the level above, the chute and bulkheads are taken out and permanent ladders and a slide for a small skip, together with the air pipes, are installed. "Little Tugger" hoists are used in these manways for the handling of steel, explosives, etc. This method of raising has proved very successful, and raises up to 450 ft. (137 m.) in height have been worked in this manner, contractors packing up all of their steel and pulling up any timbers required with small pulley blocks. Detailed average costs of manways per foot are as follows:

| | Slate | Schist | Gabbro |
|---|---|---|---|
| Labor | $5.16 | $6.99 | $5 82 |
| Power | 0.21 | 0.27 | 0 28 |
| Superintendence and general expense | 1.54 | 2 16 | 2 00 |
| Store expense | 0 27 | 0.24 | 0 24 |
| Tramming labor | 0 24 | 1.58 | 0 79 |
| Tramming | .. | . | 0 04 |
| Machine-drill expense | 0 90 | 1 87 | 1 24 |
| Explosives | 1 53 | 2 42 | 2 08 |
| Candles and carbide | 0 03 | 0 08 | 0 05 |
| Timber | 0.64 | 0 43 | 0.49 |
| Other stores | 0.02 | 0.09 | 0 03 |
| Total | $10 54 | $16 13 | $13 06 |

*Chutes and Chute Raises.*—The next work of preparing stopes is to drive chute raises and build chutes, which are usually spaced 35 ft. apart. Figs. 6, 7, 8, 9, 10, and 11 show various types of chutes and bulldozing connections. Fig. 6 shows a type of chute with a bulldozing chamber right above. This type of chute was adopted during the years 1915 and 1916, but due to excessive bulldozing it was impossible to hold the caps in place. It was then decided to do the bulldozing in the chute raise, about halfway between the chute and stope. Small manway raises were put up from the main drift at each end of the stopes, and from these an intermediate drift about 20 ft. above the main drift was driven. This drift was connected to the chute raises by short cross-cuts, the bulldozing being done in the chute raises from these cross-cuts. Figs. 7, 8, and 9 show three styles of chutes with this type of bulldozing chamber. In these chutes, the head blocks were protected by ½-in. (12.7 mm.) boiler plate. This type of chute operated satisfactorily with the exception of the head block, which after a short time was broken and torn out by the

passage of the large rocks. Figs. 10 and 11 show the type of chute now adopted. In order to build this chute, it is necessary to have firm, hard rock, as no caps are used, the bulldozing being carried on about 10 ft. (3 m.) above the chute, this drift being driven from a small raise halfway between two chutes. The ventilation of the main tramming drifts is satisfactory, the powder smoke clearing away quickly even when bulldozing is almost continuous. However, in the bulldozing drifts the air

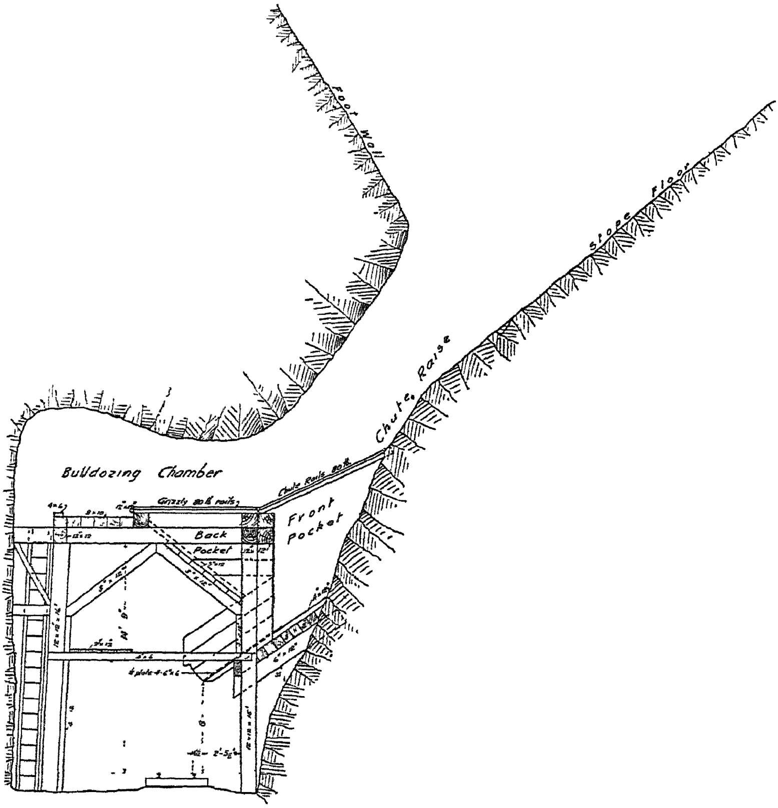


FIG. 6.

circulation is poor, and when a large amount of blasting is necessary a fan would have to be used to keep the drift properly ventilated. With the raise shown on Figs. 10 and 11, which connects with cross-cuts to two chute raises, it is possible to keep the atmosphere clear of powder smoke by turning on a little compressed air after blasting.

Before the chute raises are started, the engineers mark off the place, giving the exact height and the angle of raise. This raise is started only 4 by 4 ft. (1.2 by 1.2 m.), and carried this dimension for about three rounds. The drift is then widened out opposite to where the chute is to be built

and the back is taken up to a height of 14 ft. (4.2 m.), after which it is ready for the timberman to commence. The timberman first lines up the position of the chute, centering the raise and cutting hitches in the back of the drift to fasten the side posts the proper distance from the track. He then completes building the chute, installs the gates, loading platform, etc. When the rock is trimmed off in the mouth of the chute just flush with the sides, it gives a chute opening of 3 ft. 6 in. (1.05 m.) high by 5 ft. 4 in. (1.62 m.) wide. The chute raise is then continued until the ore-

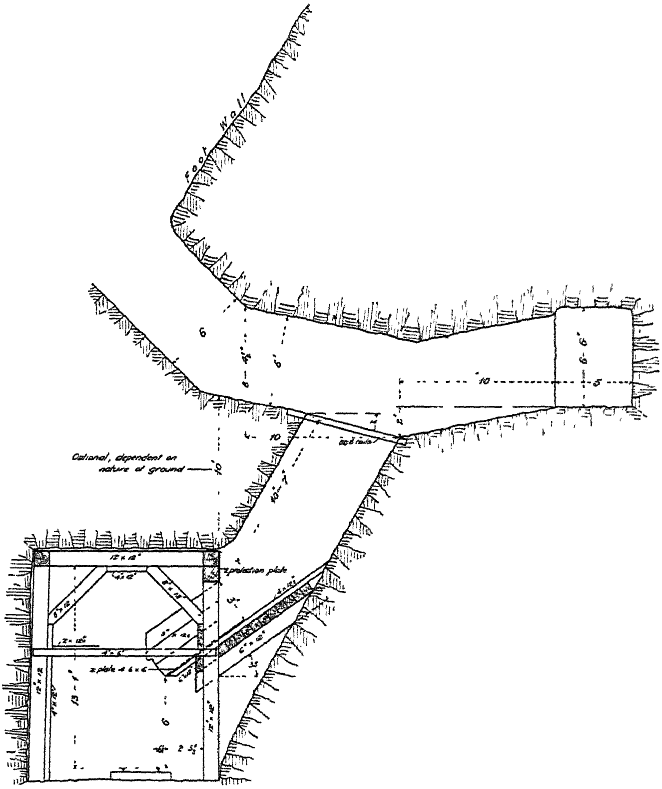


FIG. 7.

body is encountered. The bulldozing raises are then driven, and cross-cuts driven from the bulldozing raise to the chute raises. No grizzly bars are used at present, as most of the rocks can be caught before they get into the chute from the bulldozing chambers.

Fig. 12 shows a detailed drawing of a chute, underswung arc gate and check gate. The underswung arc gate was designed by the writer in 1912. It is used in all the chutes, both large and small, is very easy to construct, and can be operated without difficulty by one man when properly installed.

The costs of building these chutes, including the raise and widening out the drift, amounts to about \$615, of which \$400 is for rock and \$215 for material and labor of building the chute.

After the chute raises reach the orebody, they are connected by short raises driven up from each chute at an angle of 40°. A small raise is then driven from the end of each stope to connect with the nearest manway cross-cut, and the stope is now ready for undercutting.

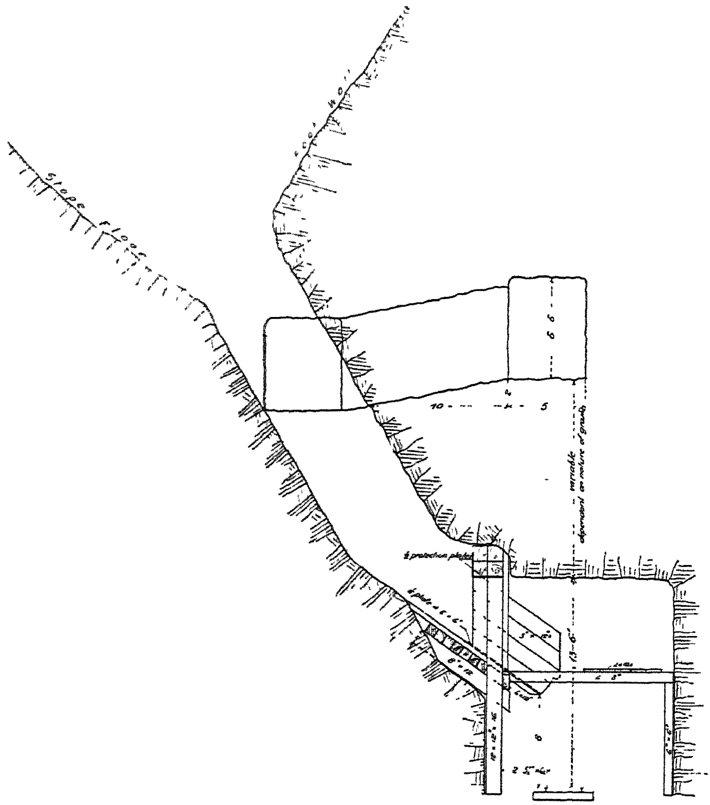

FIG 8.

## ACTUAL STOPING

A chute raise at each end of the stope is generally continued as a cross-cut raise until the hanging wall is reached. After this is sampled, it gives a much better idea of how wide the stope should be cut out. The values generally taper off as the hanging wall is reached so it is the usual practice only to undercut the stopes to a point about 20 ft. (6 m.) from where the values cease. The undercutting of the stopes is then commenced from these raise cross-cuts, this undercutting being carried for the

full length of the stope. The width depends on the values, as shown by the cross-cut raises; the height is about 7 ft. (2 m.) and at an angle of about 40°, so that broken ore will gravitate to the chutes.

There are two methods of stoping, one in which a small amount of the ore occurs in the foot-wall gabbro and the majority in the slate, and the other in which the ore occurs in a schist formation. The method adopted in the gabbro and slate stopes was introduced by the late John R. Mitchel when he was superintendent of the Perseverance mine. It consists of a combination of shrinkage stoping and caving. From 15 to 30 per cent. of the ore is broken by drilling and blasting, and the balance obtained through caving.

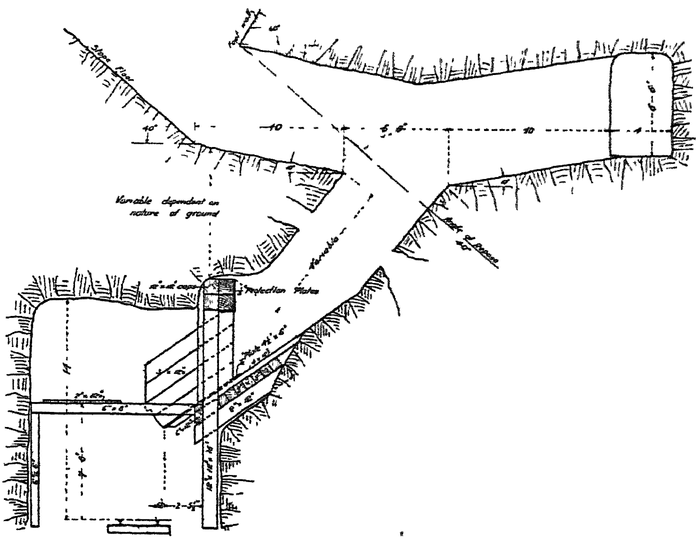

FIG. 9.

This method is described as follows: After the stope is undercut, machines are started from both ends of the stope, taking a cut along the foot wall 7 ft. (2.1 m.) high and 12 ft. (3.6 m.) wide, until the cuts meet in the center of the stope, after which the machines are brought back to the pillars and a cut of the same dimensions is taken across the pillars to the hanging wall. A small amount of caving will then commence, but it is generally necessary to take successive cuts of the same dimensions along the foot wall, and afterward across the pillars. Caving of the orebody at a right angle to the stratification takes place following these cuts, in quantities from a few tons to several thousand. The men working on the foot wall or along the pillars are perfectly safe, and continue to work up the stopes in this manner until the back of the stope approaches from 40 to 50 ft. of a worked-out stope above, at which time the caving becomes more

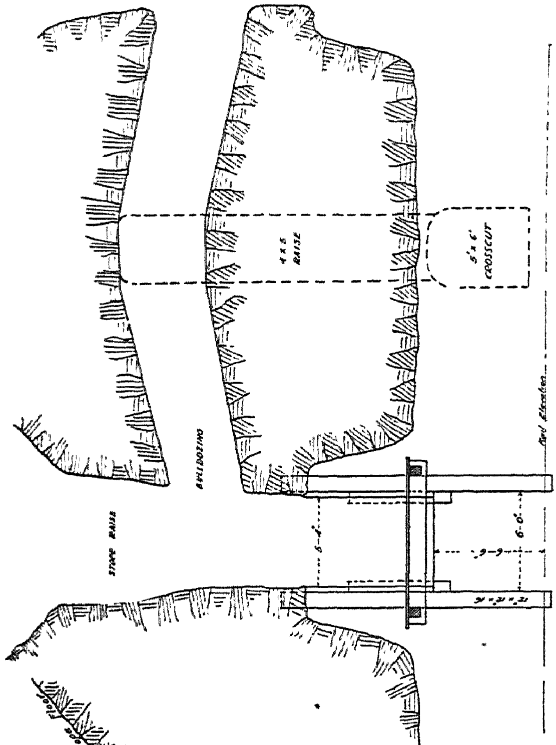
STOPE RAISE
BULLDOZING
4 x 5 RAISE
5' x 6' CROSSCUT

FIG. 11.

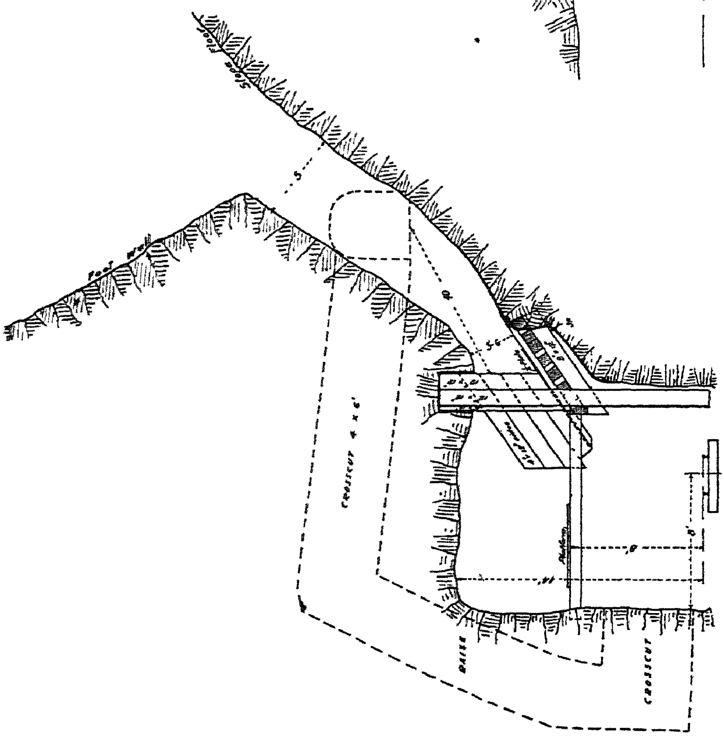
Foot Wall
Stope Floor
CROSSCUT 4 x 6'
RAISE
CROSSCUT
Platform

FIG. 10.

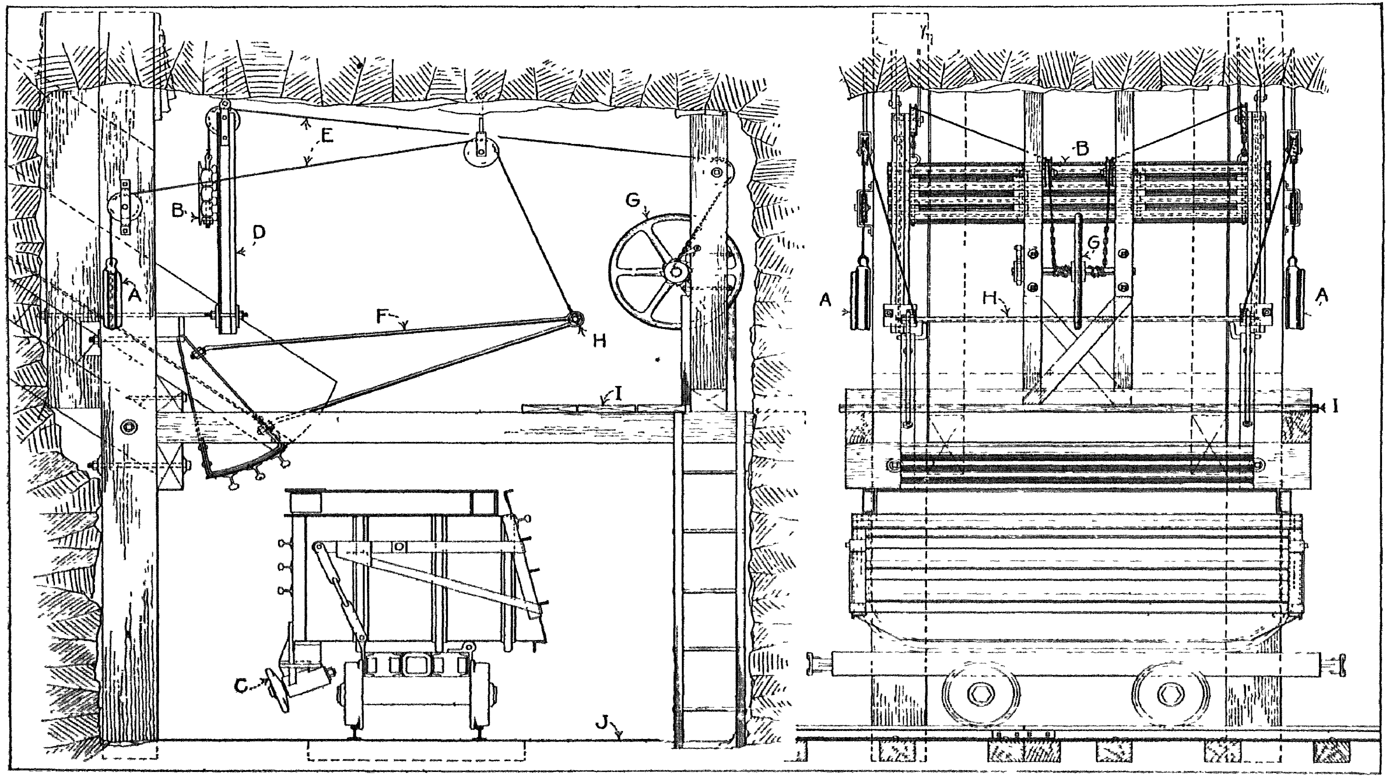


FIG. 12.—CHUTE AND GATES ON TRAMMING LEVEL.

A. Counterweight.
B. Check gate.
C. Dumping wheel.
D. 50-lb. rail check-gate guide
E. Wire ropes
F. Lever.
G. Hand wheel.
H. Operating handle
I. Platform for operators.
J. Shoveling plate for spill.

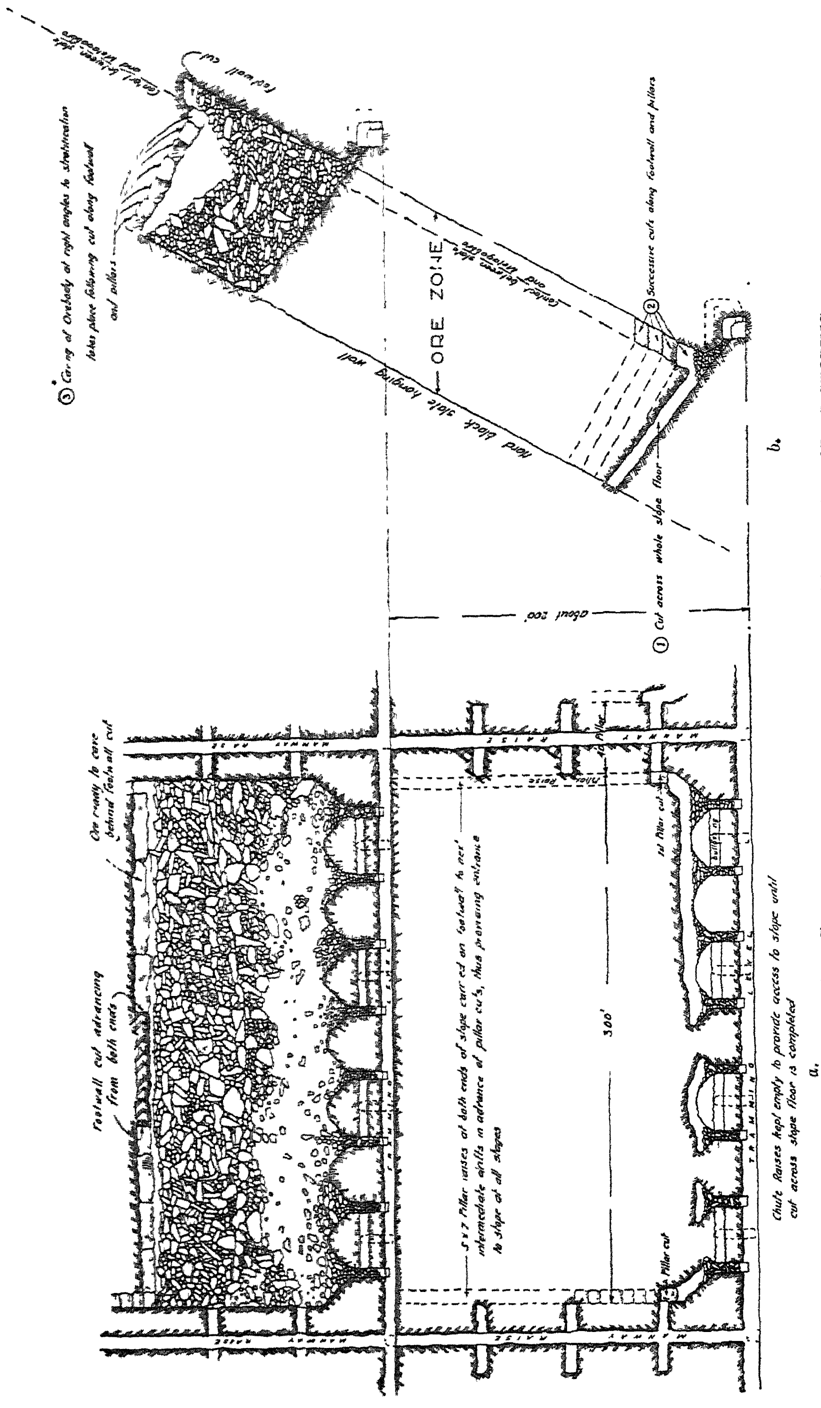


FIG. 13.—COMBINATION OF SHRINKAGE AND CAVING METHOD OF STOPING.

pronounced; the men are taken out of the stope, and the remainder of the ore caves through. As the ore is broken, the excess is drawn off through the chutes, just leaving sufficient so that the men can reach the back comfortably. Fig. 13 illustrates this method of stoping. On the bottom half of *a* are shown the chute raises at various stages of being connected; the upper half shows the stope during actual stoping and caving. Section *b* is a cross-section of this stope.

After actual stoping commences in this class of stope, 125 tons of ore are broken per machine shift. Two machines are operated on each end of the stopes, working two shifts a day. After the first month of actual stoping about 600 tons of ore a day are drawn from each stope by the tramming crews. Following are detailed costs of breaking ore in this class of stope, no account being taken for preparing or development charges:

| | Cost per Ton |
|---|---|
| Labor | $0.06025 |
| Explosives | 0 03312 |
| Miscellaneous supplies and expenses | 0 01885 |
| General underground expenses | 0 01686 |
| General mine expenses | 0 00659 |
| Total | $0.13567 |

The schist orebodies are worked by shrinkage stoping. The stopes are prepared and the undercutting done in the manner described for the slate stope, but as the ore in the schist stopes does not readily cave, or if it does, comes down in such large blocks that it cannot be handled, all the ore that occurs in schist formation must be drilled and broken with powder. The machines are started from each end of the stope on the foot wall and, wherever possible, the drilling is done at a right angle to the formation. A succession of cuts is taken across the stope from foot to hanging wall, excess ore being drawn off as required. A large number of machines can be worked in one of these stopes; and if the back is kept at a right angle to the dip of the foot wall, it is perfectly safe to work in any part of the stope. In this class of stope 45 tons of ore are broken per machine shift. Following are detailed costs of breaking of ore in schist stopes:

| | Cost per Ton |
|---|---|
| Labor | $0 12422 |
| Explosives | 0 06362 |
| Miscellaneous supplies and expenses | 0.04610 |
| General underground expenses | 0.03754 |
| General mine expenses | 0.01328 |
| Total | $0 28476 |

## Bulldozing

Two men are required to each tramming crew to break up the rocks as they accumulate in the chute raises, in order that the ore may be

drawn from the chutes. While the bulldozers work in conjunction with the tramming crews, moving from chute to chute as they become blocked, the cost of this work is charged to ore breaking.

Due to the methods of mining in the stopes and owing to the fact that a very large percentage of the ore is obtained by caving, the rock, as drawn from the chute, is of fine and medium size that can be easily loaded into cars, together with large rocks ranging from 3 ft. in diameter up to 50 tons or more in weight. When the ore is drawn from the chutes, the finer material is usually drawn first, the large rocks gradually working down until they arch over at a point where the chute raise enters the stope, thus preventing any ore from being drawn. The practice then is to tie several sticks of powder, with fuse and cap attached, to the end of a 1 by 3 in. (2.5 by 7.6 cm.) lath from 10 to 20 ft. (3 to 6 m.) in length, depending on how high the chute is hung above the bulldozing chamber. This charge is placed between two of the large rocks and set off; a large tonnage of ore may be thus loosened and the larger rocks brought down more into the chute mouth. If the rocks are very large, several charges of powder are set off in this manner, until all the loose small rock is brought down, and the large rocks completely block the chute raise. It is then safe to take a stoping machine up the raise and drill several holes into the large rocks, which are blasted in the usual manner.

In order to get the best results with the tramming crews, it is necessary that the bulldozer be a man skilled in the use of powder. Because the rocks jam and arch over, it is not safe to do much drilling until all the small rocks are first brought down. This means that a large quantity of powder is used for bulldozing operations; it usually amounts to nearly twice the powder used in the stopes. The following table shows detailed costs of bulldozing:

| | Cost per Ton |
|---|---|
| Labor | $0 01415 |
| Explosives | 0.05206 |
| Miscellaneous supplies and expenses | 0.00150 |
| General underground expenses | 0.00480 |
| General mine expenses | 0.00209 |
| Total | $0.07460 |

## Mine Tramming

This work consists of drawing the ore from the chutes. Transporting and dumping into ore ways, the upkeep costs of all motors, cars, charging stations and track are charged to tramming. On all the tramming levels 35-lb. (15-kg.) rails are used; the gage of the track is 26 in. (66 cm.). As most of the levels come out to the surface on the western end of the mine, the tunnels have a grade of 0.5 per cent. in that direction. This

means that the grade is in favor of the loaded cars on the eastern section of the mine, and against the load on the western section. Locomotives used for tramming are of the Baldwin-Westinghouse type, equipped with Edison storage batteries. Eight of these machines, weighing 6 tons each, are used on the tramming levels. Two 4-ton Jeffries locomotives, equipped with Edison storage batteries, are used for handling supplies and assisting in the tramming when necessary.

On each tramming level a motor-generator charging set is located, consisting of a 40-h.p. three-phase, 440-volt motor, direct connected to a 30-kw., 125-volt, direct-current generator. This set is large enough to charge three sets of batteries simultaneously. The locomotives operate an 8-hr. shift on a 4-hr. charge. As the tramming is on a two-shift basis, this operates very satisfactorily by having three motormen, the charging being done by the motormen between shifts. It requires one electrician, and sometimes a helper, to keep these machines in good service; that is, to see that the storage batteries have the right amount of water and that the electrical equipment is operating satisfactorily. These storage-battery locomotives, some of which have been in use since 1913, have given very satisfactory service.

Barring ordinary wear on mechanical parts and controllers, the locomotives are standing up well. The capacity and efficiency of the batteries have not yet been materially decreased by the wear on the elements, although considerable wear is evident on the plates of the older batteries, which necessitates more frequent washing of the cells to remove the sediment deposited in the bottom of the containers. This wearing of the element is natural, although it is materially increased by the erratic service and charging to which the batteries are often subjected. Considering the service and the conditions under which they are operated, the batteries are giving good results, and although the 4-yr. guarantee has expired on most of them, they will, if properly handled, give satisfactory service for several years to come. Detailed operating costs of storage-battery locomotives are shown in the following table:

| | Per Ton | Per Ton-mile |
|---|---|---|
| Maintenance cost | $0.00337 | $0.03019 |
| Maintenance charging station | 0 00024 | 0 00216 |
| Cost of power consumption | 0.00015 | 0.00132 |
| Total | $0.00376 | $0.03367 |
| Kw.-hr. consumed | 0 09948 | 0.89147 |

The cars used are of the Granby type, automatic side dump, and have a capacity of 4 tons. These cars have given very good service, and have been reinforced considerably to withstand the hard usage they get from the very large rocks which are drawn from the chutes, some of which weigh 3 tons.

A train crew, aside from the bulldozers, consists of a motorman, two chute punchers, and one mucker. The tonnage handled by a tramming crew on an 8-hr. shift varies considerably from day to day, due to the size of the rock drawn from the chutes and the number of interruptions caused by chutes hanging up. The record on an 8-hr. shift is 1962 tons. The average for the whole mine equals 489 tons per tramming shift.

A train consists of ten cars when operating with the grade in favor of the load, and eight when operating with the grade against the load. The

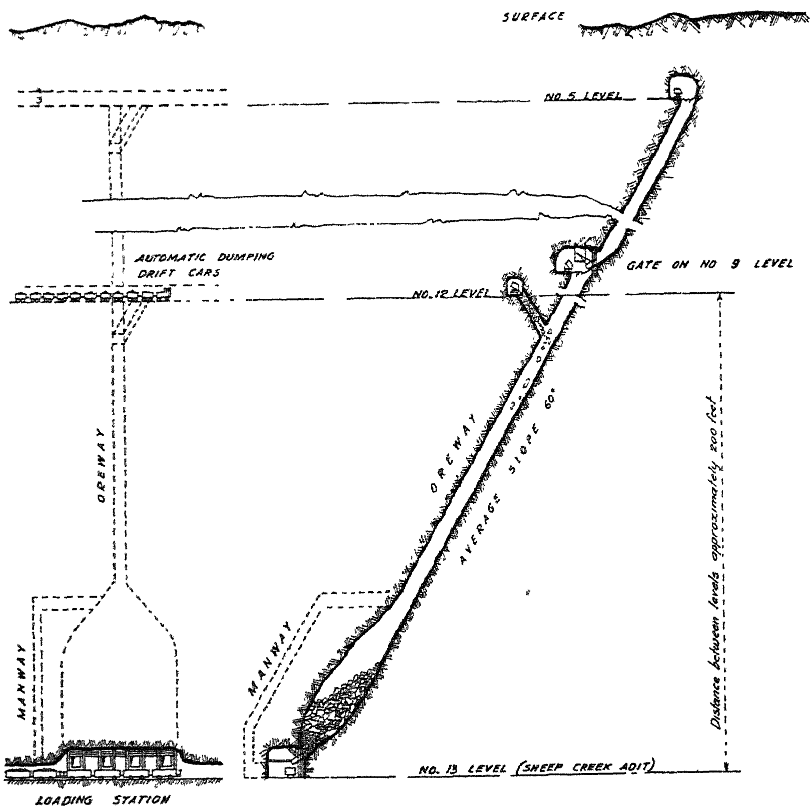


FIG. 14.—COURSE OF ORE FROM MINE LEVELS TO RAILROAD, PERSEVERANCE MINE.

blind end of the train is equipped with cow-bells, which are allowed to drag over the ties, giving an effective warning when a train is approaching. When two or more trains are operating to one ore way, a system of signal lights is used to prevent collisions, the motorman throwing in a red light when passing a given station, which prevents another motorman from proceeding to the dump until the red light is removed. Three men are required on the various levels to keep the cars oiled and in repair.

Chute maintenance costs $0.01012 per ton, four timbermen with helpers being required for this work. The boss chute puncher on each

tramming crew and the motorman keep a tally of the number of cars drawn from each stope and report to the shift boss at the end of each shift.

Fig. 14 shows the course of ore from mine levels through ore way to the loading station. Following are detailed costs of mine tramming:

| | Cost per Ton |
|---|---|
| Labor | $0 03812 |
| Power | 0 00020 |
| Maintenance equipment and tracks | 0 01461 |
| General underground expenses | 0 01052 |
| General mine expenses | 0 00316 |
| Total | $0.06661 |

## Ore Transportation

The ore is transported from the ore ways to the coarse crushing plant a distance of 3½ mi. (5.6 km.) on a single-track, 36-in. (91-cm.) gage, electric railway; 2 mi. of this distance is tunnel and the balance trestle and fill. In addition to the main line, 1¾ mi. of sidings and yard tracks are used. The ruling grade is 0.5 per cent. in favor of the load, and 50-lb. rails are used throughout. Rolling stock consists of 92 four-wheel all steel gondolas of 12-ton capacity, all cars being equipped with Simplex automatic couplers, and three 18-ton Baldwin-Westinghouse locomotives. Two 6-ton locomotives and 20 flat cars are used for the handling of freight.

The power to operate the trolley system is taken from the power lines at the mill substation, and supplied through two 300-kw., three-phase, 60-cycle, rotary converters at 550 volts direct current to the trolley system. The No. 4-0 grooved trolley wire is supported by A frames on the trestle, and hung from the roof underground. Power consumption has averaged 0.0824 kw.-hr. per ton-mile. The 18-ton locomotives are equipped with two 84-hp. motors, mounted on two pairs of 36-in. (91-cm.) steel-tired driving wheels, with a wheel base of 6 ft. 4 in. (1.9 m.) and are rated to develop a speed of 8 to 12 mi. (12 to 19 km.) per hr., with a drawbar pull of 5900 lb. (2676 kg.). Straight air brakes furnish the braking power.

The ore ways that form the connection between the mine and the railroad have a cross-section of 6 by 12 ft. (1.8 by 3.6 m.). At the lower end they are widened out to permit the location of four chutes, placed on car centers. Each chute is provided with two gates. The lower one, an arc gate 6 ft. 8 in. (2 m.) wide, is operated by hand and lets the ore into the car. The upper one, a check gate, is set with a fixed opening about 20 in. (50 cm.), and is lifted up by means of a hand wheel whenever large rocks block the passage of the ore, and falls back into place by gravity. The gates are operated from a platform hung from the roof above the cars.

During the first 4 yr., there passed through the four gates at ore way No. 1 about 6,275,000 tons. During this period the wearing plates inside the chutes have been changed only once, due to ordinary wear and tear, or after about 4,000,000 tons had passed over them. The original plates were 1/4-in. boiler plate and were replaced by 1/2-in. boiler plate. The check gates have been changed twice, due to ordinary wear and tear; several gates have been taken out and straightened, having been blasted. The underswung arc gates are still in use as originally installed, and are in very good condition.

The ore is loaded into the cars by a crew of four men, comprising three chutemen and one motorman. The cars are drawn very slowly under the chutes and the ore is dropped into them while they are in motion. Signal lights are provided for signaling the motorman from the loading platform. The actual loading of the train takes from 10 to 15 min. When loaded the train is taken by the loading crew a distance of 1/2 mi. to siding No. 2, where it is turned over to a running crew.

Two trains are used, and while the empty train is being taken by the loading crew from siding No. 2 to the ore ways and loaded, a loaded train is being taken by a running crew to the mill and dumped. The trip to siding No. 2 and return, including loading and cleaning up any spilled ore, takes about 40 min.; as this is slightly less than the time consumed by the running crew, a loaded train is always ready when the running crew arrives.

The running crew, composed of a motorman and two brakemen, covers the 3 mi. (4.8 km.) distance, between siding No. 2 and the mill in 15 min. In 20 min. more the ore is dumped, the cars being spotted in units of four in a tipple, which dumps the ore into the receiving bins by revolving on its longitudinal axis. Twenty minutes more are required to take the empty train back to siding No. 2, making a total of 55 min. for the round trip, including dumping. Thus a crew of seven men load, haul 3 1/2 mi. and dump from 3800 to 4300 tons of ore in 8 hours.

On a basis of a daily tonnage of 8000 tons, a repair and maintenance crew of ten men is used: one electrician and helper for upkeep of locomotives and trolley, light, telephone and signal lines; three repair men for upkeep of cars and chutes; and a track and tunnel crew of five men.

The following table gives detailed costs of ore transportation:

| | Cost per Ton |
|---|---|
| Labor | $0.01219 |
| Power | 0 00068 |
| Maintenance equipment and tracks | 0.01718 |
| General expenses | 0 00552 |
| Total | $0.03557 |

## Machinery and Equipment

The power necessary to operate the mine and railroad is generated in three hydroelectric plants, situated from 7 to 14 mi. (11 to 22 km.) distant from the mine. By building a dam across one creek, and tapping a large lake supplied by another creek, sufficient water is stored to supply all year power. The operating costs of these power plants are very small, and average for the past 4 yr. $0.00174 per kw.-hr. Forty-eight electric motors, of a total capacity of 2852 hp. are used to operate the machinery, the average daily power load being 1001.7 hp., which equals 0.224 hp. per ton. Five compressors of a total capacity of 9000 cu. ft. (252 cu. m.) per min. of free air are located just below the Alexander tunnel and are all electric driven. These furnish air for the machine drills and hoists at a pressure of 90 lb. per sq. in. (6.3 kg. per sq. cm.). The main hoist is operated by compressed air. It is equipped with Wuest herringbone gears, and is capable of hoisting an unbalanced load of 6 tons.

The main shaft has three compartments, in two of which double-decked cages are operated for the handling of men and supplies. The third is a small compartment for a ladder way, air pipe, and electric conduits. The large compartment, measuring 5 by 7 ft. (1.5 by 2.1 m.) is used for the hoisting or lowering of motors, cars and other large machinery.

The electricity used for operating the motor-generator sets and lighting is carried into the mine through an armored cable, at 2300 volts, to a substation near shaft No. 1, where the voltage is transformed down to 440 and 110 volts. Electric lights are installed on all the working levels, Mazda lamps being generally used; the average life per lamp is 762 hr. When one considers the adverse conditions under which the mine lamps operate, due to shocks received from concussions caused by blasting, the service rendered is considered satisfactory. A duplicate telephone system is installed, connecting the mine office and all the various levels throughout the mine. By means of this system, it is possible for the men working in the different sections of the mine to communicate with one another, or, by throwing a switch, to communicate with the mine office.

Blacksmith, machine, and carpenter shops are situated close to the portal of the Alexander tunnel. The blacksmith shop is equipped with three Leyner drill sharpeners and oil forges, and the machine and carpenter shops with suitable machinery for doing the ordinary repair work around a mine. Up to 1917, rock drills used for drifting and stoping were the 3.25-in. (8.25-cm.) piston type. These have been replaced by drills of the water Leyner type. Stoper machines are used for raising,

with Jackhamers for plugging and ditching. The following table shows comparative cost per machine shift for the different machines:

| | 3 25-in Piston | Water Leyner | Stoper |
|---|---|---|---|
| Feet drilled per shift | 31 99 | 38 12 | 33 00 |
| Repairs | $0 11277 | $0 80651 | $0 44976 |
| Steel sharpening | 1 36388 | 0 60533 | 0 61257 |
| Steel loss | 0.08461 | 0 54871 | 0 67604 |
| Labor | 6 44107 | 5 43605 | 3 93416 |
| Power | 1.75932 | 1 27332 | 1.08089 |
| Total | $9 76165 | $8 66992 | $6 75342 |
| Cost per foot of hole | $0 30512 | $0 22743 | $0 20465 |

## Cost Keeping

The costs of each drift, cross-cut, diamond-drill hole, and of the five or six operations incidental to preparing stopes, together with the actual breaking of ore in each stope, are kept separately and made up each month. Each individual piece of work is given a job number and the foremen and shift bosses are kept informed of their numbers, so that all labor and supplies used on each job can be properly charged.

The underground laborers are checked into the mine by the timekeeper. When assigned to their stations, their payroll numbers are reported by the shift bosses on mine time sheets. One of these sheets is made for each job, and on it appears, besides the labor, the number of holes and feet drilled, holes blasted, powder used, and cars of ore trammed. No powder or supplies are issued except upon requisition bearing job number and signed by the shift boss. The superintendent is furnished each day with an abstract of the shift bosses' reports, enabling him to see at a glance the number of men working on each job, number of holes drilled and blasted, powder used, and tons of ore trammed from each stope for the previous day.

The stopes are surveyed at the end of each month, the engineering department furnishing the superintendent and accounting department with a report showing the ore broken, trammed, and broken ore remaining in each stope; also a report showing development and prospecting work, contractors' footage, and work done in connection with preparing new stopes.

Ore drawn from stopes is charged into ore ways at the individual cost of each stope, which also includes an extinguishment charge of $0.08 per ton for preparing of stopes and $0.05 per ton for mine development.

Miscellaneous expense includes candles and carbide, compressed air, drill sharpening and replacements, and sundry supplies. General underground expense includes maintenance of mine tools and equipment,

foremen and shift bosses, engineering and hoisting expense, assay and sampling, mine lighting, and liability insurance. General mine expense includes mine office and general expenses, maintenance yards, trams, and building. General underground and mine expenses are distributed over preparing stopes, mine development, ore breaking, tramming, bulldozing, and chute maintenance on basis of direct labor.

SUMMARY OF COSTS, PER TON

| | |
|---|---|
| Ore breaking | $0 16169 |
| Bulldozing | 0 07460 |
| Mine tramming | 0 06661 |
| Ore ways and chutes | 0 01012 |
| Ore transportation | 0 03557 |
| Proportion, preparing stopes | 0 08000 |
| Proportion, mine development | 0 05000 |
| Cost of ore to mill | $0 47859 |

## CONCLUSION

The costs shown throughout this paper are an average for the years 1915 to 1918, inclusive, representing 6,523,873 tons of ore delivered to the mill. During the years 1915 and 1916, supplies were obtained at normal prices; 1917 was the first year of real war prices, and all supplies advanced 35 per cent. 1918 saw a further advance in material and supplies, this being equal to 70 per cent. above 1915. Labor advanced from an average wage of $3.80 per day during 1916 to $4.75 per day for 1918.

The writer hereby acknowledges the assistance received in the preparation of this paper from the following mine officials: D. J. Argall, mine superintendent; W. T. Tolch, chief engineer; Wm. Carlberg, railroad superintendent; E. M. Gastonguay, chief electrician; and H. E. Biggs, auditor.

## Milling Plant of the Alaska-Gastineau Mining Co.

BY E. V. DAVELER,* MET E , BUTTE, MONT

(New York Meeting, February, 1920)

THE milling plant of the Alaska-Gastineau Mining Co. is located at the town of Thane, Alaska, on Gastineau Channel, 4 mi. south of Juneau and directly across the channel from the Ready Bullion mine of the Treadwell mines on Douglas Island. Previous to the organization of the Alaska-Gastineau Mining Co., in 1912, the milling was done at the stamp mill of the Alaska Perseverance Co. located at the mine in Silver Bow Basin. But the contemplated increase in production made it necessary to locate the mill on Gastineau Channel, as not enough water was available at the mine during the winter months, nor was there room for taking care of the large tonnage of tailings that would ultimately be discarded from the mill.

### Character of Ore

The gold and silver values occur in quartz lenses associated with pyrrhotite, galena, arsenical pyrites, and zinc blende. These lenses occur in large bodies of slate, schist, and metagabbro, which also carry some valuable metallic contents. With the essential low cost of mining, it was necessary to mine large bodies of low-grade material containing the high-grade streaks. Of the minerals, pyrrhotite, the magnetic sulfide of iron, is predominant, and galena is the next sulfide present in quantity. The gold occurs free and coarse and associated with the iron sulfide in its finer state. The silver occurs with the lead or is alloyed with the gold. A chemical analysis of the ore shows

| | | | |
|---|---|---|---|
| Silver | 0.10 oz. | Magnesia | 0.4 per cent. |
| Silica | 59.7 per cent. | Sulfur | 1.2 per cent. |
| Iron | 5.2 per cent. | Lead | 0 1 per cent. |
| Alumina | 21.4 per cent. | Zinc | 0 2 per cent. |
| Lime | 5.0 per cent. | | |

### Experimental Work

Investigations at the old Perseverance stamp mill had developed the following points:

Roughing concentration following stamping gives a fair recovery.

* Mill Superintendent, Butte & Superior Mining Co.

Practically all gold values are liberated at 50-mesh. Amalgamation of the ore ground through 30-mesh yields 63 per cent. of the gold as bullion. An iron concentrate could be made carrying gold and silver values. Slimes were very low in value.

While the experimental plant was being built, laboratory work developed the fact that slimes were consistently low in value, which was of great importance in the later developments of the milling operations.

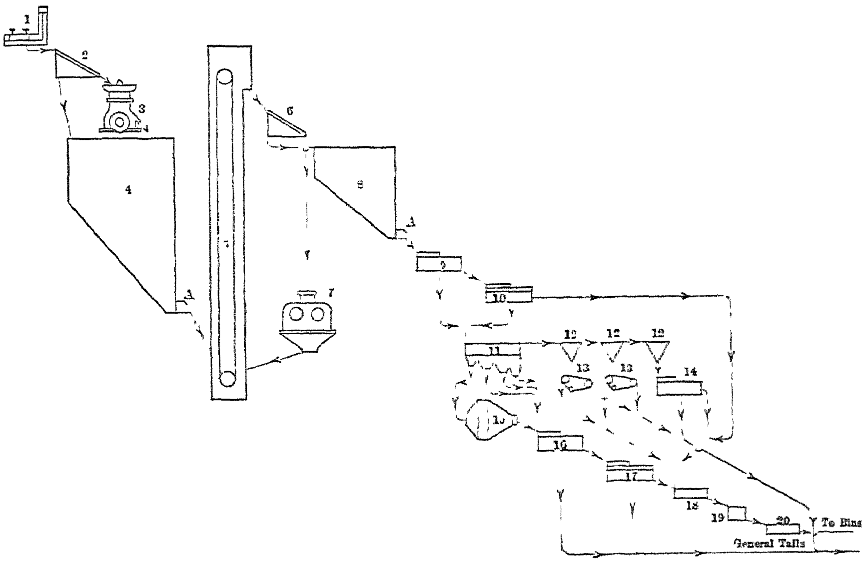


Fig. 1.—Flow sheet, experimental mill.

1. Fairbanks Morse track scales.
2. 1¼-in. grizzly.
3. No. 1 style "D" Gates crusher; capacity 5 to 9 tons per hour.
4. Coarse ore bin; capacity 100 tons.
5. 16-in. bucket elevator.
6. Two Colorado impact screens 3 by 4 ft.—660 shakes per min.; 12 mesh; 0.020 wire screen
7. 42 by 16-in. Garfield rolls; r.p.m. pulley 100.
8. Fine ore bin; capacity 150 tons.
9. 4-ft. Garfield roughing table; 260 r.p.m.; ⅞-in stroke.
10. No. 6 Wilfley table; 230 r p.m.; ⅞-in. stroke.
11. Four-compartment Richards-Janney classifier; revolution shaft 80.
12. Three cone tanks; capacity, 180 cu. ft.
13. Two 6-ft.- vanners; 190 shakes; 1⅞-in. throw.
14. No. 3 Deister slime table.
15. Hardinge mill, 30 in. by 8 ft.; 30 r.p.m.
16. 4-ft. Garfield table; 260 r.p.m. ⅞-in. stroke.
18. 4 by 8-ft. shaking plate; 1½-in. fall; 150 r.p.m.
19. Cobbe-Middleton grinding pan; 5-ft. diameter; 50-mesh screen.
20. Stationary plate; 4 by 6 ft.

A. Feeders, plunger type.

In Fig. 1 is shown a flow sheet of the experimental mill. This plant had a daily capacity of 75 tons, so all tests were run on this basis in order that results might be comparable with actual milling practice. The roll capacity, of course, was greater but the concentration department was run at normal tonnage. The concentrating department consisted of amalgamating plates, Garfield roughing tables, Wilfley and Deister tables, Janney classifiers, Frue vanners, and one 8 ft. by 30 in.

Hardinge mill. The machines were so placed that almost any variation desired in the flow could be made. All launders had ample grade and there was very little chance for concentration of values at any place in the mill. The headings and tailings were sampled automatically every 2 min. and hand samples were taken on the products of all machines coincident with the tonnage tests. These were then calculated back against the automatic samples for check.

The ore was delivered in sacks and crushed to pass through a 1-in. ring in the gyratory crusher, then ground to the roughing mesh required with rolls. A 50-ton bin fed the concentrating department so that concentration was independent of grinding.

As proper sampling of the ore was known to be very difficult, due to the coarse particles of gold free in the ore, the greatest precautions were taken. From laboratory tests before starting the experimental mill, it was known that grinding a large sample to 10-mesh was necessary to check results.

The heading sample was taken automatically after roll grinding with maximum size particles of 10-mesh, and on a mill run of 20 tons the heads sample amounted to 400 lb. This sample was rolled very carefully and then riffled through a Jones type sampler with a 1-in. opening. A cut of 50 lb. was obtained and the reject of 350 lb. was saved for other work. The 50-lb. sample was rerolled and riffled to 12.5 lb. which was ground to 100-mesh and after rolling was cut to about 4 lb. for an assay sample. The 37-lb. cut was saved for screen analysis work. The 350-lb. reject from the first cut was sacked and sent to the Arthur plant of the Utah Copper Co., where check runs were made in the laboratory unit. On the average the variation of all runs in the heading assay between the Utah and Alaska results was under 5 cents on the gold assay. In assaying the nail method was used and four three assay-ton charges made on each sample, combining the resultant button for parting and weighing. In the hand-sampling work, six men were used, two sampling each day, and all six men were alternating in the work to eliminate the personal equation as much as possible.

The experimental crushing plant was started with waste to observe the effect of roll crushing. The ore was low in moisture, running consistently under 2.5 per cent.; the ore screened very readily and under ¼-in. size the slate lost its long splintery character and became cubical. The worst problem that developed in the crushing tests was the dust, which was eradicated by sprays; they were also used in the large milling plant. In the concentrating tests, the simplest methods were used first, then these were elaborated as was found necessary.

The first tests were with grinding with the rolls to varying mesh, from 6 to 16; this was followed by primary roughing on Garfield tables and cleaning the rough concentrate on Wilfley tables. Both tables were mak-

ing final tailing. This method gave a recovery of 75 per cent., a tailing of $0.44, and a concentrate of $13 gold, showing a very satisfactory initial recovery, and work was followed upon tailings from this operation. Screen analysis showed that practically all the gold was liberated from the gangue with 50-mesh grinding. The tailings from the tables were then classified in the Janney classifier, the material on 50-mesh being reground in the Hardinge mill, and the fine sands sent direct to the secondary roughing tables.

The product from the Hardinge mill was treated separately on both the roughing and the finishing machines but with the development of the practice it was found that it could be combined with the fine sands from the classifier and treated on the roughing table, the rough concentrate being cleaned on finishing tables. Tailing made on the roughing table on the final test was $0.176; and on the finishing machine $0.32; as the tonnage of the finishing tailings was small both roughing and finishing-machine tailings were sent to waste.

The slime overflow from the Janney classifier, comprising about 17 to 20 per cent. of the original tonnage, was very low in values, an average assay on all tests being $0.178 gold. Tests were run on this to see if an economical recovery could be made but nothing satisfactory was worked out. It was then turned to the tail race as waste. This, of course, was one of the most satisfactory developments of the test work.

The above data show that a very economical treatment had been worked out giving high recovery at a low cost. Several check runs made with the flow sheet as adopted showed a tailing of $0.176 gold and an extraction of 89.30 per cent. of the gold values. The tonnage of concentrates produced in the final check runs showed that 89.30 per cent. of the gold values was concentrated into a product in the ratio of 60 to 1.

## Retreatment of Concentrates

Following this, work was started on the retreatment of the concentrates. An average analysis of the concentrates was as follows:

| | | | |
|---|---|---|---|
| Gold | $80.00 | Magnesia | 0.8 per cent. |
| Silver | 6 30 oz. | Copper | 0.1 per cent. |
| Lead | 5.2 per cent. | Zinc | 2.7 per cent. |
| Iron | 37.3 per cent. | Arsenic | 0.2 per cent. |
| Silica | 12 9 per cent. | Tin | 0.1 per cent. |
| Alumina | 10.7 per cent. | Sulfur | 27.0 per cent. |
| Lime | 2.2 per cent. | | |

The first step in the treatment was to separate the lead concentrate from the iron concentrate, which was easily done on Wilfley tables, using

a finger to regulate the cut. This lead concentrate carried all the free coarse gold, a material very hard to sample. Tests showed that this lead concentrate would be separated in a ratio of 857 to 1 as to the original ore milled. The further separation of the free gold from this was left to work out later and attention was paid to the iron concentrate remaining. This iron concentrate carried from $12 to $25 gold values, some silver, and some lead.

The first work was to grind to suitable mesh and amalgamate and reconcentrate. Grinding to 100-mesh liberated the values sufficiently or amalgamation. The iron concentrate, after plating, could then be reconcentrated and a small quantity of iron-lead concentrate of high enough grade to warrant shipping could be made. The final tests showed a tailing of $1.93 gold and $0.26 silver, which with the lead concentrate would mean a shipping concentrate in the ratio of 400 to 1 of the ore milled. At the laboratory in Utah, a tailing of $0.93 gold was made with fresh concentrates; as the concentrates used at Alaska had oxidized somewhat, it was felt that the same results could be obtained in mill practice.

During the winter of 1913, laboratory tests were run at Alaska on cyaniding the iron concentrates to determine whether higher recovery could not be made. The presence of pyrrhotite made this rather difficult but the method was finally worked out on an economic basis, giving an average recovery of 98.50 per cent. of the gold values, with a 4-lb. cyanide consumption. However, the higher cost of the process was a disadvantage so it was decided to use concentration and amalgamation on the iron concentrates.

## Mill Construction

During the experimental work, which lasted until March, 1914, active construction work had been started in the large mill. This work was carried on throughout the entire year with very few delays except a few days from very severe winter weather. By February, 1915, with the exception of the retreatment plant, enough of the mill was completed to insure starting operations. Three operating shifts were organized and on duty at the plant for 10 days before the plant started, running machinery available, so that when ore was started through there was practically no delay and the usual troubles of starting a new plant were missing. Actual mill operations were started on Mar. 1, 1915. During the construction period, carpenter bosses were paid $7 per day, first-class carpenters 60 c. an hr., second-class carpenters 50 c. an hr. and labor 35 c. an hr. At the millsite and experimental camp at Thane, Mr. C. E. Bruff was the engineer in charge of construction and the writer in charge of experimental work and milling operations.

## Flow Sheet of the Mill

The ore from the different levels is dumped into one of the two ore-ways by 4-ton cars of the Granby type. These two ore-ways are equipped with four underswung arc gates for loading into the 10-ton cars that carry the ore to the mill. At the coarse-crusher bins, the cars are spotted in groups of four and dumped by a revolving tipple, which is operated by a 50-hp. General Electric alternating-current motor. The starting and stopping of the tipple with the motor running is accomplished by a friction clutch with a bevel pinion from the operation platform. The tipple is also equipped with a band brake. The motor and operating mechanism are designed to handle two tipples but to date only one has been installed.

The ore dumped from the cars rolls down a 45° slope to a line of 8-in. steel I beams spaced 10 in. apart and equipped with manganese shoes to take the wear. After over 6,000,000 tons had been dumped on these grizzlies, a number of the I beams had to be replaced, because of bending, but the manganese shoes are still in service. To facilitate the sliding of the wet ore, boiler plate was used; later this was replaced by worn-out roll shells 1 in. thick, which were straightened under the steam hammer at a red heat. This makes a very cheap liner and has a very long life. The oversize from the grizzlies falls into steel bins ahead of the jaw crusher, while the undersize falls into another bin.

The undersize from the grizzlies is fed by four 42 in. (106 cm.) wide apron feeders to a stationary crimped-wire screen 3 ft. (0.9 m.) wide and 14 ft. (4 m.) long set at an angle of 45° with 2½-in. (6.35 cm.) openings. The undersize from this screen drops through raises cut in the rock to an underground storage bin, which has a capacity of 8000 tons. This bin was cut out of the rock during the construction period, the broken rock being crushed in a temporary plant for use as rock and sand in all the concrete used in the mill. It has thus served two useful purposes. At the bottom of the bin, there is a reinforced-concrete arch over the tunnel leading to the fine-crushing department.

The oversize from the grizzlies is fed, by air-operated arc gates, to two 36 by 42-in. (91 by 106-cm.) Buchanan jaw crushers grinding to 5 in. (12.7 cm.) and discharging on 2½-in. (6.35-cm.) stationary double-crimped screens; the undersize of these screens drops into the underground storage bin. The oversize from all of the 2½-in. screens drops into four No. 8 K Gates gyratory crushers, which reduces the feed to pass through a 2½-in. opening. The crushers are of the gun-lock type, the mantles and concaves being of manganese steel. The lower concaves last about 8 mo. on a 16-hr. daily operating basis. Changing the concaves and re-zincing an entire set in one crusher takes 8 hr. under normal conditions.

All the crushers are belt driven from a central line shaft, which is divided into two sections so that each side of the plant is independent.

The line shaft is of $5\frac{15}{16}$-in. cold-rolled shafting, each unit being driven by a 200-hp. General Electric motor. Clutch pulleys transmit the power

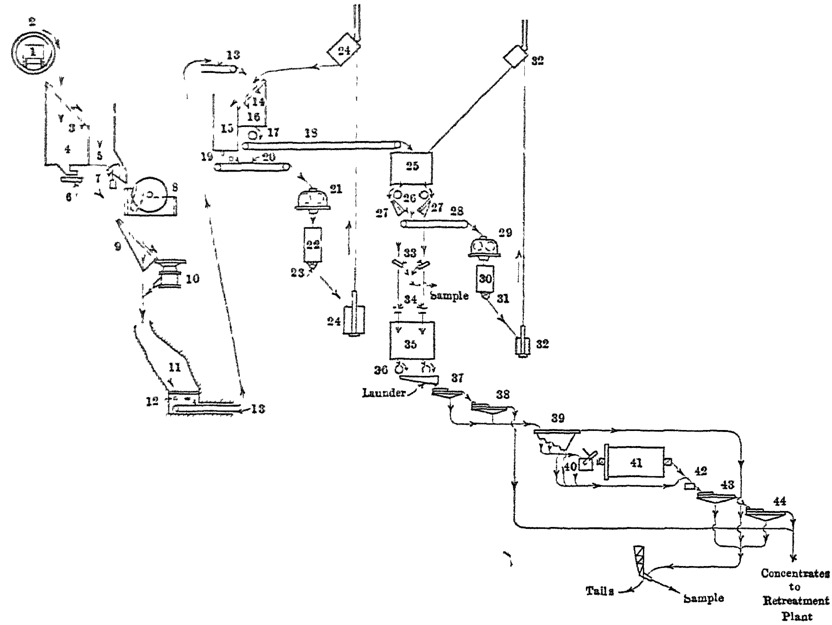


FIG. 2.—FLOW SHEET FOR C.C. PLANT, ROLL PLANT AND ONE 2500-TON SECTION OF CONCENTRATING PLANT.

1. 10-ton ore car
2. Revolving tipple, four cars.
3. Grizzly; 8 in. clear openings.
4. Bin for undersize from grizzly.
5. Bin for oversize from grizzly.
6. Four 48-in. caterpillar feeders.
7. Four air-operated gates.
8. Two 36 by 42-in. jaw crushers.
9. Grizzly, $2\frac{1}{2}$-in. clear openings.
10. Four No. 8 gyratory crushers.
11. Underground storage bin.
12. 8-ft. rotary feeders.
13. One 42-in. belt conveyor.
14. Nine 48 by 36-in. impact screens, 1 in. openings.
15. Bin for oversize from screens.
16. Bin for undersize from screens.
17. Rotary feeders.
18. One 42-in. belt conveyor with tripper.
19. Six rotary feeders.
20. Two 36-in belt conveyors.
21. Two 72 by 20-in. rolls at 80 r.p.m.
22. Hopper.
23. Automatic gate.
24. Four automatic skips.
25. Bin.
26. Rotary feeders.
27. Sixty 48 by 36-in. impact screens, 9-mesh.
28. Ten 24-in. belt conveyors.
29. Ten 54 by 20-in. rolls at 100 r.p.m.
30. Hopper.
31. Automatic gate.
32. Ten automatic skips.
33. Automatic head samplers.
34. Two 36-in. distributing conveyors.
35. Storage bin for fine ore.
36. Rotary feeders.
37. Ten double-decked Garfield roughing tables.
38. Ten No. 5 Wilfley tables.
39. Ten 4-spigot Janney classifiers.
40. Five dewatering wheels.
41. Five tube mills, 7 by 10 ft.
42. Two-way distributors.
43. Ten double-decked Garfield roughing tables.
44. Ten No. 5 Wilfley tables.

to the crushers through a belt drive. All clutches are equipped with operating mechanism. Electric switches are provided so that the operator can cut off power from either the gyratory or the jaw-crusher

floor. This plant has a capacity of 3500 tons in 8 hr. crushing to pass through a 2½-in. ring.

The operating crew consists of a crusher man, feeder man, and oiler, per shift; one man and a helper on the day shift do all the routine repair work. Changing of concaves or mantles is done by the rigger crew when needed.

The product from the coarse-crushing plant is fed from the bottom of the underground storage pocket by eight rotary feeders to a tunnel conveyor belt, which carries it to the fine-crushing section. The rotary feeders are pulleys, 36 in. (91 cm.) in diameter with a 36-in. face, which slowly revolve and pull the ore from chutes attached to the bin bottom. The rotary feeder has a capacity of about 3000 tons daily with the standard gate opening; different feeders are operated during each 24-hr. period to maintain an even character of ore, as there is some segregation in size in the ore pocket. These feeders are very economical to install and to maintain. After about 4 years' service some are still in place and the cost of repairs to them has been practically negligible. Wherever possible, these feeders are used throughout the plant. The ore is carried on a 42-in. (106-cm.) 8-ply conveyor belt, 1216 ft. (370 m.) long, to the fine-crushing department. The conveyor runs on a slight downward grade, the power consumption is very low, and discharges ore on to four all-steel impact screens of the Colorado type. These screens are a modification of the standard impact screen commonly used on finer ore but have been strengthened for heavy work. Four screens handle the mill feed of 10,000 tons daily, screening to a 1-in. opening. Double crimped wire screens are used and are made in the shops at the plant.

The oversize from these screens drops into a 2500-ton bin, which acts as storage for the feed to the 72 by 20-in. (182 by 50-cm.) rolls. The undersize from the screens drops onto a 42-in. 8-ply conveyor belt and is distributed by a tripper over bins feeding fine impact screens. The oversize is fed by a rotary feeder to a 36-in. rubber-covered conveyor belt discharging to 72 by 20-in. rolls. The belt and feeders are driven from the roll shaft, insuring stopping of the feed in case rolls are stopped suddenly for any reason. These rolls crush to through 1-in. and the two sets are capable of handling oversize from a 12,000-ton mill feed daily. The product falls into the skip storage bin of small capacity and is elevated by automatic skips to the level of the 42-in. tunnel conveyor discharging over six impact screens. The oversize from these screens then returns to a bin ahead of the 72 by 20-in. rolls.

The ore is hoisted by a 5-ton skip operated automatically by a 75–135-hp. Westinghouse hoist motor. The loading mechanism consists of an air cylinder connected to a rod, which opens the gate. The air is run into the cylinder through a three-way valve operated by a shaft which is

rotated by the descending skip. High-pressure water is available and can be used in place of air. The time of loading is determined by an oil dashpot, which at the same time throws the master switch operating the motor. The time of loading a 5-ton skip is 11 sec., and the complete cycle of operation with two skips operated from the same motor is 1 min. 50 sec. Flat plow-steel cable 3/8 in. by 5 in. (0.95 by 12.7 cm.) is used for hoisting two skips running from the same drum, one loading while the other is dumping. The hoisting distance is 100 ft. (30 m.). This is a new feature in milling practice and replaces dry elevators very effectively. The operating and maintenance cost is quite low, the cost of maintenance being $0.009 per ton milled.

The 72-in. (182-cm.) rolls, with conveyor and feeder, are driven by a General Electric 300-hp. alternating-current motor direct connected to the main line shaft; the practice throughout the plant is to use either direct-connected or back-geared motor drive instead of belt drives. The product is screened as before stated. The oversize returns to bins for further crushing and the undersize is carried by rotary feeders to the 42-in. (106-cm.) tripper conveyor, where it joins the undersize from the tunnel conveyor and is distributed over bins ahead of the impact screens over the 54-in. (137-cm.) rolls. The tripper is operated by a 15-hp. motor and can be moved over the length of the bin. The ore from this bin is screened by sixty impact screens of the Colorado type, making 600 vibrations per minute and equipped with 8-mesh 0.032-in. (8.3-mm.) wire-screen cloth and 9-mesh 0.028 in. (7.1 mm.) wire-screen cloth. The screens are equipped with manganese cams and tappets. The undersize from these screens falls to a 36-in. traveling conveyor distributing over the fine bins ahead of the concentrating department. At this point a sample of the mill heads is taken.

The oversize from the screens is conveyed by ten 24-in. (60.9-cm.) belt conveyors to ten 54 by 20-in. (137 by 50-cm.) Garfield rolls set to grind to 10-mesh. The feed to the rolls is sprayed to lay the dust. After grinding, the ore is hoisted by automatic 5-ton skips to the 54-in. roll feed bin and screened as before. Each two sets of rolls are driven by a 300-hp. motor direct connected to the line shafting. The 300-hp. motor is interchangeable with the 300-hp. motor on the 72-in. rolls and the skip-hoist motors are identical with the 72-in. hoists. The conveyor and screens are driven from the 54-in. roll shafts.

A 30-ton capacity electric traveling crane operates over the floor of the fine-crushing department and over a large repair room at the end of the floor. This crane facilitates repairs, as any part of a machine can be assembled in the repair shed and then carried to its place. By having a supply of spare parts time lost can be kept at a minimum.

The ten sets of 54-in. fine-crushing rolls have a capacity of 11,000 tons daily, or 1100 tons per roll to 10-mesh. The 54-in. roll shells give approxi-

mately 80 days' actual service and crush 96,000 tons of ore, or 17.6 tons of ore per pound of steel, when operating at full capacity.

From the impact screens the undersize drops onto two conveyors distributing over an 8000-ton storage bin with semi-cylindrical grooves in the bottom in each of which two rotary feeders for feeding the primary Garfield tables are placed.

The concentrating department is divided into four independent sections, each of which is equipped with twenty rotary feeders, ten primary Garfield tables, ten primary Wilfley tables, ten four-compartment Janney classifiers, five tube mills, ten secondary Garfield tables, and ten secondary Wilfley tables. Each section then is composed of five parts, each of which can be shut down independently of the others.

The crushed product from the rolls is fed from the fine bins by rotary feeders discharging into a 14-in. (35.56-cm.) launder to the Garfield tables. Water is here added for the first time, enough water being added to give proper consistency of feed to the tables. The launders all have sufficient slope so that water added is governed by table conditions.

The Garfield tables making the first roughing concentration are double-decked tables 4 ft. (1.2 m.) wide by 12 ft. (3.6 m.) long, with riffles extending the length of the table. The head motion is of special design, very heavy and strong, as experience had shown that exceptionally heavy duty would be required of these tables. These Garfield tables produce a rougher concentrate, which is cleaned on Wilfley tables, and a tailing, which is laundered to the Janney classifiers.

The principle of roughing concentration which has here been applied to gold milling for the first time is based on the fact that the roughing table will handle a large tonnage of material per area of deck, giving a low-grade concentrate with high extraction. The low-grade concentrate can then be cleaned on a finishing table. It can easily be seen that the tonnage handled per table will materially cut down the floor space required for mill operation on a large tonnage, thus reducing construction cost and simplifying the subsequent milling operation. In fact, the success of the milling operations at this plant are due to the low cost of roll grinding and to the low cost and high efficiency of the roughing concentration.

The Wilfley tables clean the rough concentrate of the minerals present, rejecting a tailing that joins with the Garfield table tailings and is laundered to Janney classifiers. The No. 5 Wilfley table is used, but the head motion has been strengthened by increasing the size of the shaft and making the thrust bar and pitman of steel instead of iron. The concentrate from the Wilfley table is laundered to the retreatment plant. This concentrate comprises galena, pyrrhotite, sphalerite, and arsenical pyrites, all of which act as carriers for the gold.

The tailings from the primary Garfield and Wilfley tables run to four-compartment Janney classifiers. There are ten classifiers in a section, each classifier taking its feed from one Garfield and one Wilfley table. Each classifier, on the standard feed of 2500 tons daily, is handling 250 tons daily. The maintenance of these classifiers is very small and one operator can look after forty classifiers and twenty tube-mills. The classifier makes five products and has two main functions: one to deslime the material so that this slime, which is very low in value, can be rejected and the other to separate the material coarser than 48-mesh for regrinding, as to liberate the values grinding to 48-mesh is necessary. In the first two spigots all material coarser than 48-mesh is dropped, the third and fourth spigots carry the sands finer than 50-mesh. The classifier overflow, which is all slimes, runs directly to the tail race. The first and second spigots discharge into a shovel or sand wheel box, which dewaters the feed, also discarding any fine material that might be left in the feed. This combination of classifier with shovel wheel gives an excellent feed to the tube-mills, as 90 to 93 per cent. of the feed will remain on 48-mesh screen and about 4 per cent. of over 48-mesh material passes through the third compartment of the classifier. The moisture to the tube-mill is maintained at about 33 per cent.

Each tube-mill takes the feed from the first two spigots of two classifiers. The tube-mill used is a cylindrical mill 7 ft. (2.1 m.) in diameter by 10 ft. (3 m.) long made by the Power & Mining Mach. Co. It is driven by a 75-hp. motor direct connected with a double reduction; the first reduction is through herringbone gear and pinion running in an oil bath, and the second reduction to the mill is by a heavy spur gear and pinion. The Komata-type liner is used in the mill, the plates being made out of worn-out roll steel straightened and formed in the blacksmith shop. The lifting bars and screen are made of manganese. The feed to the mill is through a three-tip spiral scoop. Each mill grinds the over-size from 491 tons daily, the product carrying about 30 per cent. on 48-mesh. No return of oversize is made to the mill.

Danish flint pebbles and adamant silica blocks are used for the grinding medium and, in case increase in capacity of the entire mill is made so that more than 10,000 tons is handled daily, additional tonnage can be ground in the mill by using steel balls and increasing the size of motor used. The consumption of pebbles from the beginning of operations has been 1 lb. per ton milled. The product from the tube-mills and the discharge from the third and fourth spigots of the classifier run to secondary Garfield tables, each mill feeding two tables. These tables make a final tailing, which runs to waste. The rough concentrate from each table feeds one No. 5 Wilfley table, which makes a tailing to waste and a concentrate which joins the primary Wilfley concentrate and is laundered to the retreatment plant.

The water used in the plant is obtained, in the summer, from the freshwater supply, which will operate the mill on a 10,000-ton basis for about 5 mo.; during the remainder of the year, make up water is supplied by the salt-water pumps. These are three two-stage Byron Jackson turbine pumps of 1000-, 2000-, and 3000-gal. per min. capacity, respectively, against a 275-ft. (83-m.) head. The water consumption is about 0.60 gal. per min. per ton ore milled per day.

## Retreatment Plant

From the concentrating department, the concentrates are laundered to elevator pits and carried to the top floor of the retreatment plant by a 12-in. bucket elevator on each half of the plant. One-half of the retreatment plant treats the concentrates from one-half of the concentrating department. From the elevator, the concentrate is distributed to twelve standard No. 5 Wilfley tables, which remove a rough lead concentrate carrying all the free coarse gold. This lead concentrate is then retabled on one No. 5 Wilfley table to separate the free gold from the shipping concentrate. This separates some of the coarse lead with practically all of the free gold, the tonnage of this product being about 1000 lb. daily. This separation is made by a separate riffle cutting into the regular riffles.

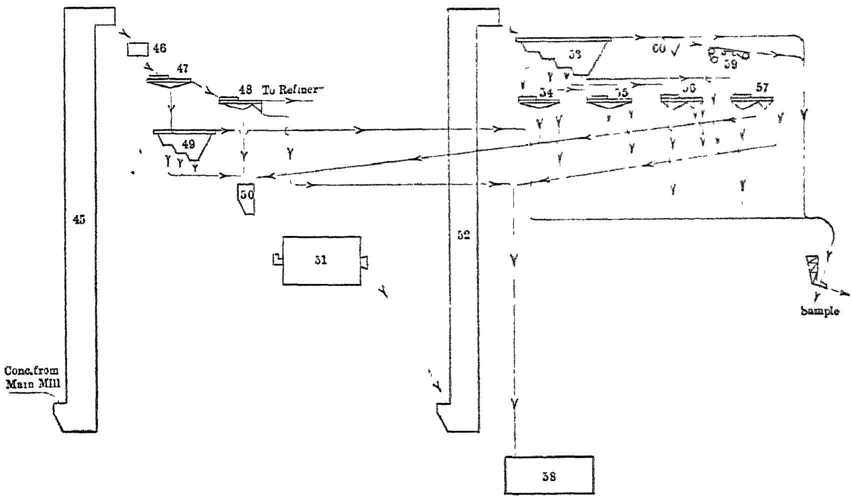


Fig. 3.—Flow sheet, one-half retreatment plant.

45. 12-in. bucket elevator.
46. Six-way distributor.
47. Twelve No. 5 Wilfley tables.
48. Two No. 5 Wilfley tables.
49. Three-compartment Janney classifier.
50. Dewatering box.
51. 7 by 10-ft. standard tube-mill.
52. 12-in. bucket elevator.
53. Four-compartment Janney classifier.
54. Three No. 5 Wilfley tables.
55. Three No. 5 Wilfley tables.
56. Two No. 5 Wilfley tables.
57. Two No. 5 Wilfley tables.
58. Two bins for shipping concentrate.
59. Three 6-ft. 1-shell vanners.
60. Two cone tanks.

This material was treated at first in crucible furnaces but is now treated in an open-hearth furnace designed at the plant. Iron shavings are added to take up the sulfur present and the resultant products are a high-grade lead bullion and a matte. This lead bullion can be very accurately sampled and is shipped direct to the smelter.

The reject from the single table in each section handling the lead is a shipping concentrate carrying about $800 gold per ton and about 50 per cent. lead. This is stored in bins and shipped as often as boat sailings permit. The tailings from the twelve primary tables are dewatered into a three-spigot Janney classifier. The spigot discharge is then ground in a 7 by 10-ft. mill to 100-mesh. After grinding, the pulp is elevated to a standard four-compartment Janney classifier. The first spigot feeds three Wilfley tables and the second spigot three tables. These tables make a concentrate for the shipping bin and a middling which returns to the tube-mill for further grinding. The third and fourth spigots each discharge to two tables, which make a concentrate to the shipping bin, a middling which returns for further grinding to the tube-mill, and a tailing that goes to waste. The classifier overflow is settled in cone tanks and then fed to six vanners, which make a shipping concentrate. The final shipping concentrate then assays about $400 gold and 40 per cent. lead, and 50 oz. of silver per ton. This shipping concentrate is in a ratio of 1 to 1000 of the original ore milled and contains about 44 per cent. of the gold recovered, with about 56 per cent. in the lead bullion. The tailings from the retreatment plant join the tailings from the concentrating mill, are sampled, and then run to the bay as waste.

## Results Obtained

Following is average of series of samples throughout the plant, showing the results of the various concentrations of gold values:

| | | | |
|---|---|---|---|
| General heads | $1.25 | Tube-mill feed | $0 49 |
| Primary Garfield tailings | 0.347 | Tube-mill discharge | 0 49 |
| Primary Garfield concentrate | 13.15 | Secondary Garfield tailings | 0.17 |
| Primary Wilfley tailings | 0 844 | Secondary Garfield concentrates | 5.45 |
| Classifier feed | 0.523 | Secondary Wilfley tailings | 0.46 |
| First spigot classifier | 0 60 | Secondary Wilfley concentrates | 5.36 |
| Second spigot classifier | 0 32 | General concentrating tails | 0.177 |
| Third spigot classifier | 0.26 | Retreatment tails | 1.00 |
| Fourth spigot classifier | 0.26 | General tails | 0.20 |
| Classifier overflow | 0.16 | | |

The average value of the concentrate going into the retreatment plant is about $70 gold per ton; and after the initial lead concentrate is made, iron concentrate which is reground and treated is left, assaying $19 per ton. After treating a tailing of $1 is made.

Characteristic screen analysis of the mill heads, mill tailings and retreatment tailings are as follows, Tyler standard screen sieves being used.

GENERAL HEADS

| Mesh | Per Cent, Material | Cumulative Per Cent, Material | Assay | Per Cent, Value | Cumulative Per Cent., Value |
|---|---|---|---|---|---|
| Plus 10........ | 2 5 | 2.5 | \$0 83 | 1 5 | 1.5 |
| Plus 20........ | 27.7 | 30.2 | 1 24 | 24 6 | 26.1 |
| Plus 28........ | 21 0 | 51.2 | 1.45 | 21 8 | 47.9 |
| Plus 48........ | 5 7 | 56 9 | 3.10 | 12 7 | 40.6 |
| Plus 65........ | 5.6 | 62 5 | 3.72 | 15 0 | 75.6 |
| Plus 80........ | 3.2 | 65.7 | 1.24 | 2.8 | 78.4 |
| Plus 100........ | 3 7 | 69.4 | 1 86 | 4 9 | 83.3 |
| Plus 150........ | 5.9 | 75.3 | 0 83 | 3 5 | 86.8 |
| Plus 200........ | 1.7 | 77 0 | 1 03 | 1.3 | 88.1 |
| Minus 200........ | 23.0 | 100 0 | 0.72 | 11.9 | 100.0 |
| | 100 0 | | \$1.39 | 100 00 | |

GENERAL TAILS

| | | | | | |
|---|---|---|---|---|---|
| Plus 20........ | 1.48 | 1.48 | \$0.15 | 1.18 | 1.18 |
| Plus 28........ | 9 08 | 10.56 | 0.15 | 7.22 | 8.40 |
| Plus 48........ | 10.43 | 20.99 | 0.21 | 11.60 | 20.00 |
| Plus 65........ | 12.59 | 33.58 | 0.26 | 17.34 | 37.34 |
| Plus 80........ | 8 08 | 41.66 | 0.26 | 11.13 | 48.47 |
| Plus 100........ | 8.88 | 50.54 | 0.26 | 12.23 | 60.70 |
| Plus 150........ | 7.95 | 58.49 | 0.26 | 6.32 | 67.02 |
| Plus 200........ | 7.17 | 65.66 | 0.15 | 5.70 | 72.72 |
| Minus 200........ | 34.34 | 100.00 | 0.15 | 27.28 | 100.00 |
| | 100.00 | | \$0.19 | 100.00 | |

RETREATMENT TAILS

| | | | | | |
|---|---|---|---|---|---|
| Plus 65........ | 2.70 | 2.70 | \$0.41 | 1.29 | 1.29 |
| Plus 80........ | 1 45 | 4.15 | 0.52 | 0.88 | 2.17 |
| Plus 100........ | 9.55 | 13.70 | 0.62 | 6.88 | 9.05 |
| Plus 150........ | 32.95 | 46.65 | 0.72 | 27.58 | 36.63 |
| Plus 200........ | 4.35 | 51.00 | 0.93 | 4.70 | 41.33 |
| Minus 200........ | 49.00 | 100.00 | 1.03 | 58.67 | 100.00 |
| | 100.00 | | \$0.86 | 100.00 | |

The work has shown that economic grinding of the ores for final concentration is through 48-mesh; that the slimes carry very low values; and that regrinding and concentration of the iron concentrate will liberate

the gold values, permitting the iron to be thrown away. This, of course, reduces the smelting costs and losses to a minimum. Also, the lead concentrate produced carries 40 per cent. lead, which is a good smelting product. The concentrates are sacked and shipped to the Selby Smelting works for treatment. At the beginning of operations we were using amalgamation as an additional process in the retreatment of our concentrates but found this unnecessary.

### Organization

The organization of the milling division is as follows: Superintendent of mill in charge of division including mill operation and construction, machine and carpenter shops, docks, warehouse, commissary, etc. The assistant superintendent of mill is directly under the superintendent. The mill-operating organization includes three general foremen, one for each shift. The master mechanic has charge of repairs in the mill division and also has charge of the shop. His force is under different foremen. Each foreman is responsible for the repairs in his department and is entirely separate from the operation. The operating force is strictly operative and when a break-down occurs the repairmen take charge. In the same way, the oil foreman is responsible for the lubrication of all machines. Whenever possible parts are sent to the shop for overhauling or else repaired in the repair sheds at the end of each floor.

The sampling, refinery, and assay office and experimental department are under the metallurgical engineer.

The accounting for all divisions is under the auditor, as is the accounting end of the warehouse and the retail stores and butcher shop. The warehouse is operated by the head storekeeper. The boarding and lodging houses, together with the club house, are under the chief steward, who is responsible to the mill superintendent. All supplies are bought through the purchasing department. A requisition purchase of material originates with the head of a department, is numbered by the storekeeper and approved by the superintendent, after which it is sent to the local supply agent, who orders material through the purchasing agent in Seattle, Salt Lake City, or New York, depending on the class of material.

### Sampling and Assaying

Great pains are taken at the mill to insure accurate sampling of all vital products. For the headings to the concentrating department, the undersize from each screen is sampled automatically and cutters ½ in. (12.7 mm.) wide are operated by a Scobey timing device throwing the four-way air valve connected to the air cylinder. A cut is taken every

12 min. and samples are collected every 24 hr. On a basis of 6000 tons of ore milled, the combined sample averages about 3000 lb., or ½ lb. per ton milled. This is carefully coned and ringed several times and then quartered, opposite quarters being saved together and about a 50-lb. sample cut from each of the two cuts by additional ringing and coning. These then are ground to 100 mesh in the laboratory and riffled to assay pulp size, weighing about 4 lb. each. The assayer runs four charges of 2 assay-tons each on each sample, combines the resultant silver-gold buttons, and then parts and weighs them together. The average of the original and the duplicate is then taken as the average for the day.

The stream of tailings from each half of the mill is cut by an automatic sampler every 10 min. operated as for the head sample. These samples, amounting to 150 lb. each, are collected at the end of every shift. They are rolled and riffled, a 10-lb. sample from each shift sample being put through the pulverizer and ground to 100 mesh. From each of the shift tailing samples, the assayer runs four samples, each of 3 assay-tons, making a total of twelve crucibles for the three tailing samples.

Concentrate production is sampled automatically and at the time of shipment an auger sample is taken from each sack. Special sampling of different products throughout the plant is kept under way at all times.

## Costs of Operation

While the plant has not treated its full tonnage, the results of operating in the year 1917, when for a period the tonnage was maintained at 7000 tons, gives an idea of what costs will be under full operating basis. These costs are as follows: Tons milled per day, 7023.

| | | | |
|---|---|---|---|
| Coarse crushing | $0 02804 | Labor cost | $0.09746 |
| Fine crushing | 0 09432 | Supplies | 0.05454 |
| Concentrating | 0 05136 | Shop expense | 0.03787 |
| General mill | 0 02778 | Sundries | 0 04970 |
| Power and light | 0 01467 | Total | $0.23957 |
| General overhead | 0 02340 | | |
| Total | $0.23957 | | |

## Power Consumption

Power is supplied from a hydroelectric plant at Salmon Creek and Annex Creek, with a combined capacity of 11,000 kw. It is transmitted by cable on steel pole lines to the mill substation, where it is transformed from 22,000 volts to 440 volts and distributed to the various machines in the mill. Each circuit has recording watt meters and Bristol recorders for measuring power used in each group of machines. A distribution of the horsepower per ton milled is as follows:

| | Horsepower Per Ton Per Day | | Horsepower Per Ton Per Day |
|---|---|---|---|
| Coarse crushing | 0 0217 | Beach pumping plant | 0 0489 |
| Fine crushing | 0.2510 | Lighting circuits | 0 0138 |
| Concentrating | 0.1993 | Motor generator | 0.0014 |
| Retreatment plant | 0 0195 | Total | 0 0641 |
| Total | 0.4915 | | |

The total kilowatt-hours per ton ore milled on normal basis is 7.946.

## Milling-plant Accessories

To care for the married men, there are erected, to date, 40 cottages, from three rooms to six rooms. These cottages are located at the beach, about 5 min. walk from the mill. All cottages are plastered and well finished and have baths. One heating stove and range is supplied each cottage by the company. For employees that cannot be accommodated in the townsite cottages, grounds are leased at a rental of $1 per month, on which any employee can erect a dwelling, which at the end of seven years reverts to the company. At the present time there are about 45 of these dwellings.

For the single men, there are six bunk houses, one being reserved for foremen; this is fitted with rugs, chiffonier, table, and bed for one man per room. The remainder of the houses are for two men to a room; but during the operating period nearly every man has a single room. These houses are equipped with dry rooms, bath and showers, are plastered and steam heated. The men board at a company boarding house, a large modern building equipped with all labor-saving devices common to a well-equipped hotel café. The food furnished is good and wholesome, with plenty of variety and is well served. There are accommodations for 350 men at each meal, which consists of two kinds of meat, two or three vegetables, and dessert. Board and room costs $1 per day, with $2.50 per month extra for single rooms.

In conjunction with these, there are maintained an up-to-date meat market with cold-storage facilities and a bakery accommodating both boarding-house and family trades. For the family trade, the company operates a retail store, which carries all standard goods and a big stock of men's working clothes, which are sold at practically the same price as in the town of Juneau.

A deduction of $1.50 per month is made from each employee for hospital and doctor services, which includes all service during the time employed. The hospital is the St. Ann's Hospital at Juneau; a modern well-equipped building with the best medical attendance at all times. A club room is maintained for the benefit of the employees and Mrs. J. R. Whipple has given a fine library room containing about 4000 volumes,

in memory of Mr. James Ray Whipple, one of the promoters and afterward assistant manager of the property, who died during the construction period.

## DISCUSSION

Graham Bright, Pittsburgh, Pa.—I would like more information about the operation of those 5-ton skip hoists, mentioned on page 495. When this proposition first came up, the question of handling this material with an automatic hoist was put up to the various manufacturing companies, and we were a little skeptical as to whether we could take care of it with an alternating-current motor. It was a little hard to produce an automatic hoist, using an alternating-current motor, without complicating the motor or the control. When we found that the skips were to be loaded automatically, that is, the same amount put into the skip each time, it made the question somewhat simpler. So that these hoists were put in with an automatic control; when the skip receives a certain amount of material, it depresses a spring, which cuts off the feed and starts the hoist operating, goes through the cycle, slows down, and dumps, then the skip on the other side fills and goes through the same operation. From what I can gather, this plan has been successful; there has been little or no complaint. The author mentions that the operation and maintenance of this system is very low, being 0.9 cent per ton-mile. Does anybody know about what the cost would be, using the ordinary elevator system?

W. L. Remick, Chrome, N. J.—I do not know that Mr. Daveler would approve of my speaking for him, but I happened to be with him during all the experimental work at Thane, through the construction work, and during 8 months operation of the mill. The skips were the Wellman-Seaver-Morgan type and, I believe, were designed for that job. Considerable difficulty was experienced before they were working well. In fact, they had one skip for each set of rolls, and for at least a month it required two men to handle each skip, the main trouble being with the tripping device at the bottom. The skips are operated with a flat cable, instead of a round one, so that as the skip goes down, the drum becomes smaller very rapidly, and with the constant speed of the motor, the skip will almost come to a stop before it lands on the cradle at the bottom. The trouble was that they could not regulate the blow at the bottom. Sometimes the cradles would operate a little too quickly; sometimes the gate, which opens automatically as the skip lands, would open a little too soon and fill the pit full of ore before the skip was under the chute. When that happened, the flow of ore could not be stopped. Finally, a dashpot was placed underneath the skip, so that the speed was gradually decreased and, before I left, they had the arrangement under perfect

control. One man would handle the whole set of skips. I think Mr. Daveler has, in a general way, covered the details quite thoroughly. He has avoided, naturally, the troubles they have encountered, but there is no doubt that his system, especially the system of crushing with rolls, has turned out better than a good many people thought it would.

Graham Bright.—I have just one more question that might help to clear this matter. In a new thing like this, we naturally would expect more or less trouble until we got it into operation. If you had a similar plant, would you put in these skips or a dry elevator system?

W. L. Remick.—I would put in this system.

# Crushing Practice, New Cornelia Copper Co.*

BY W. L. DUMOULIN,† C. E., AJO, ARIZ.

(Chicago Meeting, September, 1919)

A RATHER detailed description of the entire plant and leaching process was given in a paper recently presented to the Institute,[1] so this paper will cover briefly only the crushing practice of the New Cornelia Copper Co. for the year 1918.

The ore, which is mined by steam shovels and loaded in side-dump cars, passes through two crushing plants. These crushing plants were constructed to crush the ore required by a leaching plant of 5000 tons daily capacity, during a crushing period of 16 hours. The primary plant reduces to less than 3 in. (76 mm.) and the secondary plant, to 1/4 in. (6.35 mm.), which is sufficiently fine for satisfactory percolation of solution through the ore in the leaching tanks, and to give good extraction. The ore is then conveyed to the leaching tanks by a system of 28-in. (71-cm.) belt conveyors. On the way to the leaching tanks, it passes through an automatic sampling plant, where a 1-per cent. sample is taken.

From the primary crushing plant, the ore is conveyed by two 36-in. (91-cm.) belt conveyors to a 10,000-ton steel storage bin, with a reinforced-concrete flat bottom. The ore discharges from the bottom of this bin onto four 20-in. belt conveyors, which deliver it to the four units of crushers in the secondary crushing plant. There is no storage bin between the mine and the primary crushing plant. The ore breaks very coarse, is very hard, and contains a great many boulders, some as large as 4½ by 4½ by 10 ft. (1.3 by 1.3 by 3 m.). Jams form in the bowl of the coarse crusher in the primary crushing plant, as a consequence, and are freed by means of an immense steel hook operated from a 40-ton electric traveling crane.

## PRIMARY CRUSHING PLANT

The primary plant consists of one No. 24 Gates, style K, gyratory crusher, followed by four Gates No. 8, style K, gyratory crushers. All

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* The substance of this article was originally sent to Robert H. Richards, Professor of Mining Engineering and Metallurgy, Massachusetts Institute of Technology, Boston, Mass., in the form of a letter.

† Assistant General Superintendent, New Cornelia Copper Co.

[1] H. A. Tobelmann and J. A. Potter: First Year of Leaching by the New Cornelia Copper Co. *Trans.* (1919) **60**, 22.

the ore from the mine is dumped directly from the mine-ore cars, loaded with about 35 to 37 tons of ore each, into the bowl of the No. 24 gyratory. This crusher will take boulders as large as 4½ by 4½ by 10 ft. and reduce them to approximately 9 in at a rate of 500 tons of ore per hour. The ore crushed by the No. 24 crusher discharges into the bowls of the four No. 8 gyratories, after passing over grizzlies with 3-in. (76-mm.) spaces. The No. 8 gyratories take the 9-in. discharge from the No. 24 and reduce it to approximately 4 in. There are two No. 8s on each side of the No. 24, which discharge onto one of the 36-in. belt conveyors carrying the ore to the 10,000-ton storage bin. All crushers in this plant are belt-driven by alternate-current induction motors.

We found that when crushing to the size indicated, the discharge from the No. 24 would "flood" the No. 8s; therefore, as the head is not adjustable, the bottom row of concaves in the No. 24 was replaced by a row of thicker concaves, and a new lower section of the mantle of larger diameter was placed on the head, so that this crusher now reduces the ore to approximately 6 in. (15 cm.). This throws more work on the No. 24 crusher and slows its rate of discharge sufficiently to give a fairly uniform feed to the No. 8s, resulting in an increase in the capacity and a reduction in wear and cost of repairs in connection with these smaller crushers. The increase in the capacity of these smaller crushers was so great that it was possible to set out the bottom row of concaves, and so obtain a product of less than 3 in. In addition to more evenly distributing the work among the crushers in the primary plant, resulting in more efficient operation, this also balances the work between the primary and secondary crushing plants in a more economical manner. Previously, the rate of crushing in this plant was limited by the No. 8s, but now it is determined by the No. 24 gyratory, which has an average capacity of 400 to 450 tons per hour, crushing to less than 6 in., the rate depending on the percentage of large boulders in the ore crushed.

## Secondary Crushing Plant

The secondary plant consists of four units of 48-in. (122-cm.) Symons vertical shaft (pillar) disk crushers, each unit consisting of three crushers, one coarse Symons and two fine Symons. The coarse and fine Symons are identical with the exception of the upper disk, which, in the coarse crusher, has a grinding surface 4 in. wide, and in the fine crushers, 6 in. wide. Each crusher in this plant is driven by a special direct-connected 75-hp. alternating-current induction motor having a speed of 400 r.p.m. As the ore is harder than anticipated, it was soon demonstrated that to crush the 5000 tons in the desired time would crowd the crushers too much for economical maintenance, so a fifth unit of three 48-in. Symons vertical shaft disk crushers has just been installed.

The ore from the primary crushing plant is conveyed to the storage bin of 10,000 tons capacity, from which it is fed to the coarse Symons at a uniform rate, and crushed to approximately ¾ in. (19 mm.) The discharge from these coarse crushers is divided, one half passing to each of the two fine Symons crushers in each unit, where it is crushed to approximately ¼ in. Screens ahead of the fine Symons crushers bypass a large percentage of undersize. The capacity of a unit depends somewhat on the character of the ore, and varies from 90 to 100 tons per hour. The coarse Symons have a capacity of 100 tons per hour, crushing to ¾ in. with small cost of repairs; but the fine Symons have each a capacity of only 50 tons per hour crushing to ¼ in. with rather heavy repair costs. As this type of crusher may be considered in an experimental stage for crushing as fine as ¼ in., it developed that certain parts of the machine required strengthening, when it was found advantageous to substitute cast steel for cast iron as well as to otherwise strengthen this design. But even under these circumstances, this type of crusher has proved satisfactory for the service required of it and gives the most uniform product that has the minimum amount of oversize and fines. A great deal of trouble has been experienced with tramp iron in the Symons disk crushers, but this trouble has been eliminated largely by the suspension of large powerful magnets over the conveyors leading from the primary crushing plant. These magnets were installed in addition to the magnetic pulleys already in service.

The records show that the manganese steel wasted is as follows:

| | Lb per Ton of Ore Crushed |
|---|---|
| Primary Crushing Plant | |
| In all mantles of gyratory crushers | 0 0032 |
| In all concaves of gyratory crushers | 0 0081 |
| Secondary Crushing Plant | |
| In all disks of Symons crushers | 0 059 |
| Total Babbitt Used in Each Plant | |
| In primary crushing plant | 0 0045 |
| In secondary crushing plant | 0.0097 |

The above items are given as they represent figures on the parts most subject to wear.

In the primary crushing plant, the average power consumed by one gyratory crusher to crush one ton of ore was 0.21 kw.-hr.; in the secondary crushing plant, the average power consumed by a Symons disk crusher to crush one ton of ore was 0.701 kw.-hr.

All the preceding figures are averages for the year 1918. The total tonnage crushed during the year was 1,775,000. The following is the average of screen sizing tests for the year. The samples were taken from

the discharge from the fine Symons disk crushers as it passed to the conveyor carrying this crushed ore to the leaching tanks:

| | | | |
|---|---|---|---|
| On 3 mesh | 26 7 | On 10 mesh | 7 0 |
| On 4 mesh | 17 3 | On 14 mesh | 4 9 |
| On 6 mesh | 12 7 | On 20 mesh | 4 1 |
| On 8 mesh | 9 0 | Through 20 mesh | 18 3 |

Ore crushed as indicated by these average screen sizing tests gives very satisfactory results in the leaching tanks.

# Chilean-mill Practice at Portland Mill

BY LUTHER W. LENNOX,* E. M., M. E., VICTOR, COLO.

(Chicago Meeting, September, 1919)

THE purpose of this article is not to compare one type of grinding machinery with another and to conclude from a series of tests that one particular machine is superior to all others. Neither is the reader asked to tire himself with the details of many tests to prove the statements herein made. The Chilean mill stands today almost an outcast from the milling world. It ranks with the applicant for a job who has been given a trial along with others and condemned because he was not able to perform the definite task assigned as well as the others, and was forever eliminated. The judges did not tarry long enough to try this aspirant in a field of a slightly different nature. Even though it has been condemned, the writer contends that the published records will indicate that the trials were never shifted into the field of greatest efficiency for the Chilean mill. Those of us who cling to this outcast as a fine grinder in our milling operations do not wish, naturally, to be condemned also, with the machine, and yet we feel that we are classed as behind the times, at least in this phase of our milling judgment. Hence there is considerable personal feeling in our efforts.

During the last few years there have been published several papers on the relative merits of the various grinding machines. In two of these articles a comparison was made between Chilean mills and cylindrical grinders. In each case the Chilean mill came out second best from about all standpoints. The outstanding feature in these comparisons would seem to be the decided attempt to keep all conditions the same, so that each machine could be judged on its merit. As Mr. David Cole states,[1] "We are inclined to regard a direct comparison of grinders arranged side by side, and making a product that affords as nearly as possible the same screen measure, as the Supreme Court in these grinding matters." The writer agrees that this is the very best method for true comparison between machines. The results in the two detailed tests should give the relative merits of the two grinders in doing the work assigned. And they probably do. The conclusion drawn in each case was adverse to the Chilean mill as a fine-grinding machine. However, the duty placed upon the machines was that of converting material that had already been

*Assistant Superintendent, Portland Mill.

[1] *Trans.* (1916) **55**, 705.

made to pass a ½-in. (1.27-cm.) screen to a product that would practically pass a 20-mesh screen. Its competitor in each case was a machine known to give the greatest efficiency in making a reduction upon just this type of feed or finer.

The two articles mentioned, after recording the data from the tests, draw their conclusions: The outstanding and important one is the inferior mechanical efficiency of the Chilean mill. The other conclusion wherein the Chilean mill fails is its inability to compete in comparative costs of operation and repair. The same conditions that made the Chilean mill mechanically inefficient also made the cost comparison place it in the same category. Hence this paper will be confined to showing how the efficiency of a Chilean mill may be increased greatly over that recorded in either of these two tests.

The first of these articles, that by Robert Franke[2] has been given wide publicity and its results apparently accepted with two possible exceptions; one a protest in favor of fairer treatment of the Chilean mill by Herbert Megraw[3] and the other a doubt as to conclusions drawn by Mr. Franke, as indicated by Arthur O. Gates.[4]

The test was made over a considerable period of time with parallel arrangement of machines. The feed and product were such as to allow one machine to perform with its best efficiency. Mr. Franke, in upholding this particular mill as a result of the Miami tests, states that this mill is not yet out of the experimental stage and points out the possibility of better performance. In the light of developments since that time he was undoubtedly right, as the mill has become a more flexible machine as to the character of its feed and output. By his silence in reference to the Chilean mill one may conclude that its experimental stage was long ago past, and yet a minus ½-in. feed was apparently considered fair for it. The Chilean mill would seem to be the less understood of the two machines.

The second article dealing with a comparison based upon data gathered from a parallel run is that by F. C. Blickensderfer.[5] These tests were conducted at the concentrator of the Detroit Copper Mining Co. at Morenci, Ariz., in order to test the relative grinding efficiencies of three different mills. These mills were operated under practically parallel conditions. The criticisms as a result of these tests by Mr. Blickensderfer are similar to those of Mr. Franke based on the Miami tests, as far as the Chilean mill is concerned.

In his comparison of costs, Mr. Blickensderfer condemns the Chilean mill for its numerous auxiliary parts, as screens, screen frames, mullers, feed spouts, plows, mortars, and the power. Of the points mentioned

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[2] *Trans.* (1913) **47**, 50. [3] *Eng. & Min. Jnl.* (Nov. 15, 1913).
[4] *Trans.* (1913) **47**, 58. [5] *Trans.* (1916) **55**, 678.

that are serious enough to be considered, the screens must be made up upon the frames ready to put into the mill; the mullers and dies too must be replaced when worn out, but as for feed spouts and mortars, there is no cause for considering the matter of costs after original installation. Plows are not used at all by the Portland.

My purpose in this paper is to indicate where a very radical change might have been made in these tests in the performance of the Chilean mill. In justice to that machine I contend that a comparative test has not yet been made, or at least not published, where the true relative efficiencies have been obtained.

## Fine-grinding Installation at the Independence Mill

The primary grinders consist of six 6-ft. (1.8-m.) Wellman-Seaver-Morgan Akron Chilean mills, taking as a feed the product from two 20 by 72-in. (50.8 by 182.8-cm.) Allis Chalmers rolls of the Garfield type. Secondary grinding upon the classified sands from the Chilean mills is done with six 6 by 6-ft. ball-mills made by the Colorado Iron Works. The ball-mills discharge to Akin classifiers, which return the sands in closed circuit. Each Chilean mill and ball-mill is driven by a 100-hp. motor. The twelve Akin classifiers are operated from one line shaft driven by a 10-hp. motor. All plunger feeders supplying the ore to the mills are driven by one motor of 8-hp. from a common line shaft.

This installation of Chilean mills was the result of 8 years of experience on Cripple Creek ores at the Victor plant. There, however, the discharge passed through 18- or 30-mesh screens directly to gravity tables. This installation was always satisfactory and would undoubtedly have been followed at the Independence plant had there been no possibility of handling higher grade of ore than at the Victor mill. Metallurgically, a Chilean mill will give a Cripple Creek ore of $3 or less grade an efficient commercial grind for cyanide purposes through a 30-mesh screen, making unnecessary a regrind, except upon a small bulk of middlings, which may be taken care of in one ball-mill in a plant of 1400 tons original feed. Due to the possibility of a higher grade ore than $3, it was deemed advisable to install a classification and ball-mill system of regrinding. This combination would assure any necessary flexibility of operation as a result of varying grades of heads to the mills. High tonnage and the consequent coarser grind could be practised with $3 or less mill heads, while a fine grind with a less resultant tonnage would assure good metallurgical work upon the higher grades of ore when they came. Four ball-mills at present take care of the classified sands from six Chilean mills and are capable of adding to the work already done by the Chilean mills so as to give a combined overflow from both sets of classifiers of sufficient fineness to assure the commercial tailing on the

present grade. Five Chilean mills are driven at 37 r.p.m., the sixth at 29 r.p.m. due to its lack of proper steel shafting. The direction of rotation is not the same on all mills: this allows pinion gears to be worn on both sides of the teeth by changing to a mill of opposite direction of rotation. The power is maintained at approximately 110 hp. Feed to mills is a maximum of 2½ to 2¾ in., while the ratio of feed pulp is maintained at about 3 to 1.

## Delays

The regular shutdowns in the ordinary operation of Chilean mills is, first, a daily one on each mill for oiling and examination of steel and other parts, if necessary; this is an actual loss of 20 min. running time per mill. Screens and steel each necessitate a shutdown for replacement. One source of delay is avoided at the Independence plant; that is the stopping of the mills to take out stray pieces of steel, as drill bits, sledge hammers, and similar material that the magnet has missed. While oiling, the operator generally throws out some of this. However, it is not a menace, but on the other hand probably aids the attrition action on the ore. With a fine feed, where mullers and die are in close proximity, stray steel is a cause of delay as well as a source of danger of breakage of mill parts, due to the bumps the mullers get under these conditions. See Table 10.

## Estimation of Tonnage

Since practically all input to the mill is Portland ore, the necessity is lacking for an elaborate weighing system. The mine department has a check based on cars delivered, while the method used by the milling department has proved very satisfactory at the Victor mill, where there was less tonnage and an opportunity for a careful comparison with the mine methods. Once every other day, each mill is stopped and the feed is made to discharge into a box for twenty strokes of the feed plunger. This is weighed and the average of the five latest weights is kept and applied as a factor to the number of strokes taken by the plunger as indicated on the recording counter for the previous 24 hr. This constitutes the mill tonnage for each day's run. This method may appear crude, but it has the advantage of allowing complete data for each mill on steel, screens, and delays based on the individual tonnage of that unit.

## Screens

During operation at the Victor mill, which is now shut down, 30-mesh screens were used on the better grades of ore, from $2.60 to $3.20 average, and 18-mesh on that from $2.60 down to $2. The reason for this screen choice was to obtain the most economical balance of grinding costs,

tonnage, and tailings losses, while still being able to subject the product to tabling.

At the Independence plant conditions are somewhat different, and the necessity for a finer grind than is possible with 30-mesh screens would probably arise. However, to obtain commercial success so far, tonnage has been the governing element rather than a fine degree of grinding. In other words, the grade of the ore has so far justified only a comparative coarse grind and has demanded a considerable tonnage. This has necessitated the use of coarse screens—6-mesh on five of the Chilean mills with the consequent high tonnage and the corresponding decreased grinding reduction. One mill has been maintained with 18-mesh screens to take care of any higher grades of ore. Many screens with openings ranging between 6- and 30-mesh have been used. Made-up screens upon frames are at hand for each mill and, consequently, the delay due to taking out old and putting in new screens is only 10 min. on the average; with 6-mesh screens, the life is so long that total delays due to this replacement are small. The helper makes up the screens ready for use during spare time from regular operation.

It is not necessary for the operator to give as close attention to small breaks in the screens as it was in the Victor plant where oversize was immediately a menace upon the tables, and in the launders and pumps. The ball-mills will take care of small amounts of oversize for a short time without serious inconvenience in the concentrator department. See Table 12.

## Steel

The dimensions of the dies and mullers are given in the table on steel consumption with the corresponding weights into and out of the mill. Wearing of steel is the all-important element, for by the proper setting of the mill from day to day the maximum tonnage and grind may be obtained with the least consumption of power and steel. As the steel of dies and mullers is wearing at a definite rate for every ton of ore, it is necessary for the head operator to set the mill each day, in order to balance the relative position between die and mullers. The latest mills are so constructed that the trunnion shaft is, in reality, hinged on the drive head, allowing the muller to raise and lower without any stresses on the mill proper. This arrangement places the weight of the muller core plus the muller on the die at all times. The mill is so set that the muller will revolve upon practically a horizontal trunnion shaft when running at 110 hp.; that is, the plane through the center of the muller will be a vertical one. Each day the drive head must be lowered to maintain this plane vertical for the mullers and thereby assure even wear of the mullers and die. For example, if a mill is continually set for 110 hp. and run at 90 hp., the steel will wear too fast on the inner edge of the die and outer

edge of the mullers and will come out of the mill heavy. The raising or lowering of the drive head is done with a hand wheel and may be accomplished while the mill is running. When a new die or muller set is placed in the mill, the best work will not be possible until the concaving of the steel is well started. Aside from all varieties of steel, stone and cast-iron dies have been tried; these latter were not successful. For those who may not be familiar with the shape the steel takes in wearing, reference may be made to Fig. 1. While the setting of steel each day requires a man who gives considerable attention and study to this work, we have had no trouble in training men to do it.

The method of replacing steel tends to the least possible loss of time. A traveling Whiting 8-ton electric crane is installed over the length of

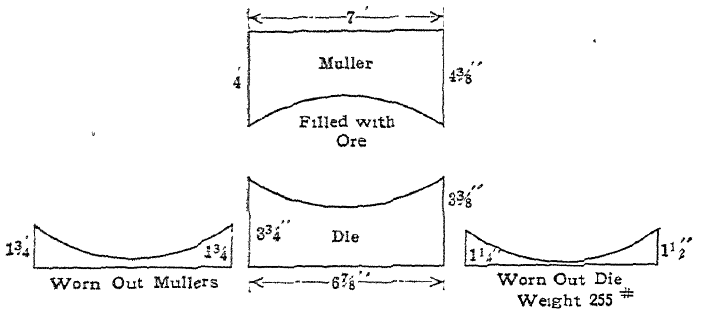



Fig. 1.—Mullers and die about as they appear when operating with ore between. Also the shape taken by them. The worn out steel is as good as we ever get and better than the average.

the Chilean mill floor and one end extends over points of steel storage and working space. An extra set of muller cores on trunnion shafts with mullers wedged on and ready for immediate installation on any mill, is always at hand. Mullers are wedged to their cores with both soft- and hard-wood wedges. Hard wood is more effective in preventing slippage when the mullers become thin. Dies are keyed in and concrete used to insure no slippage when the die becomes thin. The time consumed for these changes is indicated in the accompanying tables. See Table 11.

## Effect of Speed of Mill

During our experience with Chilean mills, the speeds have varied from 29 to 41 r.p.m. The increased speed will naturally give a greater capacity, and efficiency is also increased. Steel and screen consumption are approximately in proportion to the tonnage, while total delays at high speed are greater, as would be expected. The mills are equipped with special steel trunnions shaftings and with correspondingly strong thrust bearings to take care of the increased mechanical strains. See Table 1.

### Effect of Feed Pulp Dilution

Apparently a Chilean mill does its best work when the feed pulp dilution is about 3 to 1. The splash under the dilute conditions gives the mill the appearance of having an excess of "Pep." Operators, on this account, very much prefer to run under these conditions. The actual gain over 1 to 1 feed conditions is capacity. The efficiency remains the same, or nearly so, under either condition, since the reduction of the ore is greater when a thick pulp passes to the mill. Classification demands about 3 to 1 feed dilution at the Independence mill. See Table 3A.

### Other Features of Chilean Mills

Early in our experience gear bolts on the bevel drive gave much trouble. Split gears were considered at that time a necessity from the standpoint of accessibility to the mill in case of gear breakage. Solid gears were tried and proved satisfactory in every respect, even though the mill must be torn down to replace. Fortunately, these replacements are seldom necessary.

Standards and protection plates demand attention due to the ordinary wear, which is small.

The distributer used has been evolved from a less satisfactory one. The feed pipes were 4 in. (10 cm.) in diameter, which allowed sticks of wood and slabs of rock to block them. This necessitated a shutdown and the punching of the distributer pipe. Aside from the delay the clutch suffered greatly. These pipes were increased to 6 in. (15 cm.) and gave very little trouble and then only when long sticks of wood were not picked and the ore was extremely slabby and coarse. To make negligible this possible delay, a new distributer with six 6-in. openings instead of three has been in use for the past 5 mo. and has practically eliminated the plugging of these pipes, and the consequent loss of running time.

The construction whereby the space between the die and screen is made as small as mechanically practical is essential for efficient work in the mill.

The clutches on the mills at the Independence are 100 hp. and too light for the work; this has caused loss of time that would otherwise be saved.

### Relation of Tonnage to Efficiency

Tonnage and efficiency are closely allied to the position of the splash with reference to the screen. The efficiency does not rise with tonnage to any great extent. Unless the splash against the screen is free, the efficiency will not be a maximum. With a high or new die it is easily possible to increase the tonnage to a point where the consumption of power is out of proportion, or the mechanical efficiency is poor. This is

due to part of the force of the splash being lost by hitting above the screen. A low die gives the best chance for the mill to maintain the greatest duty, although the increase over a comparatively new die is not great. Tons per horsepower are not greatly at variance whether 100 hp. is maintained or 120 hp.

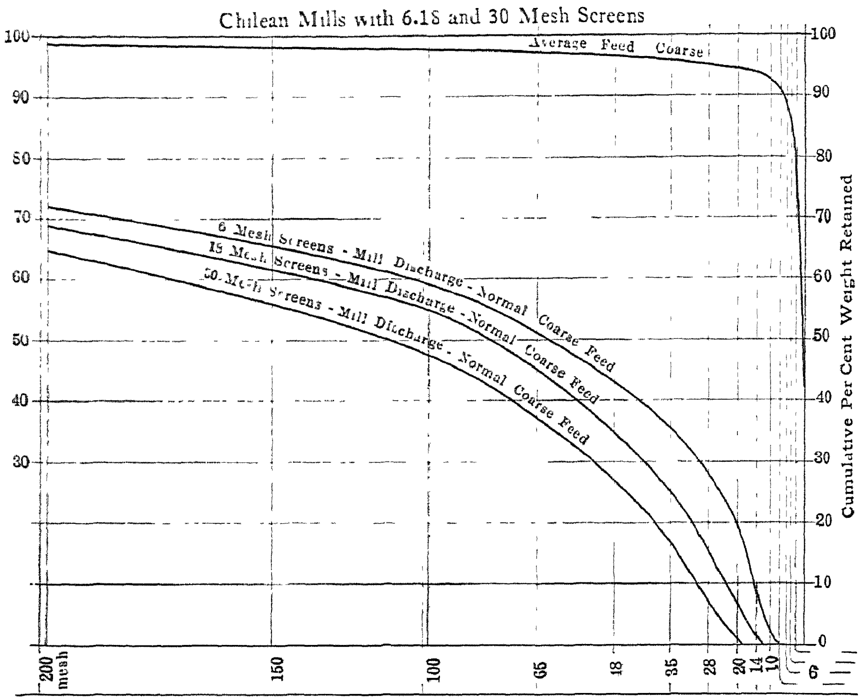



FIG. 2.—PLOT SHEET NO. 1

The grinding problem on Cripple Creek ores is much more serious than similar operations in most districts. The ore consists of phonolite, syenite, breccia, and some granite and resists greatly the grinding effort applied, whatever type of grinding machines is used. It may be termed a very hard and tough ore. I regret that we are not equipped to carry on parallel tests between Chilean mills and other types of machines, as it is admitted that this is the only absolutely satisfactory method. This paper therefore cannot compare two grinders directly. The object is to point out in detail the fundamental cause of failure of Chilean mills where parallel runs have been conducted.

Test runs have been made in order to compare the work done at the Independence plant and that done at Miami and Morenci by the Chilean mill, even though the ore is of an entirely different nature. The mills at the Independence are fed with a rolled product of which the largest

piece may be 3 to 4 in. in length, 2 or 3 in. wide and 1 in. thick. This shape of rock is the result of the slabby nature of certain of the ores—phonolites. The feed may therefore be rated as coarse for fine grinding work. The reduction made upon this feed with a Chilean mill varies greatly with the tonnage and screen used.

The Chilean mills at the Miami and Morenci plants, and in many other plants, have been subjected to a fine feed condition—less than ½ in. at Miami and 4-mesh at Morenci. A method has been devised at the

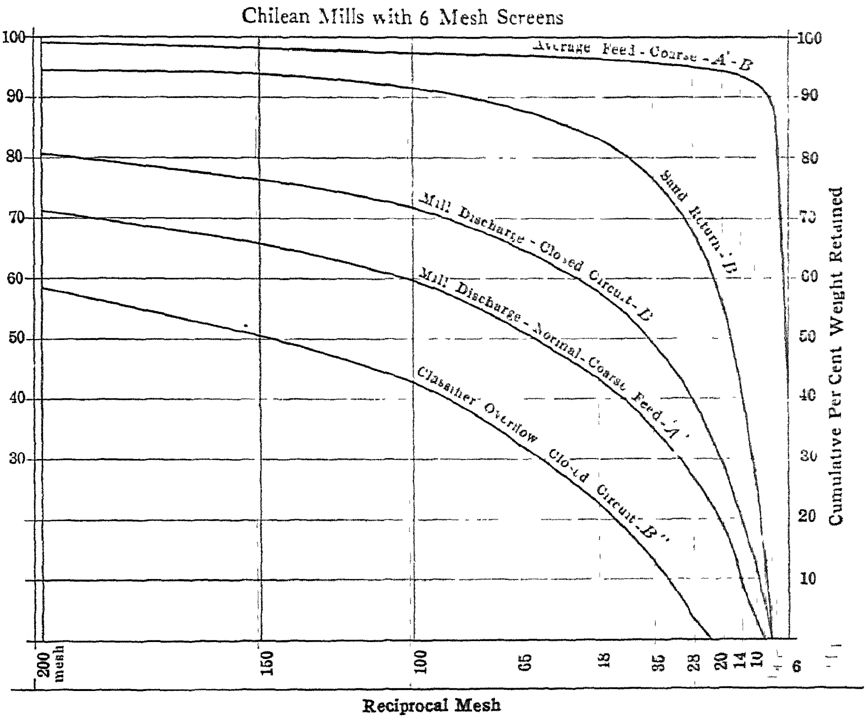



FIG. 3.—PLOT SHEET NO. 2.

Independence to approach the product used as a feed during the comparative tests at Miami and Morenci. This product was the classified sand from Chilean mills where 6-mesh screens were used and became the feed to one of our regular Chilean mills. Using this mill as a representative of the Miami and Morenci tests in competition with one of the regular mills with normal coarse feed, we have a comparative test on two Chilean mills with all conditions the same except feed size. In anticipation of a criticism I will state that it was impossible to get a screened product of original unground ore for this purpose. However, the return and mixed feed tests will indicate that the classified sands did not interfere with the good work of the mill, while small tests have proved to us that the sands were not appreciably harder than the average of the ore.

Under the fine-feed conditions, the power upon the mill was held at a maximum, but at that it was found impossible to get more than 80 per cent. of that normally used during coarse-feed conditions. As tonnage was increased, the mill gradually filled up above the screen; yet the power could not be raised. But in return for the limited power that may be forced into the mill for conversion into work, by reduction of the ore, that degree of reduction, combined with its tonnage, is not sufficient to bring the mechanical efficiency to within 80 per cent. of that where coarse

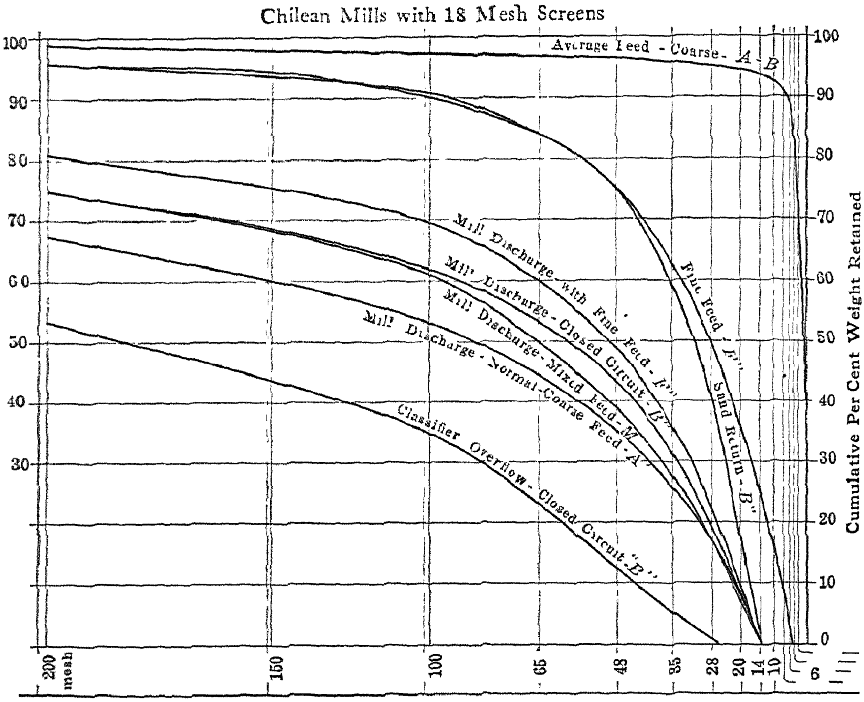



FIG. 4.—PLOT SHEET NO. 3.

feed is maintained. See Table 3. The Chilean mill is not a fine grinder, at least in being able to produce efficiently the extreme fines that a ball-mill is capable of giving. Hence, it is natural to expect its efficiency to fall when given a feed such that only extremely fine grinding will allow it to attain a respectable mechanical efficiency. Greater tonnage of this fine feed is its only hope, and that it cannot handle due to the lack of quick-enough reduction to pass out through 20- or 30-mesh screens.

If the mill is running normally with coarse feed at 110 hp. and is suddenly stopped, the mullers and die will be found to have between them about 2 in. (5 cm.) at the closest point of contact, of combined original feed of coarse mesh, and from that every size down to the fines

which have dropped back after having been rejected by the screen. This 2 in. at the edge of mullers and die becomes 5 in. of mixed sizes at the middle. With fine feed, the muller is closer to the die than it is under the coarse condition of feed, and no matter how great the tonnage the muller does not raise to the extent that it does where coarse feed covers the die. The mullers are plowing their way through the ore to be ground instead of passing on top of it. This characteristic of the muller when working on fine feed is, I believe, the underlying cause of the mechanical inefficiency of the Chilean mill when no coarse rock exists to give stability to the bed. No matter how heavy the load, a wagon will travel on top,

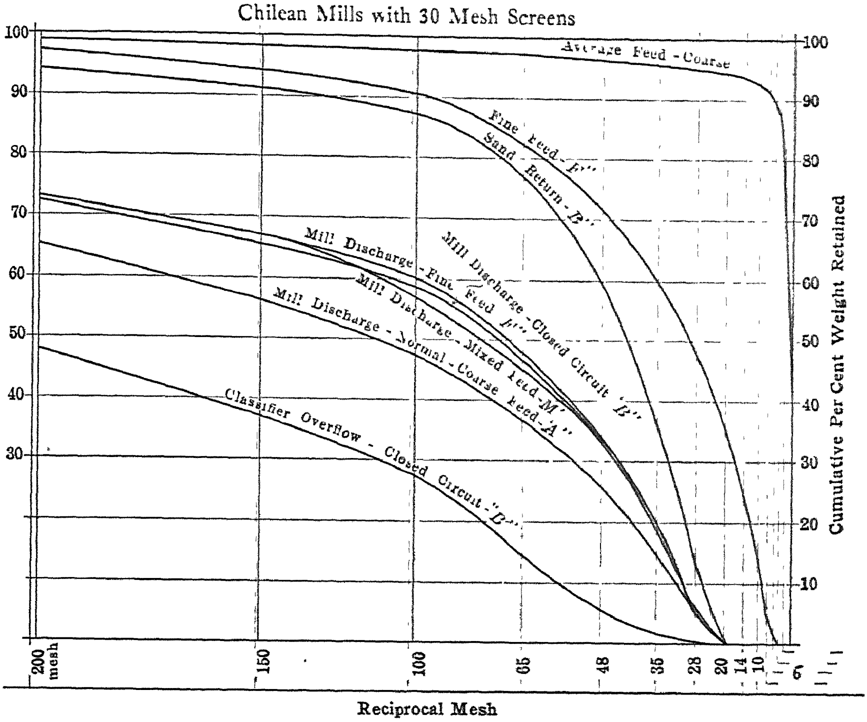


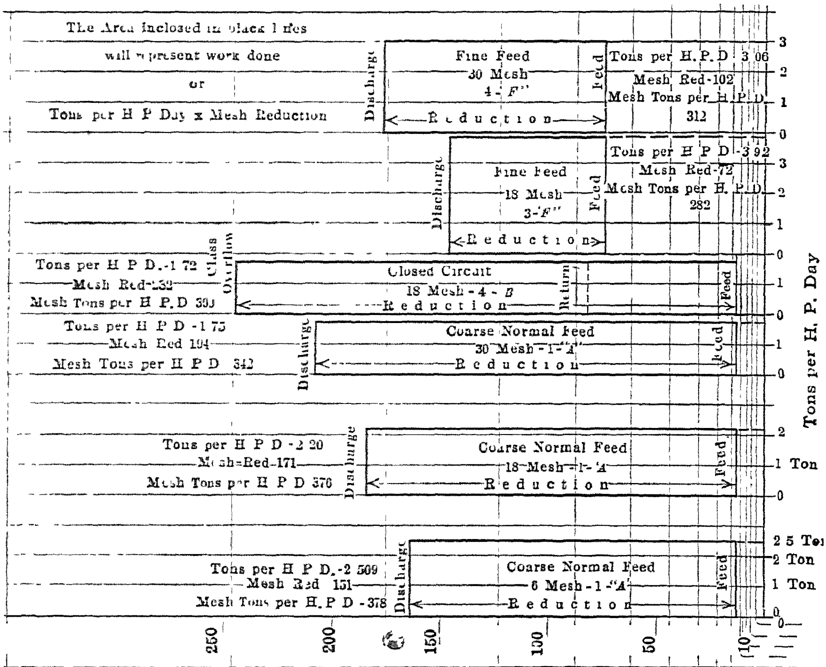

FIG. 5.—PLOT SHEET No. 4.

or nearly so, of a crushed rock roadbed, but when it comes to a muddy part of the highway it pushes the mud aside in an attempt to find a solid foundation.

The reduction of the ore in the mill is the result of two actions taking place as the muller makes its passage over the die. First, a crushing and grinding of the particles occurs as the heavy muller passes over the rock bed—the dead weight crushing and the friction due to the stationary die of steel on rock and rock on rock giving an attrition action that may be termed grinding. Second, the mullers revolve about the vertical axis of the mill and over the face of the die in a 6-ft. circle at the rate of

37 r.p.m. They are set so as to wear evenly in thickness on the inside and outside edges. Consequently, when revolving, the tendency is to continue in a straight line or at a tangent, rather than to take the path over the die. Forcing the muller to take this circular path will, therefore, cause an additional grinding or attrition action between steel and rock and rock and rock. This action in the mill may be likened to the hand work with a muller on a bucking board. We draw the muller back and

Performance of Chilean Mills on Portland Independence Ore, Victor, Colo. 1918–19




The relative mechanical efficiency is considered as proportional to work done or mesh-tons per horsepower day.

FIG. 6.—PLOT SHEET NO. 5.

forth in a straight line to crush, but on a hard ore we unconsciously find ourselves giving the muller a twist as we draw it along the board.

Many tests that have been made have indicated that there is an unexplored field of operation for the Chilean mill in which considerable reduction may be given the ore without interfering with its efficiency. The natural and simple installation, where a fine product is desired, is to allow the Chilean mill to discharge through a comparatively coarse screen and classify and regrind in cylindrical grinders for the finer work. This is at present the practice at the Independence mill using 6-mesh screens.

TABLE 1.—*Average of Long-time Runs under Normal Conditions. Refer to Fig. 2.*

| Size | | Chilean-mill Screen Discharge | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | A 4 Mills 6-mesh Screens 37 r p m | | B 1 Mill 18-mesh Screens 37 r p m | | C 1 Mill 30-mesh Screens 37 r p m | | D 1 Mill 6-mesh Screens Slow Speed 29 r p m | |
| Mesh | Inch | Per Cent. | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent |
| Plus 10 | 0 065 | 2 2 | 2 2 | .. | ... | .... | . | 1 0 | 1 0 |
| Plus 14 | 0 046 | 5 6 | 7 8 | 0 8 | 0 8 | ... | ... | 6 7 | 7 7 |
| Plus 20 | 0 0328 | 11 7 | 19 5 | 5 1 | 5 9 | 1 1 | 1 1 | 11 3 | 19 0 |
| Plus 28 | 0 0232 | 8 5 | 28 0 | 9 1 | 15 0 | 5 1 | 6 2 | 8 0 | 27 0 |
| Plus 35 | 0 0164 | 7 5 | 35 5 | 9 8 | 24 8 | 9 0 | 15 2 | 8 1 | 35 1 |
| Plus 48 | 0.0116 | 7 4 | 42 9 | 10 2 | 35 0 | 11 8 | 27 0 | 7 9 | 43 0 |
| Plus 65 | 0.0082 | 8 3 | 51 2 | 10 0 | 45 0 | 10.0 | 37 0 | 8 5 | 51 5 |
| Plus 100 | 0.0058 | 8 3 | 59 5 | 10 2 | 55 2 | 10 9 | 47 9 | 8 0 | 59.5 |
| Plus 150 | 0.0041 | 6 1 | 65 6 | 6 3 | 61 5 | 8 1 | 56 0 | 5 9 | 65.4 |
| Plus 200 | 0 0029 | 6.2 | 71 8 | 7 5 | 69 0 | 8 5 | 64.5 | 5 9 | 71 3 |
| Through 200 | 0 0029 | 28 2 | . | 31 0 | . | 35 5 | .... | 28 7 | |
| Coarse feed, tons per 24 | | 282 0 | . . | 256 0 | ... | 175 0 | ... | 186 0 | |
| Power, hp . . . . . | | 112.8 | .. | 114 2 | ... | 100 0 | . . | 93.0 | |
| Tons per hp-day .... | | 2.509 | . | 2 20 | .. | 1 75 | . . . | 2 00 | |
| Chilean-mill discharge, in average mesh . | | 164 0 | . . | 184 0 | | 207 0 | | 166 0 | |
| Chilean-mill feed, in average mesh . . | | 13.0 | .... | 13 0 | | 13 0 | . | 13 0 | |
| Reduction, mesh . | | 151 0 | ... | 171 0 | . | 194 0 | | 153 0 | |
| Mesh-tons per hp-day | | 378 0 | ... | 376 0 | | 342 0 | | 306 0 | |

In order to gain information as to the possibility of a finished product in the one operation and the corresponding mechanical efficiency, a pump returned the sands from a Chilean-mill classifier, normally discharging to the ball-mills, to the Chilean-mill feed spout; that is, a closed circuit was maintained. The average results are shown in Tables 4–5. They would prove that the Chilean mill is capable of producing a product that is practically a finished one for the average plant, maintaining an efficiency comparable with its maximum under the usual method of operation. In referring to the various mesh tests and curves, it will be interesting to note the effect on the mesh of the mill discharge when handling this large tonnage, which includes the return. An original tonnage of 220 to 240 per 24 hr. where 6-mesh screens are used, and 185 tons where 18-mesh screens are used is a fair average. The corresponding reduction may be noted in the tables.

These tests and resulting tabulations will indicate the possibility of the Chilean mill in the field of regrinding, where, however, a coarse original feed is maintained in the regrinding mill. Whether the mill is used as a regrinder in series with another or in closed circuit, the out-

TABLE 2.—*Special Tests. Six-mesh Screens on Chilean Mills. Refer to Fig. 3. All Mills at 37 R p.m.*

| Size | | A Normal Chilean-mill Discharge | | B Closed-circuit Chilean-mill Discharge | | B Closed-circuit Sand Return | | B Closed-circuit • Classifier Overflow | |
|---|---|---|---|---|---|---|---|---|---|
| Mesh | Inch | Per Cent | Cum Per Cent | Per Cent. | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent |
| Plus 10 | 0 065 | 3 5 | 3 5 | 12 5 | 12 5 | 28 0 | 28.0 | | |
| Plus 14 | 0.046 | 6 1 | 9.6 | 7 5 | 20 0 | 14 5 | 42 5 | | |
| Plus 20 | 0.0328 | 9 7 | 19 3 | 9 3 | 29.3 | 13 2 | 55 7 | | |
| Plus 28 | 0 0232 | 7 5 | 26 8 | 10 7 | 40 0 | 11 8 | 67 5 | 3 5 | 3 5 |
| Plus 35 | 0 0164 | 8 2 | 35 0 | 9 3 | 49 3 | 8 5 | 76 0 | 9 5 | 13 0 |
| Plus 48 | 0 0116 | 8 2 | 43 2 | 8 2 | 57 5 | 7 0 | 83 0 | 9 5 | 22 5 |
| Plus 65 | 0 0082 | 8 1 | 51 3 | 7 3 | 64 8 | 4 8 | 87 8 | 9 8 | 32 3 |
| Plus 100 | 0 0058 | 8 2 | 59 5 | 6 8 | 71 6 | 3 7 | 91 5 | 10 3 | 42 6 |
| Plus 150 | 0 0041 | 6 0 | 65 5 | 4.4 | 76 0 | 2 0 | 93 5 | 7 7 | 50 3 |
| Plus 200 | 0 0029 | 5 3 | 70 8 | 4 1 | 80 1 | 0 5 | 94 0 | 7 7 | 58 0 |
| Through 200 | 0 0029 | 29 2 | | 19 9 | ... | 6 0 | .... | 42 0 | |
| Coarse feed, tons .. | | 301 0 | . . | 216 0 | | | | | |
| Fine return, tons . | | ..... | .... | 302 0 | .... | 302 0 | | | |
| Total screen discharge, tons ....... .... | | 301 0 | ... | 518 0 | | | | | |
| Power, hp. .. .... .. | | 111 0 | . . | 108 0 | | | | | |
| Tons per hp-day . . | | 2 70 | .... | 2 0 | | | | | |
| Screen discharge, mesh | | 166 0 | . | 130 0 | | | | | |
| Classifier overflow mesh | | .. | . . | 222 0 | ... | .. | .... | 222 0 | |
| Coarse feed, mesh | | 13 0 | | 13 0 | | | | | |
| Sand return, mesh .. | | . | | 60 0 | .. | 60 0 | | | |
| Combined feed, mesh | | 13 0 | | 40 4 | | | | | |
| Reduction, mesh. .. | | 153 0 | . | 209 0 | | | | | |
| Mesh-tons per hp-day | | 413 0 | . | 418 0 | | | | | |

standing feature is the gaining of a high efficiency while doing this class of work. In the case of regrinding of sands in series with another mill, the product or screen discharge is coarser than where original or coarse feed alone is applied, but the increased tonnage maintains the efficiency at a good figure. In the case of closed circuit, while the screen discharge is comparatively coarse, the product or weir overflow is sufficiently fine to offset the reduced tonnage.

These tests, in which sands are a considerable part of the feed, would seem to be in opposition to the theories and statements in the earlier part of this article regarding the necessity of a coarse and the inadvisability of a fine feed. Apparently, the extremely coarse material gives such stability to the mass as to force the heavy mullers to pass over, instead of through, with every assurance that the interstices are well filled with granular fines ready for attrition action. Reference to Tables 6–7 will show that where the sand return is heavy or is the predominating portion of the feed, the mechanical efficiency falls.

I have attempted to take up all the features that have led to the present position of the Chilean mill in the scale of popularity. As stated

TABLE 3.—*Performance of Chilean Mill with Fine Feed. Refer to Figs. 4–5. All Mills at 37 R.p.m.*

| | Chilean Mill with 18-mesh Screens A "F" | | | | Chilean Mill with 30-mesh Screens B "F" | | | |
|---|---|---|---|---|---|---|---|---|
| | Feed | | Discharge | | Feed | | Discharge | |
| | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent |
| Plus 10 | 16.3 | 16.3 | | | 15.8 | 15.8 | | |
| Plus 14 | 8.5 | 24.8 | 1.4 | 1.4 | 9.2 | 25.0 | | |
| Plus 20 | 10.3 | 35.1 | 7.1 | 8.5 | 10.1 | 35.1 | | |
| Plus 28 | 14.6 | 49.7 | 14.4 | 22.9 | 12.6 | 47.7 | 5.5 | 5.5 |
| Plus 35 | 12.2 | 61.9 | 12.6 | 35.5 | 12.0 | 59.7 | 15.0 | 20.0 |
| Plus 48 | 13.2 | 75.1 | 13.3 | 48.8 | 11.7 | 71.4 | 14.1 | 34.1 |
| Plus 65 | 8.9 | 84.0 | 11.0 | 59.8 | 10.2 | 81.6 | 13.6 | 47.7 |
| Plus 100 | 6.6 | 90.6 | 10.0 | 69.8 | 9.0 | 90.6 | 12.5 | 60.2 |
| Plus 150 | 2.7 | 93.3 | 5.4 | 75.2 | 3.4 | 94.0 | 6.7 | 66.9 |
| Plus 200 | 2.2 | 95.5 | 5.2 | 80.4 | 2.9 | 96.9 | 6.2 | 73.1 |
| Through 200 | 4.5 | .... | 19.6 | . | 3.1 | .. | 27.0 | |
| Ratio pulp | | | 1.3 to 1 | | | | 2.0 to 1 | |
| Coarse feed, tons | | | None | | | | None | |
| Fine feed, tons | | | 306.0 | | | | 252.0 | |
| Power, hp | | | 78.0 | | | | 82.4 | |
| Tons per hp.-day | | | 3.92 | | | | 3.06 | |
| Screen, discharge mesh | | | 146.0 | | | | 176.0 | |
| Feed, mesh | | | 74.0 | | | | 74.0 | |
| Reduction, mesh | | | 72.0 | | | | 102.0 | |
| Mesh-tons per hp.-day | | | 282.0 | | | | 312.9 | |

Compare with Table 4, Coarse feed Mesh-tons per horsepower day, 372–390–406.

earlier, most of the criticism revolves about its relative mechanical efficiency—the work done in return for the dollars expended in power. The only method that appeared possible to adopt, in order to prove that a Chilean mill is efficient, is the one I have adopted in order to gain a comparison between a Chilean mill in Arizona and one at the Independence plant—that at Miami working on fine feed and at Independence on coarse feed. At the Independence plant, the fine feed deprives the Chilean mill of at least 20 per cent. of its efficiency. At Miami and Morenci all conclusions were based upon fine-feed conditions and the Chilean mills were discarded. In referring to relative feeds at Miami and Morenci and the fine feed at Independence, it may be noted that the feed at Miami is coarser than the prepared feed at Independence, yet the largest piece in the Miami feed is not as large as 65 per cent. of the normal coarse feed at Independence. Undoubtedly the Miami feed of minus ½ in. would allow the Chilean to do better work than that at Morenci on minus 4-mesh; yet the same general criticism of all fine feeds involved at Miami, Morenci, and Independence may be made—that the coarse bed necessary to hold these fines was lacking. As for other criticisms, reference may be made to Tables 9 to 12 for costs and loss

TABLE 3A

EFFECT OF FEED-PULP DILUTION TO MILL AVERAGE OF TESTS WITH 18-MESH SCREENS REGULAR COARSE FEED

| | Discharge From Mills A: Per Cent | A: Cum. Per Cent | B: Per Cent | B: Cum Per Cent |
|---|---|---|---|---|
| Plus 35 | 23 8 | 23 8 | 27 2 | 27 2 |
| Plus 48 | 9.0 | 32 8 | 9 9 | 37 1 |
| Plus 65 | 8.1 | 40.9 | 8 2 | 45 3 |
| Plus 100 | 9 4 | 50 3 | 9 4 | 54 7 |
| Plus 150 | 6 4 | 56 7 | 6 2 | 60 9 |
| Plus 200 | 7 2 | 63 9 | 6 9 | 67 8 |
| Through 200 | 36 1 | ... | 32 2 | |
| Ratio pulp | 1.68 to 1 | ... | 3 66 to 1 | |
| Input, tons | 215 0 | .. | 241 0 | |
| Power, hp | 110 4 | ... | 110 5 | |
| Tons per hp -day | 1 99 | ... | 2 18 | |
| Screen discharge, mesh | 199 5 | .. | 182 5 | |
| Coarse feed, mesh | 13 0 | .. | 13 0 | |
| Reduction, mesh | 182 5 | ... | 169 5 | |
| Mesh-tons per hp -day | 363 0 | . | 370 0 | |

AVERAGE OF ALL FEED SAMPLES TO CHILEAN MILLS OVER LONG PERIOD USED IN ALL CALCULATIONS OF COARSE FEED TO MILLS

| | Per Cent. | Cum. Per Cent. | | Per Cent. | Cum. Per Cent. |
|---|---|---|---|---|---|
| Plus 2 in | 12 2 | 12 2 | Plus 20 mesh | 0 9 | 94 3 |
| Plus 1½ in. | 14 0 | 26 2 | Plus 28 mesh | 0 9 | 95 2 |
| Plus 1 in. | 15 8 | 42 0 | Plus 35 mesh | 0 7 | 95 9 |
| Plus ¾ in | 12.8 | 54 8 | Plus 48 mesh | 0.7 | 96 6 |
| Plus ½ in. | 12 0 | 66 8 | Plus 65 mesh | 0 5 | 97.1 |
| Plus 4 mesh | 17 2 | 84.0 | Plus 100 mesh | 0 6 | 97 7 |
| Plus 6 mesh | 5 5 | 89.5 | Plus 150 mesh | 0 4 | 98 1 |
| Plus 10 mesh | 3 0 | 92.5 | Plus 200 mesh | 0 4 | 98 5 |
| Plus 14 mesh | 0 9 | 93 4 | Through 200 mesh | 1.6 | |

of running time. These items are hard to compare on machines operating in different plants on different ores, but I do feel that the criticism made against the Chilean mill in these lines, being based on conditions that involved inferior mechanical efficiency, would necessarily be unfavorable as to costs and delays as well.

The production of excess slimes for which the Chilean mill has been at times condemned may be decidedly improved by proper screens and classification.

Before referring to the plots and the conclusions deduced from them, I wish to state that a great many test runs have been condensed into a comparatively small number of averages. From these records certain conclusions have been drawn. I ask those who do not approve of Rittinger's law of grinding, and Gates' application to plots to take the data or log of tests and from these draw their conclusions. These conclusions, I believe, will bear out in general the statements herein made. The main object and conclusions for which this paper is written do not depend on whether Rittinger or Kick and Stadler is correct.

TABLE 4.—*Averages of Special Tests. 18-mesh Screens on Chilean Mills. Refer to Fig. 4. All Mills at 37 R.p.m.*

| Size | | Normal | | Closed Circuit | | | | | | Mixed Feed | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Chilean-mill Discharge | | Chilean-mill Discharge | | Sand Return | | Classifier Overflow | | Mill Discharge | |
| Mesh | Inch | A | | B | | | | | | C"M" | |
| | | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum. Per Cent | Per Cent | Cum Per Cent. | Per Cent | Cum Per Cent |
| Plus 14 | 0 046 | . | | | | 2 0 | 2 0 | . | .. | 1 4 | 1 4 |
| Plus 20 | 0 0328 | 8 0 | 8 0 | 8 0 | 8 0 | 15 3 | 17.3 | . | ... | 5.7 | 7 1 |
| Plus 28 | 0 0232 | 8 9 | 16 9 | 11 2 | 19 2 | 23 5 | 40 8 | 0 8 | 0 8 | 10 9 | 18 0 |
| Plus 35 | 0 0164 | 8 8 | 25 7 | 12 1 | 31 3 | 18 7 | 59 5 | 4 2 | 5 0 | 9.5 | 27 5 |
| Plus 48 | 0 0116 | 9 3 | 35 0 | 11 9 | 43 2 | 15 5 | 75.0 | 7 4 | 12 4 | 11 4 | 38 9 |
| Plus 65 | 0 0082 | 9 0 | 44 0 | 10 2 | 53 0 | 9 0 | 84 0 | 10 6 | 23 0 | 10 4 | 49.3 |
| Plus 100 | 0 0058 | 8 9 | 52 9 | 8 8 | 61 8 | 6 0 | 90 0 | 11 8 | 34 8 | 12 2 | 61.5 |
| Plus 150 | 0 0041 | 7 1 | 60 0 | 6 9 | 68 7 | 4.3 | 94 3 | 8 9 | 43 7 | 6 8 | 68 3 |
| Plus 200 | 0 0029 | 7 0 | 67 0 | 5 9 | 74 6 | 0 7 | 95 0 | 9 3 | 53 0 | 6.7 | 75.0 |
| Through 200 | 0 0029 | 33.0 | .. | 25 4 | | 5 0 | . | 47 0 | . . | 24 6 | . . |
| Coarse feed, tons .. . . | | 225 0 | . | 184 0 | . | . | | .. . | ... . | 193 0 | |
| Fine return, tons . | | . . | . . | 190 0 | | 190 0 | | . . . | . .. | 154 0 | |
| Total screen discharge, tons........ . . . | | 225 0 | | 374 0 | | . | | | .. | 347 0 | |
| Power, hp.... .. . | | 105 7 | . | 107 0 | | . . | .. | . | . . | 104 5 | |
| Tons per hp -day.. . | | 2.13 | .. | 1 72 | . | | | | . | 3 32 | |
| Screen discharge, mesh | | 187 7 | . | 162 2 | . . | . | . | . | . | 167 0 | |
| Classifier overflow, mesh | | ... | | 245 0 | . | . | | 245 0 | | | |
| Coarse feed, mesh . | | 13 0 | | 13 0 | . . | . . | .. | . .. | . | 13 0 | |
| Sand return, mesh.. . | | . | . | 80 6 | | 80 6 | . | .. | . . | 84 0 | |
| Combined feed, mesh | | 13 0 | | 48 0 | | .. | | .... | ... | 44 5 | |
| Reduction, mesh . . . | | 174 7 | | 232 0 | . | | . . | ..... | .... | 122 5 | |
| Mesh-tons per hp -day | | 372 0 | . | 399.0 | . | . | . . | ... | .... | 406 0 | |

The detailed records contained in tables may be condensed to the following: The Chilean mill is capable of reducing the hard, refractory, Cripple Creek ore from minus 2½ in. in the following degrees: with 6-mesh screens, 275 to 300 tons per 24 hr. to 28.5 per cent. through 200-mesh at the rate of 2.5 tons per horsepower-day; with 18-mesh screens, 240 tons to 32 per cent. through 200-mesh at the rate of 2.2 tons per horsepower-day; with 30-mesh screens, 175 tons to 35 per cent. through 200-mesh at the rate of 1.75 tons per horsepower-day.

*In Closed Circuit.*—With 6-mesh screens, 200 tons to 42 per cent. through 200-mesh at the rate of 1.86 tons per horsepower-day (these figures have been reduced from actual test averages, to make the statement absolutely conservative from the standpoint of long-time runs, where results were not as good as test runs); with 18-mesh screens, 185 tons to 47 per cent. through 200-mesh at the rate of 1.72 tons per horsepower-day; with 30-mesh screens, 130 tons to 52 per cent. through 200-mesh at the rate of 1.31 tons per horsepower-day.

TABLE 5.—*Averages of Special Tests. 30-mesh Screens on Chilean Mills. Refer to Fig 5 All Mills at 37 R.p m*

| Size | | Normal | | Closed Circuit | | | | | | Mixed Feed | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Chilean-mill Discharge | | Chilean-mill Discharge | | Sand Return | | Classifier Overflow | | Fine Feed | | Chilean-mill Discharge | |
| | | A | | B | | | | | | C"M" | | | |
| Mesh | Inch | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent |
| Plus 10 | 0.065 | . | ... | . . | .. | . | | | . | 15.3 | 15.3 | | |
| Plus 14 | 0.046 | ... | . | | | | | .. | | 9.0 | 24.3 | | |
| Plus 20 | 0.0328 | 0.5 | 0.5 | 0.2 | 0.2 | 0.7 | 0.7 | .. | | 10.7 | 35.0 | | |
| Plus 28 | 0.0232 | 3.3 | 4.8 | 5.9 | 6.1 | 12.6 | 13.3 | .. | . . | 12.5 | 47.5 | 5.0 | 5.0 |
| Plus 35 | 0.0164 | 10.0 | 14.8 | 12.4 | 18.5 | 21.9 | 35.2 | 1.1 | 1.1 | 12.0 | 59.5 | 15.0 | 20.0 |
| Plus 48 | 0.0116 | 11.0 | 25.8 | 13.6 | 34.0 | 24.4 | 59.6 | 4.5 | 5.6 | 11.6 | 71.1 | 13.3 | 33.3 |
| Plus 65 | 0.0082 | 10.6 | 36.4 | 12.5 | 46.5 | 16.6 | 76.2 | 9.0 | 14.6 | 10.6 | 81.7 | 11.7 | 45.0 |
| Plus 100 | 0.0058 | 10.6 | 47.0 | 12.0 | 58.6 | 10.8 | 87.0 | 12.8 | 27.4 | 8.5 | 90.2 | 12.0 | 57.0 |
| Plus 150 | 0.0041 | 9.0 | 56.0 | 6.8 | 65.4 | 4.1 | 91.1 | 9.6 | 37.0 | 3.8 | 94.0 | 9.8 | 66.8 |
| Plus 200 | 0.0029 | 9.0 | 65.0 | 6.9 | 72.4 | 2.8 | 93.9 | 10.9 | 47.9 | 2.9 | 96.9 | 6.2 | 73.0 |
| Through 200 | 0.0029 | 35.0 | . . . | 27.6 | | 6.1 | . | 52.1 | . | 3.1 | . | 27.0 | |
| Coarse feed to Chilean mill, tons | | 184.0 | | . 131.0 | | . | . | | . . | ... | ... | 94.0 | |
| Fine input, tons | | . | | 147.0 | | 147.0 | | ... | . . | ... | .. | 188.0 | |
| Total screen discharge, tons | | 184.0 | . | 278.0 | | | .. | . . | .... | .... | ... | 282.0 | |
| Power, hp. | | 104.5 | .. | 100.0 | . | . | | .. | . . | . . | ... | 101.8 | |
| Tons per hp-day | | 1.76 | .. | 1.31 | | ... | | . . | . | . . | ... | 2.77 | |
| Screen discharge, mesh | | 203.0 | .. | 180.0 | . | . | | | | | | 181.4 | |
| Classifier overflow, mesh | | . | | 263.5 | . | .. | . | 263.5 | | | | | |
| Coarse feed, mesh | | 13.0 | .. | 13.0 | ... | .. | | ... | . | . . | ... | 13.0 | |
| Sand return, mesh | | . . | .. | 105.5 | .. | 105.5 | . | | . . | . . | ... | 77.0 | |
| Combined feed, mesh | | . | ... | 62.0 | . | . | | . . | . | . .. | .. | 55.8 | |
| Reduction, mesh | | 190.0 | . . | 250.5 | . | | | | . | . . | . . | 125.6 | |
| Mesh-tons per hp-day | | 332.0 | . | 328.0 | | .. | .. | . . | .. | | ... | 348.0 | |

The results in Table 5 indicate that a Chilean mill is not as efficient with 30-mesh screens as with the coarser mesh screens

The Chilean mill is capable of taking the classified sands from a 6-mesh screen and grinding them with minus 2½-in. original feed at the same time maintaining a good efficiency. The Chilean mill is not capable of maintaining a good efficiency on a feed of fine material as minus 6-mesh sands unless a coarse feed containing 1½- to 2½-in. size material is fed with the fines. Compare results, Table 4C"M" and Table 3.

The costs, including operation, repairs, replacements, and power, are not out of proportion for the class of work done. The loss of running time due to repairs and replacements on the mills is not excessive. While the Chilean mill is not capable of furnishing a finished product of, say, 5 per cent. on 48-mesh as a screen discharge, efficiently, there is a very promising field of experiment with every chance of success in obtaining this product, and with mechanical efficiency.

TABLE 6.—*Special Tests to Indicate the Range of Final Discharge from Mill in Closed Circuit with Light and Heavy Return. 6-mesh Screens on Mills. All Mills at 37 R.p.m.*

| Mesh | A | | | | B | | | |
|---|---|---|---|---|---|---|---|---|
| | Chilean-mill Screen Discharge | | Classifier Overflow | | Chilean-mill Screen Discharge | | Classifier Overflow | |
| | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent. | Per Cent | Cum Per Cent | Per Cent | Cum Per Cent |
| Plus 10 | 13.0 | 13.0 | ... | .... | 14 7 | 14 7 | | |
| Plus 14 | 7.3 | 20 3 | .. | ... | 7 1 | 21 8 | | |
| Plus 20 | 7.7 | 28 0 | ... | .... | 7 8 | 29.6 | | |
| Plus 28 | 9.6 | 37 6 | . .. | .. . | 10 7 | 40 3 | | |
| Plus 35 | 7 2 | 44 8 | 16.2 | 16 2 | 8 7 | 49 0 | 9 9 | 9 9 |
| Plus 48 | 7 9 | 52.7 | 9.6 | 25 8 | 8 8 | 57.8 | 8.4 | 18.3 |
| Plus 65 | 6.6 | 59 3 | 8.9 | 34 7 | 6.7 | 64.5 | 9 2 | 27.5 |
| Plus 100 | 7.5 | 66.8 | 10 6 | 45 3 | 7 0 | 71.5 | 11 2 | 38 7 |
| Plus 150 | 4 6 | 71 4 | 7 1 | 52 4 | 4 0 | 75.5 | 7.9 | 46.6 |
| Plus 200 | 5 4 | 76.8 | 8 0 | 60 4 | 4.5 | 80.0 | 8.9 | 55 5 |
| Through 200 | 23 2 | .... | 39 6 | .. . | 20 0 | .... | 44 5 | |
| Coarse feed, tons ..... | | 251 0 | ... | . . | .. . | 198 0 | | |
| Fine return, tons ..... | | 217 0 | ... | .... | .... | 310.0 | | |
| Total screen discharge, tons ... . ......... | | 468 0 | .... | .... | .... | 508.0 | | |
| Power, hp.............. | | 117 5 | .... | .. . | .... | 106 0 | | |
| Tons, per hp -day. ..... | | 2.14 | .... | .. . | .... | 1 87 | | |
| Screen discharge, mesh. . | | 141.0 | .... | .... | .... | 129 0 | | |
| Classifier overflow, mesh | | 212.0 | . .. | .... | .. . | 232.0 | | |
| Coarse feed, mesh ..... | | 13.0 | .... | .... | . . | 13.0 | | |
| Sand return, mesh...... | | 59 0 | .... | .... | .... | 63 0 | | |
| Combined feed to mill, mesh .... .... . ..... | | 35 0 | .... | .... | .... | 44.0 | | |
| Reduction on original, mesh................. | | 199.0 | .... | .... | .... | 219.0 | | |
| Mesh-tons per hp -day .. | | 426.0 | .... | .... | .... | 410.0 | | |

I have not intended to convey the idea that because Chilean mills are not universally used the world is wrong, nor do I pretend to intimate that any particular installation of other type of grinders is not justified. I am in no position to attempt to judge these points. But I do want to feel that the long-standing and continued attack upon the Chilean mill has at last brought forth a defence, which I hope will be deemed a worthy one. If the Chilean mill is as bad as it is painted, the years of grinding operations on Cripple Creek ores have been years of large, unnecessary expense to the Portland Gold Mining Co. in inefficient work, and we do not feel that this is so; rather, on the other hand, that the necessary grinding on this hard, refractory ore has been done with the least expense possible. Finally, I contend that the Chilean mill has just as much right to remain a member in good standing of the grinding-machinery association as any of those machines that are today making the wonderful strides they are in popularity.

TABLE 7.—*Special Tests to Indicate the Range of Final Discharge from Mill in Closed Circuit with Light and Heavy Return. 18-mesh Screens on Mills. All Mills at 37 R.p.m.*

| Mesh | A | | | | B | | | |
|---|---|---|---|---|---|---|---|---|
| | Chilean-mill Screen Discharge | | Classifier Overflow | | Chilean-mill Screen Discharge | | Classifier Overflow | |
| | Per Cent | Cum. Per Cent | Per Cent. | Cum. Per Cent. | Per Cent | Cum Per Cent. | Per Cent. | Cum Per Cent. |
| Plus 14 | 0 8 | 0 8 | .... | ... | 0 6 | 0 6 | | |
| Plus 20 | 6.6 | 7.4 | .... | .... | 6 6 | 7.2 | | |
| Plus 28 | 11 3 | 18 7 | ... | ... | 13 5 | 20.7 | | |
| Plus 35 | 9.4 | 28.1 | 8.8 | 8 8 | 12 1 | 32.8 | | |
| Plus 48 | 10.2 | 38 3 | 8 4 | 17 2 | 14.3 | 47.1 | 2 7 | 2 7 |
| Plus 65 | 8 3 | 46 6 | 9 0 | 26 2 | 11 2 | 58.3 | 7.1 | 9 8 |
| Plus 100 | 9.2 | 55 8 | 11 3 | 37 5 | 10 3 | 68 6 | 12 8 | 22.6 |
| Plus 150 | 6 0 | 61 8 | 7 9 | 45 4 | 5 6 | 74 2 | 10 1 | 32 7 |
| Plus 200 | 6.8 | 68 6 | 9 5 | 54 9 | 5 4 | 79 6 | 11 6 | 44 3 |
| Through 200 | 31.4 | ... | 45.1 | .. | 20.4 | .... | 55 7 | |
| Coarse feed, tons | | 199 0 | . | . . | .. | 165 0 | | |
| Fine return, tons | | 106 0 | . . | ... | ... | 281.0 | | |
| Total screen discharge, tons | | 305 0 | ... | ... | . . | 446.0 | | |
| Power, hp | | 104.0 | . | ... | ... | 113.3 | | |
| Tons per hp.-day | | 1.90 | .... | .... | .... | 1.45 | | |
| Screen discharge, mesh | | 180 0 | . | ... | .. | 147.0 | | |
| Classifier overflow, mesh | | 235 0 | . | . . | .. | 273 0 | | |
| Coarse feed, mesh | | 13 0 | . | . . | .. | 13.0 | | |
| Sand return, mesh | | 84 5 | ... | ... | . | 85.0 | | |
| Combined feed to mill, mesh | | 37 6 | . | ... | .... | 58 0 | | |
| Reduction on original, mesh | | 222.0 | ... | ... | .... | 260.0 | | |
| Mesh-tons per hp.-day | | 421.0 | .... | .... | ... | 377.0 | | |

TABLE 8.—*Performance of Chilean Mill with Low Power Consumption. 18-mesh Screens. All Mills at 37 R.p.m.*

| Mesh | Chilean-mill Discharge | |
|---|---|---|
| | Per Cent. | Cum. Per Cent |
| Plus 14 | 1.0 | 1.0 |
| Plus 20 | 4.8 | 5 8 |
| Plus 28 | 8.6 | 14.4 |
| Plus 35 | 7.6 | 22 0 |
| Plus 48 | 9.6 | 31.6 |
| Plus 65 | 8.6 | 40 2 |
| Plus 100 | 10.1 | 50.3 |
| Plus 150 | 7.2 | 57.5 |
| Plus 200 | 7.5 | 65.0 |
| Through 200 | 35 0 | |

| | |
|---|---|
| Ratio pulp | 2.10 to 1 |
| Coarse feed, tons | 171 |
| Fine feed, tons | None |
| Power, hp | 84.1 |
| Tons per hp.-day | 2.04 |
| Screen discharge, mesh | 195 |
| Feed, mesh | 13 |
| Reduction, mesh | 182 |
| Mesh-tons per hp.-day | 370 |

TABLE 9.—*Fine-grinding Operation Costs at Independence Mill, for 1918: Tonnage, 441,435*

| | Chilean Mills and Clutches, Cost | Classifiers, Cost | Ball-mills, Cost | |
|---|---|---|---|---|
| Labor, operating | $0.0120 | $0.0030 | $0 0050 | |
| Repairs, general | 0 0089 | 0 0006 | 0.0015 | |
| Steel | 0.0022 | | 0 0015 | |
| Supplies, general | 0 0192 | 0 0008 | 0.0048 | |
| Oils and grease | 0 0014 | 0 0002 | 0 0010 | |
| Steel | 0 0674 | | 0.0457 | |
| Screens | 0 0088 | | | |
| Miscellaneous | 0 0007 | | 0.0004 | |
| Power, electric | 0.0764 | 0.0010 | 0.0445 | |
| Total | $0.1970 | $0.0056 | $0.1044 | $0 3070 |
| Power feeders to mills | | | | 0.0012 |
| Total fine grinding | | | | $0.3082 |

Overhead of $0.0065 not included in above.

TABLE 10.—*Delays in Fine-grinding Department of Independence Mill in* 1918

| | Hours | Per Cent., Running Time. | Hours per Day, Lost[a] |
|---|---|---|---|
| Dies | 296 9 | 0.594 | 0.855 |
| Mullers | 175.3 | 0.351 | 0.505 |
| Total steel | 472.2 | 0.945 | 1.360 |
| Screens | 198.2 | 0.396 | 0.571 |
| Other mill repairs | 392.4 | 0.785 | 1.131 |
| Total repairs and replacement | 1062.8 | 2.126 | 3 062 |
| Oiling | 694 0 | 1.389 | 2.000 |
| Total delays | 1756.8 | 3.515 | 5.062 |
| Delays per mill | 304.2 | 3.515 | 0.879 |
| Clutch and clutch parts | 90.2 | 0 185 | 0.266 |

[a] These data are based upon 2083 Chilean-mill days or 49,992 Chilean-mill hours. Five mills operating for year and one mill for 9 mo., or a basis of six mills for 347 days or 5.77 mills for 361 days.

The record, or best, time for changing steel from time mill is shut down until started up with three men is: mullers, 2.6 hr., dies, 3.3 hr.

### Table 11.—*Steel Data for Year* 1918

*6-, 10-, and 18-mesh screens used*

Muller rings, three to a set, out. diam 58 in ; in. diam. 46¼ in.; width 7 in.
Die rings, out. diam 70 in.; in diam 56¼ in.; thick 6 in.

| Weights In | Weights Out | Tons Ore | Pounds Steel per Ton Ore | Cost Steel per Lb 1914 | Cost Steel per Lb 1918 | Cost Steel per Ton Ore 1914 | Cost Steel per Ton Ore 1918 |
|---|---|---|---|---|---|---|---|
| | | | *Muller Rings* | | | | |
| 5,844 | 717 | 14,380 | 0.406 | $0 058 | $0 0960 | $0.0236 | $0 0386 |
| | | | *Die Rings* | | | | |
| 2,342 | 391 | 8,269 | 0 284 | $0.062 | $0 1010 | $0 0176 | $0.0288 |
| | | | *Total Chilean-mill Steel* | | | | |
| | | | 0.690 | | | $0 0412 | $0.0674 |
| | | | *Victor Mill*, 1915, 18- *and* 30-*mesh Screens used* | | | | |
| | | | *Muller Rings* | | | | |
| 6,070 | 600 | 10,170 | 0 538 | $0 058 | $0 096 | $0.0312 | $0 0517 |
| | | | *Die Rings* | | | | |
| 2,345 | 388 | 6,089 | 0.385 | $0 062 | $0 1010 | $0 0238 | $0.0389 |
| | | | *Total Chilean-mill Steel* | | | | |
| | | | 0.923 | | | $0 0550 | $0 0906 |

Due to the fact that some steel was available that was purchased several years ago, the steel costs are less in 1918 than they would otherwise have been. Steel bought in 1918 cost: muller rings $0.122 and dies $0.127 per lb. at Victor, Colo.

### Table 12.—*Screen Data for Year* 1918

Five Screens to Each Mill. Average Area Available, 52 by 19¾ In

| Trade No. | Mesh | Size Slot Width, In. | Size Slot Length, In. | Size Wires Cross | Size Wires Long | Tons Ore, per Screen | Cost per Ton Ore, 1918 Prices |
|---|---|---|---|---|---|---|---|
| 45 | 6 | 0.130 | 0 438 | No. 11 | No. 13 | 850 | $0.0039 |
| 345 | 18 | 0.035 | 0.255 | No. 13 | No. 19 | 274 | 0 0115 |
| 482 | 10 | 0 063 | | | | 444 | 0.0073 |
| Cost screens, Independence mill, 1918 | | | | | | | $0.0083 |
| | | | *Victor Mill*, 1915 | | | | |
| 345 | 18 | 0.035 | 0.255 | No. 13 | No. 19 | 230 | $0 0125 |

The above screens are Ludlow-Saylor Rek-Tang brand.

### Table 13.—*General Data Fine Grinding Department, Independence Mill*

| | Horsepower |
|---|---|
| Power consumed running Chilean mill light, without muller or muller cores, revolving the head only, including motor and transmission losses | 5.7 |
| Power consumed as above except mullers are on and revolving on a new die. Steel on steel, no ore | 10.7 |
| Power to drive six feed plungers | 8.2 |
| Power to drive eleven Akin classifiers, full load | 6.7 |

## DISCUSSION

R. B. T. Kiliani, Denver, Colo. (written discussion*).—Mr. Lennox's paper is decidedly interesting in that he shows that a field exists for the Chilean mill, in which its efficiency compares favorably with that of more modern grinding devices. The figures he has given clearly indicate that a coarse feed is conducive to higher efficiency than the fine feeds which the Chilean mill has usually been called upon to grind. The fact is also apparently indicated that operating the Chilean mill in closed circuit with a mechanical classifier results in still higher efficiency, and that in such cases the percentage of returns should be kept below 100 per cent. to obtain best results. It would be interesting to see the results of further tests with what Mr. Lennox calls "Mixed Feed," since the data given in Table 4 seem to point to the fact that a mixed feed, with the mill in open circuit, may result in maximum efficiency for the Chilean mill, even higher than that obtained on a coarse feed and in closed circuit.

Mr. Lennox has mentioned the published data on tests between Chilean and Hardinge mills at the Miami Copper Co.'s plant several years ago, and has shown that the efficiency of the Chilean mill may be very materially increased over that obtained at that time. The developments in the use of the Hardinge mill at Miami have also been as great, since their present practice is grinding in conical ball-mills in two stages, the last in closed circuit, from mill bin to pass 48-mesh. I realize, of course, the danger in trying to make comparisons between different machines operating at different plants, but the work done by Mr. Lennox on the crushing resistance of various ores[6] gives a basis for such a comparison. I will show later that the constants derived by Mr. Lennox to represent the "grindability" of various ores can be so used.

The present practice at the Portland mill is to grind the coarse feed in Chilean mills equipped with 6-mesh screens, and operating in open circuit. Under these conditions, an average efficiency of 396 mesh-tons per horsepower is obtained, with a maximum of 413. As compared with this, an 8-ft. diameter Hardinge ball-mill in open circuit (the first stage mentioned above) has an efficiency, corrected for the "grindability" of the ore, of 486 mesh-tons per horsepower. Operating the Chilean mill in closed circuit is conducive to higher efficiency, as already mentioned, than open-circuit grinding. But even under these conditions of maximum efficiency for the Chilean mill, it can only show 418 (426) mesh-tons per horsepower.

Again, let us consider the tests on the Chilean mill in closed circuit, with a coarse feed, and with the finest grinding—or finest classifier overflow—mentioned in the paper. These are the two tests mentioned in

* Received Sept. 22, 1919.

[6] Grinding Resistance of Various Ores. *Trans.* (1919) **61**, 237.

Tables 5 and 7, with efficiencies of 328 and 377. Taking the average of these two tests, we obtain a final product containing 4.0 per cent. on 48-mesh and an efficiency of 353 mesh-tons per horsepower. As compared with this, one entire section of three Hardinge mills at Miami, grinding to 0.4 per cent. on 48-mesh, has an efficiency, corrected for "grindability" of the ore, of 457 mesh-tons per horsepower.

To summarize the above data, the following tabulation, giving efficiencies corrected for the character of the ore, is of interest:

| | Chilean Mill | Hardinge Mill |
|---|---|---|
| Efficiency when grinding in open circuit | 396 | 486 |
| Relative efficiency | 100 | 123 |
| Efficiency when grinding in closed circuit | 353 | 457 |
| Per cent. on 48-mesh | 4 0 | 0 4 |
| Relative efficiency | 100 | 129 |

The Miami Copper Co. and the Nevada Consolidated Copper Co. are both using Hardinge mills of the same diameter for practically the same range of work, namely as secondary or regrinding machines. Two runs at Miami show an average efficiency of 478 mesh-tons per horsepower. This, when multiplied by the constant of 0.70 for the Miami ore, gives a corrected efficiency of 335. In a similar manner, five runs at Nevada Consolidated show an average efficiency of 556 mesh-tons per horsepower which, when multiplied by the constant of 0.61 for this ore, gives a corrected efficiency of 339. This is a remarkably close check, being within about 1 per cent. Other data on Hardinge mills at other plants only further bear out the correctness of Mr. Lennox's factors.

C. H. Benedict,* Calumet, Mich.—At one plant we had 48 Chilean mills. At present 16 units are still so equipped, which we will probably continue to operate. The others have been replaced by another well-known type of mill. I am not here to act as spokesman for one mill or another, but I wish to point out, for the benefit of some who have not used Chilean mills, some of the possible difficulties that will be met. Of course, we were doing fine grinding, something that Mr. Lennox says these mills are not suited for. We attempted to grind 3/16-in. material and we were aiming for as fine a product as we could get; we were actually obtaining about 35 per cent. through a 200-mesh, the ore being a very hard material to handle, and we found, as compared with other types of mills, that our lost time on the Chilean mills was very large. I think Mr. Lennox gives a figure approaching 4 per cent. as lost time. That would correspond fairly closely with our experience; in other words, one day a month. You who have used ball-mills know that you can reduce that time to one day in five, six, or eight months.

Another thing that bothered us a great deal, inasmuch as we wished

* Metallurgist, Calumet & Hecla Mining Co.

to grind very fine and did not grind in closed circuit, was the fact that we had a great deal of trouble with our screens, our Chilean mills would not grind alike two days in succession. That trouble increases as you attempt to grind the product finer; your screen must then be thinner and finer and accordingly you get an increasingly complex problem.

Conditions of this description led us to replace the Chilean mills in one of our plants with pebble mills; and the only reason that we do not replace the other 16 Chilean mills is simply because the space and conditions are such that it would be impossible for us to do so. I give this as our experience and not necessarily as any criticism.

G. M. TAYLOR,* Colorado Springs, Colo.—When we installed our Chilean mills at our Victor mills we expected to get about 150 tons a day; when the mills were installed we got about 90 tons in 24 hr. when feeding them with a ¾-in. material. I went away just at that time so the manager of our Colorado Springs plant, who was in charge, decided that if we fed a much finer mesh we would get a greater tonnage—which was a natural thing to assume. He therefore installed two additional sets of rolls and crushed everything through a ¼-in. mesh and the tonnage immediately dropped to 75 tons from 90. I returned about that time and put everything through a 1¼-in. screen, when the tonnage went up to 140 tons the next day.

P. H. ARGALL, Magdalena, N. Mex. (written discussion†).—The Chilean mill has generally been considered a fine-crushing machine and as such has had many advocates; it has also had quite a number of denouncers; hence, the action of the Portland engineers in using the Chilean mill for the preparatory work usually accomplished in rolls or rock breakers and adding a ball-mill to do the fine grinding is novel, though of doubtful utility in the long run, even though a fair efficiency is obtained for the combination machines, as measured by Chilean mill practice.

In order to follow the development of Chilean mill practice on Cripple Creek ores, I might say that two 6-ft. Chilean mills were placed in operation in Stratton's Independence mill in April, 1908, treating ore crushed by rolls to pass ¼-in. screen aperture. Screening was, however, found to be an unnecessary refinement, as was also the use of plows in the mills; both screening and plowing were abandoned before the Portland engineers finished the experiments in stamp milling upon which they were then engaged. Two additional 6-ft. Chilean mills were added in the spring of 1911, three of them giving the required 10,000 tons per month capacity, one being always in reserve. The Portland company began using Chilean mills in June, 1910, at its Victor mill, and the same com-

* Manager, Milling Dept., Portland Gold Mining Co.

† Received Feb. 12, 1920.

pany purchased the Stratton's Independence mill in June, 1915, and remodeled it; it is now known as the Independence mill. A description of Stratton's Independence milling practice is given in the November, 1911, issue of *Mining Magazine* (London).

These Chilean mills were used as fine grinders and that they accomplished their purpose is shown by the screen analyses, in which 62 per cent. of the pulp passed 150-mesh screens. To make coarse crushers of these Chilean mills, the Portland engineers have increased the speed, substituted steel in various parts for cast iron, provided new and enlarged thrust bearings to take care of the centrifugal force incident to higher speed, and have, thereby, greatly increased the cost of the machines. They have also reduced the width of the tires and dies from 8 in., as used in the original mill, to 7 in., a reduction of 12½ per cent. in the crushing surfaces, to say nothing of the loss of abrasion or attrition incident to the larger surface. At the same time, they have apparently increased the weight of the rollers by using tires 5⅞ in. thick as against tires varying between 3 and 4 in. on the original mills, thereby increasing the crushing pressure per square inch of tire face in contact with the ore. The mills, as described by Mr. Lennox, are in reality but 5 ft. 10 in. in diameter, instead of 6 ft. as in the original mills.

Taking a circle following the center line of the die, we have a diameter of 5 ft. 4 in. by 33 r.p.m. or a velocity of 552.58 ft. per min. in the original mill against 5 ft. 3 in. by 37 r.p.m., or 610.24 ft. annular velocity in the redesigned Chilean. The speed of the rolls, as measured along the center line of the die and rolls, should not exceed 500 ft. per min. for the best results, and that is the speed I maintain in the Chilean mill here at the Ozark Plant. The economical speed of the Chilean mill is limited by centrifugal force and the increase of thrust bearings to balance such force is not to be commended.

*Size of Feed.*—After a trial run in Stratton's Independence mill in April, 1908, the sizing screens were removed and the feed from a 5K Gates rock breaker passed direct through two sets of 16 in. by 36 in. rolls to the Chilean mills. The size of the coarsest pieces seldom exceeded 1 in. (I refer now to phonolite ore) and probably would not average over ¾ in.; indeed, tests made on various occasions showed that 75 per cent. of the feed would pass a ¾-in. screen aperture. In 1912, a mill was erected by the Ozark Smelting & Mining Co. near Magdalena, N. Mex., for the purpose of treating zinc sulfide by flotation. In order to obtain the separation required, it was found that over 60 per cent. of the ore must be crushed to pass 150-mesh, for which reason one standard 6-ft. Akron Chilean mill was erected for an expected capacity of 150 tons per day. The ore for this mill was crushed by one 9 by 12-in. Blake crusher, and one set of 36 by 14-in. rolls to pass screen apertures of 1¼ in. in diameter.

*Crushing Qualities of Ores.*—I have always considered the Cripple Creek ores by no means difficult to crush. Rocks that break with a snap, like phonolite, which also has the good feature of breaking in slabs, are ideal for reduction in rock breakers and rolls. Granite and syenite, partly decomposed as they are at Cripple Creek, are easily reduced by almost any form of crushing or grinding machines and particularly by rolls. Phonolite is, however, a tough rock and offers considerable resistance to fine comminution, say from 3/16 in. down; but on the whole I consider the Cripple Creek ores of average crushing quality, if I may use so loose an expression—they are neither very hard nor very soft.

The ores from the Ozark mines contain considerable magnetite and pyroxenes, which are more difficult to crush than any Cripple Creek ore that I know of. On the other hand, the hematite in the Ozark ores is more easily crushed than the Cripple Creek granite and breccias, so that, on the whole, the Cripple Creek ores and those of the Ozark company compare fairly well in crushing quality. But since the density of the Ozark ores is greater, the proper factor for a true comparison would be cubic feet per hour instead of tons per hour. As an illustration of the crushing qualities of the Ozark ores in Chilean mill and tube-mill, I submit the accompanying partial test—partial because the ore was not weighed to either mill nor had we electric instruments to determine the exact horsepower—which is only valuable in showing that a 14 by 4-ft. tube-mill delivered a coarser product below 50 mesh than did the Chilean mill, both mills being fed from the same ore stream, all of which had passed screen apertures 1¼ in. in diameter. The Chilean-mill pulp contained 20 per cent. solids, the tube-mill pulp 36 per cent. solids. As previously stated, this ore was prepared for the fine crushers by crusher and roll. The wear of steel tires on the rolls is 0.0643 lb. per ton of ore.

Ozark Mill, March, 1916

| Week Ending March 11 | Chilean Discharge Per Cent. | Tube Discharge Per Cent. | Week Ending March 18 | Chilean Discharge Per Cent | Tube Discharge Per Cent. |
|---|---|---|---|---|---|
| +150 | 3.8 | 1.4 | + 50 | 3 1 | 1.4 |
| +100 | 9.7 | 11 4 | +100 | 9 3 | 10 3 |
| +150 | 12.6 | 20 0 | +150 | 12 4 | 17.9 |
| +200 | 9.8 | 10 9 | +200 | 9.4 | 12.3 |
| −200 | 64.1 | 56.3 | −200 | 65.8 | 58.2 |
| | 100.0 | 100.0 | | 100.0 | 100.1 |

In Table 14 some of the test runs of Mr. Lennox are assembled with other tests from normal mills so that the nature of the crushing may be shown at a glance. It is apparent that the Chilean mill proper is a fine crusher and that the combination method introduced at the Portland-Independence mill, namely redesigned Chilean and baby ball-mill, scarcely comes within the category of fine-crushing machinery.

TABLE 14

| | A | B | C | D | E | F | G | H | I |
|---|---|---|---|---|---|---|---|---|---|
| Screen number | 6 | 18 | 12 | 12 | 12 | 12 | | | |
| Aperture, in. | 0.130×0 438 | 0.035×0 255 | 0 046 | 0.046 | 0 046 | 0 046 | | | 0 046 |
| Area, sq. in. | 0.0569 | 0 0089 | 0.0021 | 0.0021 | 0 0021 | 0.0021 | 0 0569 | 0.0089 | 0 0021 |
| Mesh Ap. | | | | | | | | | |
| + 50, 0 0096 Pass 0.0116 per cent. | 42 9 | 35 | 21 | 26 5 | 6 3 | 6 5 | 43 | 35 | 3 8 |
| +100, 0.0056 Pass 0 0058 per cent. | 16 6 | 20 2 | 12 | 20 9 | 13 4 | 12 0 | 16 5 | 17 9 | 9 |
| +150, 0 003 Pass 0 0041 per cent. | 6.1 | 6 3 | 5 | 2 0 | 11 8 | 14 8 | 5 9 | 7 1 | 12 1 |
| −150, per cent. | 34.4 | 38.5 | 62 | 50.6 | 68 5 | 66.7 | 34.6 | 40 | 74.9 |
| Tons feed, in 24 hr. | 282 | 256 | 115 | 130 | 204 | 160 | 186 | 225 | 132 |
| Power, horsepower | 112 8 | 114.2 | 52 | 56 | 68 | 60 | 93 | 105 7 | 63 |
| Tons per horsepower-day | 2.509 | 2 20 | 2 2 | 2 33 | 3 | 2.66 | 2 | 2 13 | 2.09 |
| Revolutions of mill | 37 | 37 | 33 | 33 | 30 | 30 | 29 | 37 | 30 |
| Total steel used, pounds | 0 69 | 0 93 | 0 62 | 0 62 | 0 83 | 0 85 | 0 69 | 0 93 | 0 864 |

A and B are the average of the long-time runs in Table 1, of the combination of Chilean and ball-mills at the Portland-Independence mill. C is a test run at Stratton's Independence mill April, 1911. D is a test run at Stratton's Independence mill April, 1914. E is a test run at the Ozark Smelting & Mining Co. mill in April, 1915. F shows the average results at the Ozark mill for the fiscal year 1916. G is the Chilean-mill screen discharge at the Independence mill, Portland Co., as given in Table 1. H is the average of the special tests in Table 4 at the Independence (Portland) mill. I shows the results at the Ozark mill from August, 1917 to September, 1918.

Mr. Lennox thinks that the Chilean mill is not a fine grinder "at least in being able to produce efficiently the extreme fines that a ball-mill is capable of giving." He refers, of course, to the redesigned mill with the narrow tires and increased weight of rolls, the high-speed mill in which the abrading action has been considerably reduced and which, under the conditions of fine feed, could only utilize 80 per cent. of the power used with the coarse feed conditions explained in his paper. The reason is not far to seek. With the fine feed, the roll, he claims, is close to the die but it apparently did not occur to him that by increasing the width of the tires or by lessening the pressure per square inch of crushing surface between the tire and die, fine ore could be fed of any desired thickness and the results that he disclaims could have been easily attained. The analogy of the wagon on the hard and soft road bed is not pertinent, the narrow tire will be good for a certain load on the hard road, a wider tire will carry a similar weight on a softer road. The narrower the tire in the Chilean mill, the less abrasion is set up and, per contra, the wider tire gives the condition necessary for fine feed. With the narrow tires, heavy rolls, high speed, and coarse feed of thin slabby ore, the redesigned Chileans go "bumpty-bump," from which it is evident the Portland engineers have not quite abandoned the stamp idea started in their experimental mill about 10 years ago.

A word as to the Chilean mill as a coarse crusher. The angle of nip between a 58-in. diameter tire and the flat die is large, hence coarse feed can be easily nipped and passed over by the rolls. The wavy effect of the rolls being lifted to pass over and crush the large pieces and then drop heavily on the finer ore, many times during a revolution, gives a sort of stamping effect; but the die and mortar are scarcely designed to resist such action for any length of time. It is, however, on the application of the power that the ultimate analysis rests—a horizontal belt-driven shaft operating beveled gears that rotate a vertical spindle and drive head, standing out from which three vertical rolls rotate in heavy thrust bearings carried in the steel drive head. The leverage is quite considerable in a 6-ft. Chilean mill and the conditions entirely at variance with the rigidity and rugged strength required in a coarse-crushing

machine. Against this elaborate and intricate multi-jointed, multi-bearing, and geared apparatus, we may consistently compare a set of rolls, four simple bearings, and two drive wheels; can any one for a moment believe that the coarse-crushing Chilean developed at the Portland-Independence mill will replace the average roll? or even the average fine rock breaker for rough crushing? My preference would be entirely toward rolls for the reduction now being carried out in the coarse-crushing Chilean.

I have been a warm supporter of the Chilean, so has Mr. Lennox, at least of his redesigned coarse-crushing Chilean, and he tells us that his paper was written with the object of showing how the outcast Chilean had been restored by the Portland engineers to a field of great utility and high efficiency. I fear, however, that he has, instead, succeeded in burying it, possibly beyond the hope of resurrection. I feel that the Chilean's usefulness is entirely in the line of a fine grinder, particularly for small mills, such as we have here, in which a comparatively coarse feed can be reduced to a very fine condition in one operation and in one machine; but in making a coarse crusher out of a Chilean and running it in combination with a baby ball-mill to do the fine crushing, a combination is set up that has no right to exist. Either mill can satisfactorily do the fine crushing, while rolls or rock breakers can do the coarse crushing with better and higher efficiency than ever can be obtained in a Chilean mill. Finally, the cost of such a Chilean unit is quite excessive, few engineers would assume the responsibility for such an outlay realizing that the cost thereof must be returned in a few years as a legitimate item of cost per ton of ore milled.

Luther W. Lennox (author's reply to discussion).—It is true that the Independence Co., under Mr. Argall, was using Chilean mills at the time that the Portland engineers were running an experimental mill in which stamps were employed, but the stamps were used only because of the small unit capacity necessary, for the plant was an experimental metallurgical mill and was not for the purpose of trying out grinding machinery.

Regarding the fine-grinding installation of the Portland Co. at the Independence mill to take a 2½-in. (6.4-cm.) feed from the rolls, giving it an intermediate grind in Chilean mills and a final grind in ball-mills, Mr. Argall has only condemnation. The bases for his statements are derived from practice with Chilean mills at the Independence mill previous to its sale to the Portland Co., July 1, 1915, and since that time with the Ozark Co. at Magdalena, N. Mex. Data from these two plants are published by Mr. Argall, together with some Portland figures from the Independence mill since July, 1915.

Mr. Argall thinks that when the Portland engineers increased the

speed of the Chilean mills from 33 to 37 r.p.m., and reduced the width of the tires and dies from 8 to 7 in., at the same time increasing the thickness of tires from 4 to 5⅞ in., the Chilean mill was buried "possibly beyond hope of resurrection." Briefly, the reason for making these changes in the size of dies and tires was to give more flexibility in the operation of the mill by increasing the space between tire and protection plate on the one side and tire and standard on the other. This allowed the mill to be more easily controlled by the operator under varying feed conditions and with less danger of wrecking the mill. Also, the thicker tires call for less frequent changes than thinner ones. The increase in speed allows greater capacity, but we have not noted that this change contributes to the making of a coarse crusher out of the mill, as Mr Argall contends, nor have we found that it causes any mechanical difficulties that are out of proportion to the increased tonnage.

I will not dispute his statement that 500 ft. per min. gives the best results with rolls, but a Chilean mill is not altogether a roll. Mr. Argall does not give any parallel figures to support his condemnation of 37 r.p.m. as against 33 r.p.m., except the comparison of the product from his machines and that from the higher speed mills of the Portland, which would be sufficient if all conditions except speed were parallel with Portland, as unfortunately they are not.

We have no data to prove or disprove the correctness of his contention regarding the results of our changes of steel size, except that given by him comparing the Argall and Portland mill products. In support of his assumption that these two radical changes—speed and pressure of crushing surface per square inch—have made a coarse crusher of the Chilean mill, Mr. Argall gives, aside from his table of comparisons, a discussion of his reasons, which are largely based upon the increased pressure per square inch of crushing surface of the Portland mills over his mills. He contends that this is the reason that we are unable to force more than 80 per cent. of the power into the mill when we give it a fine feed as against a coarse feed. The weight of the roller, according to Mr. Argall,[7] with a new 8-in. tire is about 7200 lb. If the tire weighs 1350 lb., the heart, or roller without tire, will weigh 5850 lb. (2653 kg.). In the case of Portland 7-in. tires ,the total will therefore be 7800 lb. When a Portland tire is three-fourths worn out, it will give the same pressure per square inch upon the ore as a new Argall tire. Our experience with fine feed has not shown that the mill could be fed with "any desired thickness" of bed between die and tire even when the tire was three-fourths worn out. Neither has it shown that the new tire made the mill a coarse crusher, and one three-fourths worn out allowed the

[7] *Min. Mag.* (November, 1911).

mill to be a fine grinder. A nearly worn-out tire, however, does give less capacity through the mill.

Mr. Argall has no justification for his statement that the redesigned Chileans go "bumpty bump." The mills run smoothly giving their 250 to 300 tons per 24 hr. with no indication of bumps unless we decided to give them a fine feed, which would allow tramp steel to become a menace by the tire riding upon the steel and thus causing the tire to bump. Pages 519–522 give the reasons for giving the mills a coarse feed instead of a fine.

It is quite natural for Mr. Argall to include data from his Magdelena mill, since he is now located there. At this plant he also operates a Chilean mill at 33 r.p.m. and uses 8-in. tires. Were the Magdalena ore known to have the same grinding resistance as the Cripple Creek ores, the figures would be comparable. Mr. Argall states, however, that "on the whole the Cripple Creek ores and those of the Ozark Co. compare fairly well in crushing qualities." The data he has published show that about two tons of Ozark ore could be ground to the same mesh and for the same power as one ton of Cripple Creek ore. As compared with Cripple Creek ores, the magnetite in the Ozark ore is probably considerably less resistant to grinding rather than more so, as Mr. Argall believes.

He is a bit severe on cylindrical mills when he condemns the crushing qualities of the tube-mill at Magdelena for giving a coarser product than the Chilean, when he feeds the 4-ft. tube-mill with a 1¼-in. product. While an advocate of the Chilean mill in its proper place and with proper feed, as is also Mr. Argall, I feel that he is not helping the cause when, in his desire to discredit the statement that a Chilean mill "will not give efficiently the fine product that a ball-mill is capable of giving," he feeds his 4-ft. tube-mill with an unknown tonnage and compares the discharge with that of a Chilean mill, and states that the partial test is only valuable in showing that a "4 by 14-ft. tube-mill delivered a coarser product below 50 mesh than did the Chilean mill, both mills being fed from the same ore stream." As a result of this partial test, he states that "It is apparent that a Chilean mill proper is a fine crusher and that the combination method introduced at the Portland-Independence mill, namely redesigned Chilean and baby ball-mill, scarcely comes within the category of fine-crushing machinery." If he should install a classifier, placing his tube-mill discharge in closed circuit, and feed the proper tonnage and mesh, on this comparatively soft ore he would obtain a finer product than his Chilean mill can possibly give even when running inefficiently.

More important is the relative resistance to grinding effort of the Independence ore and dumps, as compared to those of the Portland. Unfortunately, we have no comparative data; we have only the fact that most of the Independence dumps were rejected from upper levels of the Independence mine, 800 ft. or above. The dumps contained a great deal of granite and breccia and some phonolite, all of which was

partly oxidized both in place and after years of exposure on the surface. The Portland ores sent to the Independence mill, since 1915, are a combination of Portland dump and ore-house reject, all containing a greater proportion of phonolite and unoxidized material than the Independence ores, as the Portland is now mining on 2300-ft. level. Beyond the statement that the ores treated, since 1915, at the Independence mill, are of a more refractory nature than those treated by Mr. Argall in 1911–15, I am able to make no claim, for I lack comparative data on the quality of the two types of rock. We have recovered some Independence dump left by Mr. Argall and have always been able to increase capacity with this softer rock.

The design Mr. Argall advocates as ideal would be to feed the Chilean mills with a product from rolls 75 per cent. of which would pass a ¾-in. aperture, all passing 1 in. He would allow the Chilean to do the fine grinding. This was the installation used by the Portland for seven years, except that the roll product was 2½-in. However, the discharge from the Chilean mills, although fine enough for low-grade ore ($3.00 or less) is not fine enough for higher grades.

Mr. Argall has not given any arguments or reasons backed by definite figures for his stand and radical statements as to the results of the change of design of the Chilean mill by the Portland engineers, except his table of mesh tests, comprising two test runs from the Independence during his regime—one in April, 1911, and one April, 1914. During July, 1915, The Portland Co. was in possession of the Independence mill, but made no radical change, at first, either in personnel or methods. It was noticed that four daily mesh tests with 30-, 50-, 100-, and 150-mesh screens were made upon Chilean-mill discharge, tube-mill feed, tube-mill discharge, and an intermediate sand; there was also a weekly test of the sand-tank head and filter tail, the last two for mesh and assay and consequently upon a different set of screens including a 200-mesh. When making a concentrate mesh test, the first set of screens was used as the meshes were not sent in for assay. A complete daily report of mesh tests with the weekly sand and filter rail for Apr. 27, 1915, is given here. This is a duplicate of many similar mesh tests, and is taken to illustrate the important points to be advanced.

Screen Analysis

| | +30 | +50 | +100 | +150 | −150 | +200 | −200 |
|---|---|---|---|---|---|---|---|
| Tube-mill feed, per cent . . . | 20 0 | 32.0 | 25.0 | 0 0 | 23.0 | | |
| Tube-mill discharge, per cent. | 0 0 | 7 0 | 32 0 | 0.0 | 61.0 | | |
| Chilean-mill discharge, per cent.................... | 8.0 | 20.0 | 33.0 | 0.0 | 39.0 | | |
| Sand-tank head, per cent . | 6.4 | 21.6 | 44.8 | 10.8 | .... | 12 4 | 4.0 |
| Filter tail, per cent ... ... | 0 0 | 0.0 | 0 0 | 0.0 | .... | 1.0 | 99.0 |

The striking feature about these tests is the unusual appearance of 0.0 per cent. retained upon the 150-mesh screen, whether it be Chilean-mill

discharge, tube-mill feed, or tube-mill discharge. A 100-mesh screen substituted for the 150-mesh will apparently give the same results.

Mr. Argall gives an aperture of 0.003 in. for his 150-mesh screen, and 0.0056 in. for his 100-mesh. The difference between these two is almost as great as that between a standard Tyler 200-mesh (0.0029 in.) and a 100-mesh (0.0058 in.) screen. If this 150-mesh screen is 0.003 in., Mr. Argall's work at the Independence is sufficiently good to justify his claim that the Chilean mill will produce a fine discharge and I am forced to concede that his mill was efficient even when grinding the semi-oxidized ores of the Independence dump and mine. Were the screen a Tyler Standard of 0.0041 in. opening, his discharge would still be good. But, the best that can be allowed, in light of his mesh tests taken from laboratory records of 1912 to 1915, is to alter the daily report shown to appear as follows:

Screen Analysis

| | +30 | +50 | +100 | −100 | +150 | +200 | −200 |
|---|---|---|---|---|---|---|---|
| Tube-mill feed, per cent. | 20 0 | 32 0 | 25 0 | 23 0 | | | |
| Tube-mill discharge. per cent. | 0 0 | 7 0 | 32 0 | 61 0 | | | |
| Chilean-mill discharge, per cent | 8.0 | 20 0 | 33 0 | 39.0 | | | |
| Sand-tank head, per cent | 6.4 | 21 6 | 44 8 | .... | 10 8 | 12.4 | 4.0 |
| Filter-tail, per cent | 0 0 | 0 0 | 0 0 | ... | 0 0 | 1 0 | 99.0 |

Table 15 gives an average of mesh tests taken from the laboratory and daily-report sheets; both a Chilean-mill and tube-mill discharge are included. A combination of these two in the correct proportion, which would be close to 50 per cent. each, will give the input to the cyanide department, ignoring the small withdrawal of 1.5 per cent. concentrates. The sand-tank head combined with the filter tail in the correct proportion, which is approximately 55 per cent. sand and 45 per cent. filter tail or slimes, will constitute the output from the cyanide department, as far as mesh is concerned. Obviously, the input and output should check within limits. As an example, the first six months of 1915 may be cited. From daily-report sheets, the sand-tank tonnage comprises 55 per cent. and the filter tails 45 per cent. of the output. Using these proportions and the table of mesh, we obtain the following average mesh as output:

| | | | | | |
|---|---|---|---|---|---|
| Screen analysis | +30 | +50 | +100 | −100 | (ignore the 150 mesh) |
| Output, sand and slime, per cent | 4.6 | 12.3 | 24.2 | 59.1 | |

The input tonnage will become, upon a basis of 50 per cent. Chilean-mill discharge and 50 per cent. tube-mill discharge,

| | | | | | |
|---|---|---|---|---|---|
| Screen analysis | +30 | +50 | +100 | −100 | (ignore the 150 mesh) |
| Input, Chilean mill, Tube, per cent. | 4.3 | 12.0 | 24.7 | 59 0 | |

By this means it may be seen that the input and output are consistent, using Mr. Argall's figures for the meshes 30, 50, 100, and through 100. The withdrawal of concentrates would have little effect upon this check.

Table 15 shows that in 1912 and the first six months of 1913, the meshes on 150 are much more consistent than after these dates. During this early period, it might be possible to combine input Chilean-mill and tube-mill discharges in such proportions that the input would check, within reason, the output, correctly proportioned between sands and slimes. However, after June, 1913, it would be impossible to do this.

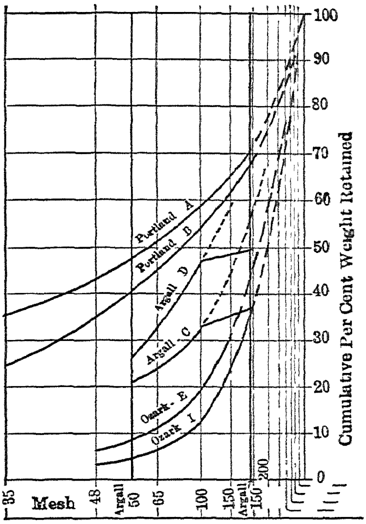



FIG. 7.—DIRECT SCALE. SOLID LINES ARE PLOTS FROM MR ARGALL'S TABLE; DOTTED LINES, THE PROBABLE CONTINUATION OF CURVES. LETTERS REFER TO CORRESPONDING MESH IN TABLE.

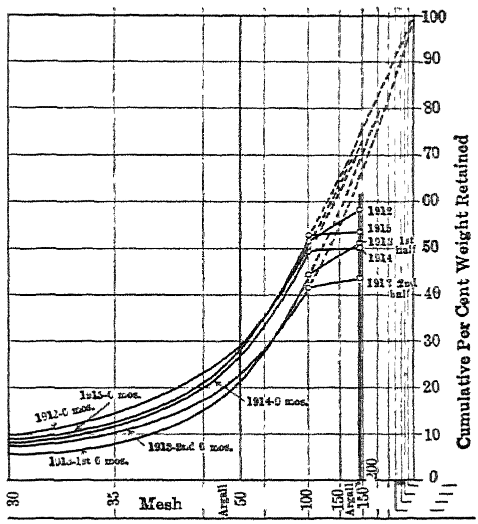



FIG. 8.—DIRECT SCALE. SOLID LINES ARE PLOTS OF CHILEAN-MILL DISCHARGE DURING YEARS 1912–15, INDEPENDENCE MILL. DOTTED LINES REPRESENT THE PROBABLE CURVE.

The data indicate that, according to the assay screens, which appear to be correct, the output from the mill has not varied appreciably in mesh, but the input has varied decidedly. Upon a daily report will appear mesh tests that give practically no tonnage between 100- and 150-mesh screens or between openings of 0.0056 and 0.003 in., as input and yet on the same report the output will have 6 or 7 per cent. between these meshes, or even more than this if the assay screens are standard, as I presume they are.

Further reference may be had to the plots of mesh tests (Figs. 1 to 3). The Ozark ore seems to take its place in the category of one easily ground, but it is consistent in its mesh. These plots are only a duplication of the mesh in the tables, but may appeal more in explaining the discrep-

ancies existing. I can give no other explanation, especially when having found the so-called 150-mesh screen to be the same as the 100-mesh to all appearance.

I do not feel that Mr. Argall has proved that the Chilean mill will give the extreme fines his tables and statements indicate, and I still feel that the Portland Co. has not committed an error in its grinding installation of Chilean mills for intermediate and ball-mills for fine-grinding work for either low or high grades of ore.

Table 15 is a copy of the form of daily report sheets at the Argall Independence Mill for years 1912–1915.

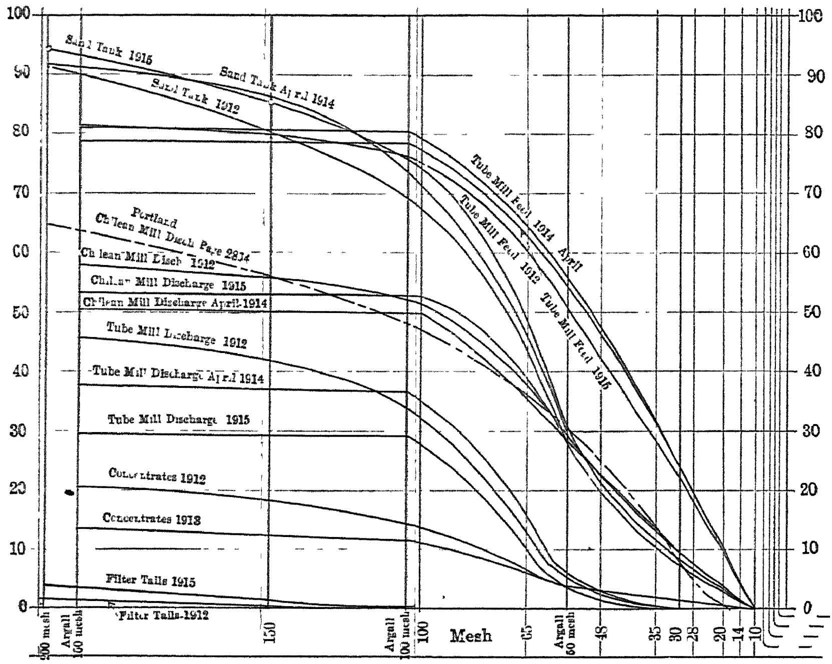



FIG. 9.—RECIPROCAL SCALE. FROM MESH TESTS OF INDEPENDENCE MILL. 6 MONTHS, 1912; APRIL. 1914; 6 MONTHS, 1915.

### TABLE 15.—*Average Mesh Tests of Independence Mill*

Average of 172 mesh tests made during six months of 1912.

| Aperture, inch | | 0 0096 | 0.0056 | 0.003 | | | |
|---|---|---|---|---|---|---|---|
| Screen analysis | +30 | +50 | +100 | +150 | −150 | +200 | −200 |
| Tube-mill feed, per cent | 22 5 | 28 4 | 25.3 | 5 0 | 18.3 | | |
| Tube-mill discharge, per cent | 0.0 | 4.0 | 28.9 | 11.8 | 55.7 | | |
| Chilean-mill discharge, per cent. | 9.7 | 19.8 | 22.1 | 6 4 | 41.6 | | |
| Sand-tank head, per cent | 7.2 | 21 0 | 39.3 | 12.7 | .... | 11.1 | 8.7 |
| Filter tail, per cent | 0.0 | 0 0 | 0.0 | 0 8 | .... | 3.5 | 95.7 |
| Concentrates (2.2 per cent.), per cent | 1.3 | 2 7 | 10 4 | 6.3 | 79.2 | | |

Average of 179 mesh tests made during first six months of 1913

| Screen Analysis | +30 | +50 | +100 | +150 | −150 | −200 | −200 |
|---|---|---|---|---|---|---|---|
| Tube-mill feed, per cent | 22 7 | 30 9 | 24 9 | 4 9 | 16 2 | | |
| Tube-mill discharge, per cent | 0 0 | 2 3 | 22.3 | 10 9 | 64 2 | | |
| Chilean-mill discharge, per cent | 6.0 | 15 7 | 22 9 | 6 4 | 48.6 | | |
| Sand-tank head, per cent | 7 3 | 20 5 | 38 9 | 13 1 | .. | 13 7 | 6.6 |
| Filter tail, per cent | 0 0 | 0 0 | 0 0 | 0 8 | | 2 9 | 96.3 |
| Concentrates (1.5 per cent.), per cent | 0 4 | 2 7 | 8 3 | 1 7 | 87 1 | | |

Average of 173 mesh tests made during second six months of 1913.

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Tube-mill feed, per cent | 27 7 | 29.0 | 21 9 | 1 3 | 20 0 | | |
| Tube-mill discharge, per cent | 0.0 | 4 5 | 22.5 | 3 4 | 69 5 | | |
| Chilean-mill discharge, per cent | 7.7 | 15.5 | 18 2 | 2 0 | 57.1 | | |
| Sand-tank head, per cent | 8 4 | 19.6 | 40 3 | 14 2 | ... | 9 2 | 7 9 |
| Filter tail, per cent | 0.0 | 0.0 | 0 0 | 0 8 | .. | 2 0 | 97.2 |
| Concentrates (1.5 per cent.), per cent | 0 4 | 2 7 | 8 3 | 1 7 | 87 1 | | |

Average of 91 mesh tests made during six months of 1914

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Tube-mill feed, per cent | 23 7 | 32.3 | 24.6 | 0 4 | 19.1 | | |
| Tube-mill discharge, per cent | 0 0 | 5 4 | 27 1 | 1 3 | 66 9 | | |
| Chilean-mill discharge, per cent | 8.1 | 19.0 | 22.1 | 0 9 | 49.8 | | |
| Sand-tank head, per cent | 7.9 | 22 4 | 44 1 | 12.4 | . | 6 4 | 6 9 |
| Filter tail, per cent. | 0 0 | 0 0 | 0 0 | 0 7 | ... | 1.5 | 97 8 |

Average of 169 mesh tests made during first six months of 1915.

| | | | | | | | |
|---|---|---|---|---|---|---|---|
| Tube-mill feed, per cent | 24 1 | 30 8 | 24 0 | 0 05 | 21.0 | | |
| Tube-mill discharge, per cent. | 0 0 | 3 6 | 25 6 | 0 6 | 70 2 | | |
| Chilean-mill discharge, per cent | 8 5 | 20 4 | 23 9 | 0 5 | 46 6 | | |
| Sand-tank head, per cent | 8.3 | 22.3 | 43.9 | 10 7 | ... | 9.2 | 5.6 |
| Filter tail, per cent | 0 0 | 0 0 | 0 0 | 0 4 | ... | 1.3 | 98.4 |

Tube-mill feed gave 9 mesh tests of 1.0 per cent. and 160 of 0.0 per cent. on 150-mesh screen.

Tube-mill discharge gave 8 mesh tests of 2.0 per cent., 95 of 1.0 per cent., and 66 of 0.0 per cent. on 150-mesh screen.

Chilean-mill discharge gave 5 mesh tests of 2.0 per cent., 80 of 1.0 per cent. and 84 of 0.0 per cent. on 150-mesh screen.

Output from Mill

| | |
|---|---|
| Sand-tank head, per cent. | 54.2 |
| Filter tail, per cent | 44.3 |
| Concentrates, per cent | 1.5 |

During April, 1914, test "D" quoted by Mr. Argall was made.

| Screen analysis | +30 | +50 | +100 | +150 | −150 | +200 | −200 |
|---|---|---|---|---|---|---|---|
| Tube-mill feed, per cent. | 23.4 | 31.4 | 25 6 | 1.3 | 18.3 | | |
| Tube-mill discharge, per cent | 0 0 | 5 5 | 31 0 | 1.2 | 62.1 | | |
| Chilean-mill discharge, per cent | 10 0 | 17.9 | 22 0 | 1.3 | 48.7 | | |
| Sand-tank head, per cent | 7 4 | 22.4 | 41.5 | 15.1 | .... | 5.4 | 8.3 |
| Filter tail, per cent | 0.0 | 0 0 | 0 0 | 1.0 | .... | 2.0 | 97.0 |

## Mill Operations at United Eastern during 1917 and 1918

BY WHEELER O. NORTH,* OATMAN, ARIZ.

(Chicago Meeting, September, 1919)

THE United Eastern Mining Co.'s property is in the Oatman, Gold Roads mining district of Mohave County, Ariz. The mine and mill are 26 mi. (41.8 km.) southwest of Kingman, the nearest railway connection. The ore consists of a mixture of calcite and quartz with some andesite. The values, which are mainly gold, alloyed with approximately 34 per cent., by weight, of silver, are extremely fine so that fine grinding is a most important factor in extraction. The milling process, briefly, consists of single-stage, coarse-crushing, two-stage ball milling in cyanide solution, followed by combined mechanical and air agitation, and by straight counter-current decantation. The Merrill zinc-dust precipitation method is used to recover the values, and is preceded by the Crowe vacuum treatment of the solution.

The most notable features of the plant are the absence of filters, and the use of short peb mills loaded with steel balls, for fine grinding. As originally designed, the capacity of the mill was to be 200 tons per day, although the grinding end was to be capable of handling double that amount. By adding extra agitators and thickeners and by some alterations to the grinding end, the daily capacity has been raised to 285 tons. This, it is hoped, may be increased to 300 tons during the warm months of the present year.

TONNAGE MILLED PER DAY

| | | | |
|---|---|---|---|
| 1917 average | 234; maximum month | 272; minimum month | 146 |
| 1918 average | 253; maximum month | 276; minimum month | 217 |

As this article primarily deals with plant operations, those interested in the mechanical description are referred to the article by Mr. Otto Wartenweiler,[1] the designing engineer, which gives the details and costs of design and construction. The following changes have been made since that article was published:

Two special simplex Dorr classifiers have been substituted for the two duplex Callow screens and bucket elevator, which classified the Marcy discharge and returned the oversize for regrinding. Three Dorr

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* Assistant General Manager, United Eastern Mining Co.

[1] The United Eastern Mining and Milling Plant. *Trans.* (1918) **59**, 274.

agitators 14 by 24 ft. (4.26 by 7.31 m.) diameter, and two Dorr thickeners, one 40 by 12 ft. (12.19 by 3.65 m.) and one, 30 by 8 ft. (9.14 by 2.43 m.), have been added to the cyanide plant. A 4-in. (10.1-cm.) Krogh centrifugal solution pump and a second Ingersoll-Rand compressor, 9 by 8 in. (22.8 by 20.3 cm.) were also necessary additions.

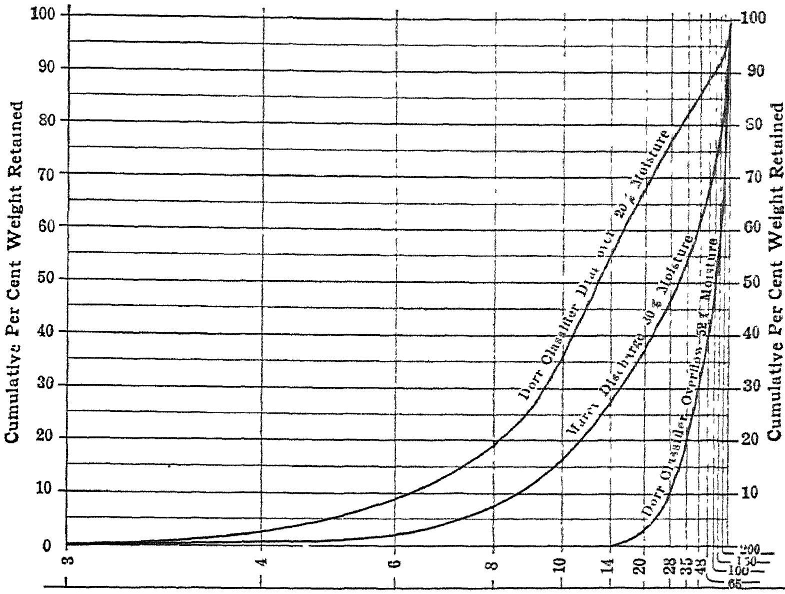



FIG. 1.—CUMULATIVE DIRECT DIAGRAM OF SCREEN ANALYSIS ON SAMPLE OF MARCY DISCHARGE AND CLASSIFIER PRODUCTS. MILL FEED, 270 TONS 2-IN. CRUSHER RUN.

## FACTORS IN OPERATING COSTS

*Water.*—Water for milling is pumped from a neighboring shaft and costs approximately 52 c. per 1000 gal. (3785.3 l.) or 11 c. per ton of ore milled. The consumption per ton of ore is 210 gal. (794.9 l.).

*Power.*—Power is supplied by the Desert Power & Water Co. from Kingman, at a cost of just under 2½ c. per kw.-hr. The current is metered on the low-voltage side of the transformers The average power load of the entire mill is 246 kw., or 330 hp., equivalent to 0.932 kw.-days, or about 1.25 hp. per ton of ore. The cost of power is approximately 52 c. per ton, or 24 per cent. of the total.

*Labor.*—Labor, which is 26 per cent. of the total cost, is entirely American, the wage scale being $5 for helpers, $5.50 for mill men, and $6 for solution shift foremen. The repair crew receive $4.50 and up. The total mill crew (exclusive of repair gang) of three shifts is normally fifteen men. The total labor cost, including superintendence, operation and repairs, amounts to 56 c. per ton; the average wage, $5.17 per shift;

the tons milled per man on shift, 14.27; per man on repairs, 22.79; average 8.76.

*Supplies.*—Supplies, which are 48.3 per cent. of the milling costs, come to $1.06 per ton milled. All supplies are hauled from Kingman over a good mountain road, at a cost of $8.50 per ton.

TABLE 1.—*Consumption of Chief Items of Supplies*

| | Pounds per Ton of Ore Milled | Pounds per Ton of Solution Precipitated | Per Ounce Fine Gold Recovered |
|---|---|---|---|
| Sodium cyanide. | | | |
| 1917 | 0 722 | 0 198 | 0.717 |
| 1918 | 0 747 | 0 216 | 0 709 |
| Average | 0 735 | 0 208 | 0 713 |
| Zinc dust: | | | |
| 1917 | 0 548 | 0 150 | 0 544 |
| 1918 | 0.388 | 0 113 | 0 368 |
| Average | 0.464 | 0.141 | 0 450 |
| Lime: | | | |
| 1917 | 4 150 | 1 140 | 4.121 |
| 1918 | 4.930 | 1 430 | 4.675 |
| Average | 4 550 | 1 290 | 4 417 |
| 5-in. chrome balls: | | | |
| 1917 | 0 925 | | |
| 1918 | 0.929 | | |
| Average | 0 927 | | |
| 1½- and 2-in chrome balls: | | | |
| 1917 | 2 12 | | |
| 1918 | 2.14 | | |
| Average | 2.13 | | |

Table 2 shows the operating costs per ton of ore milled for the past two years:

TABLE 2.—*Cost of Operations*

| | Operating Labor | Repair Labor | Supplies | Power | Miscellaneous | Total |
|---|---|---|---|---|---|---|
| 1917 | $0.4738 | $0 0902 | $1 0675 | $0 4751 | $0 0448 | $2.1514 |
| 1918 | 0.4664 | 0.0834 | 1.0463 | 0 5592 | 0 0297 | 2.1850 |
| Average | 0.4698 | 0.0866 | 1 0565 | 0.5197 | 0 0374 | 2 1691 |

## OPERATIONS

***Crushing.***—In the mine before reaching the skip pockets, all the ore passes through grizzlies set at a maximum opening of 10 in. (25.4 cm.). The 2½-ton skips dump automatically into the flat-bottomed coarse-ore bin, which is 16½ by 16½ by 20 ft. (5.03 by 5.03 by 6.09 m.), holds

250 tons, and will deliver by gravity about 180 tons. The feed is taken from the center of the bottom of the bin by a 42-in. (1.06 m.) maple-lined, link-belt, pan conveyor, using 2 kw. average load, and operating at a speed of 3 ft. (0.914 m.) per min. Hand picking for waste on this drag conveyor yields about 2 per cent. of the mine run at a cost of 85 c. per ton of waste.

The coarse crusher, a Telsmith No. 6 primary breaker, is set to deliver a 2-in. (5-cm.) product, and handles about 35 tons per hour, using 10 kw. average power load. Lately, the crusher has been preceded by 1¾-in. (4.44 cm.) grizzlies which by-pass most of the fines to the conveyor belt direct, thus relieving the crusher load and materially reducing crusher repairs.

From the crusher the product is conveyed by an 18-in. (45.72 cm.) belt set at 20° incline over a Merrick weightometer, and delivered to the fine-ore bin. This belt, operating at a speed of 250 ft. (76.2 m.) per min. handles up to 50 tons per hour, taking 7.6 kw. average power. The weightometer is checked monthly by a roller chain and, periodically, by weighing a quantity of ore on platform scales. The results have been entirely satisfactory.

In the analysis of crushing costs, the increase in the second year is largely due to repair or replacement items, which naturally would not appear in the first year's operations.

TABLE 3.—*Coarse Crushing Costs per Ton of Ore*

| | Operating Labor | Repair Labor | Supplies | Power | Total |
|---|---|---|---|---|---|
| 1917............ | $0.0211 | $0.0092 | $0.0170 | $0.0112 | $0.0585 |
| 1918........ .... | 0.0209 | 0.0153 | 0.0359 | 0.0100 | 0.0821 |
| Average.......... | 0 0210 | 0.0123 | 0.0269 | 0.0106 | 0.0709 |

Two-stage crushing would be desirable to secure finer ball-mill feed and doubtless could be effected at an equal, or even lower, cost than here obtained.

A nice point of design is the relation of ore storage, particularly mine run, to the daily output. A storage of several days' ore should be provided for, as it is an important factor in the continuous and even operation of both mine and mill. This point is too often overlooked in otherwise well-designed plants for smaller sized mines.

*Fine-ore Bin.*—The fine-ore bin is a 24 by 24-ft. (7.3 by 7.3-m.) flat-bottom wood tank furnished by the Pacific Tank Co., and made of 4-in. (10.16-cm.) merchantable Oregon pine; its capacity is 500 tons. This makes a most economical and satisfactory bin for this purpose. The belt dumps above the center of this bin and the feed is drawn from the

center of the bottom by a rack-and-pinion gate feeding an 18-in. link-belt, pan conveyor running at 33 in. (83.82 cm.) per min., using 2 kw. on average load, and delivering up to 15 tons per hour. The amount of feed is controlled by the opening of the bin gate. A record is kept of this opening, as measured by the number of teeth or notches on the bin-gate operating rack shown above a stationary marker at one side. These notches multiplied by the time give notch-hours from which a known factor gives a fairly accurate estimate of the tonnage milled for each shift or for any desired period.

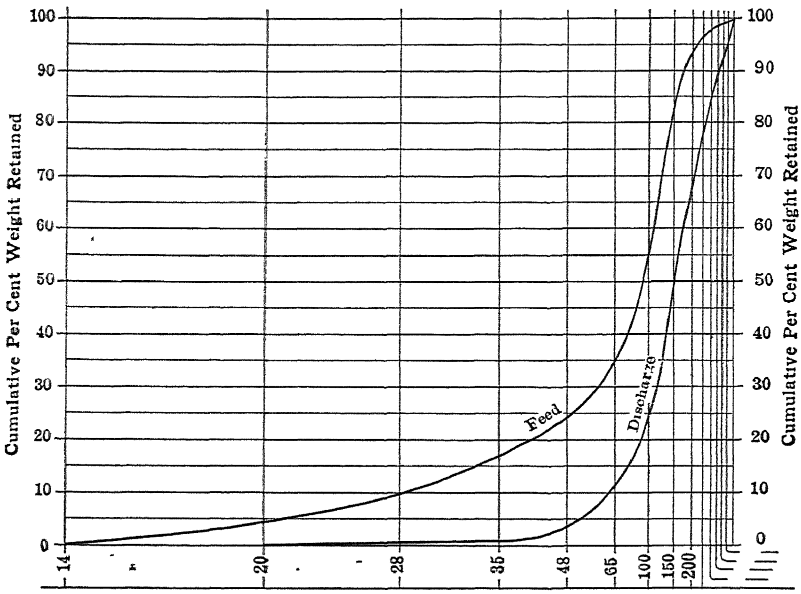



FIG. 2.—CUMULATIVE DIRECT DIAGRAM OF SCREEN ANALYSIS ON SAMPLE OF "BALL PEB" MILL FEED AND DISCHARGE. FEED 85 TONS PER 24 HR., 31 PER CENT. MOISTURE. ALL PASS 14 MESH.

*Coarse Grinding.*—The coarse grinding is done in a No. 64½ Marcy mill, working in closed circuit with a special simplex Dorr classifier, set on a 3-in. (7.62 cm.) slope and making 34 strokes per minute. This mill operates at 26 r.p.m. and carries a load of 9000 lb. (4082.3 kg.) of 5-in. (12.7 cm.) forged chrome balls. It reduces 260 tons of ore as it comes from the crusher, all of which would pass through a 2-in. (5.08-cm.) ring to all through 20 mesh. The average power used is 67 kw. The grinding is done in a 1.6 (725.7 gm.) KCN solution showing 1 lb. (453.5 gm.) protective alkali. The accompanying tables and screen analysis give the details of the mill operations and supplies.

Control of the coarse grinding is maintained by varying the specific gravity of the pulp in the special Dorr classifier, and by the bin gate.

TABLE 4.—*Ball-mill Data*

| | R p m | Peripheral Speed of Mill, Ft Per Min | Tons Per Hour, Original Feed | Average Power Load, Kw-hr | Kw-hr. Per Ton, Ground | Ball Load, Pounds |
|---|---|---|---|---|---|---|
| No. 64½ Marcy mill 2-in. feed ground to pass 20 mesh . . . | 26 | 490 | 11 00 | 67 | 6 09 | 9000 |
| Allis-Chalmers 5 × 6 Peb Mills 20-mesh feed ground 82 per cent. − 200 . . . . . . . . . . . . . . . . . . . . | 28 | 440 | 3 75 | 48 | 12 80 | 8000 |

TABLE 5.—*Total Steel Consumption*

| No 64½ Marcy Mill | Lb Per Hr, Including Scrap | Lb Per Ton, Including Scrap | Per Cent of Metal Lost as Scrap | Cost Per Ton, Ore Milled |
|---|---|---|---|---|
| 5-in. chrome-steel balls . . . . . . . . . . . . | 10 58 | 0.925 | None | $0.0544 |
| Feed-end liners, manganese steel . . . . . . | 0 717 | 0.072 | 32 2 | 0.0110 |
| Shell liners, manganese steel, step type | 1 650 | 0 163 | 45 5 | 0.0268 |
| Discharge grates, chrome steel . . . . . . . | 0.566 | 0.055 | 48.4 | 0.0137 |
| Bolts, clamp bars, center liners, etc. . . . . | 0 214 | 0.022 | 63.8 | 0.0068 |
| Total liner consumption . . . . . . . . . | 3.147 | 0.312 | 43 | 0.0583 |

The discharge of the mill is maintained at about 30 per cent. moisture, which is determined and recorded hourly. The normal return circuit from the classifier is about two to one of original feed, but this ratio increases rapidly as the rate of original feed is lowered, as it is necessary to return more oversize to keep the mill from pounding.

As in coarse crushing, the second year's costs are increased by renewals and replacements not occurring in proportionate amount in the first year's operation. Other factors tending to increase the 1918 costs were increases in wages of 28 per cent., power 15 per cent., and supplies in some cases, over 100 per cent.

TABLE 6.—*Coarse Grinding Costs per Ton of Ore*

| | Operating Labor | Repair Labor | Supplies | Power | Miscellaneous | Total |
|---|---|---|---|---|---|---|
| 1917 . . . . . . . . . . . | $0.0429 | $0.0140 | $0 1097 | $0.1183 | $0.0001 | $0.2850 |
| 1918 . . . . . . . . . . . | 0 0387 | 0.0165 | 0 1424 | 0 1465 | 0.0021 | 0.3462 |
| Average . . . . . . . . | 0.0407 | 0.0153 | 0.1267 | 0.1330 | 0.0014 | 0.3171 |

*Fine Grinding.*—Fine grinding is accomplished in two 5 by 6-ft. (1.52 by 1.82 m.) Allis-Chalmers ball granulators. These mills carry a ball load of 8000 lb. (3628.7 kg.) of 2-in. (5.08 cm.) chrome steel pebs and consume 48 kw. average load. They grind some 90 tons each from all-

through-20-mesh, to 82 per cent. minus-200-mesh. These mills work in closed circuit with standard model C, duplex 54-in. (137.16 cm.) Dorr classifiers, set at 2¼-in. (5.7-cm.) pitch and operating at 18 strokes per rake per minute. As this size classifier was found to be greatly overloaded for such fine separation, an 8-ft. (2.4-m.) Callow cone was placed to take the overflow of each classifier. The spigot product of the cone is returned by Frenier pump to the classifier feed boxes. The slime overflow of the classifier was raised 4 in. (10.16 cm.) which also helped materially in the classification.

In the fine grinding, the moisture content of the discharge is maintained at about 30 per cent., and, just as in the coarse grinding, the main

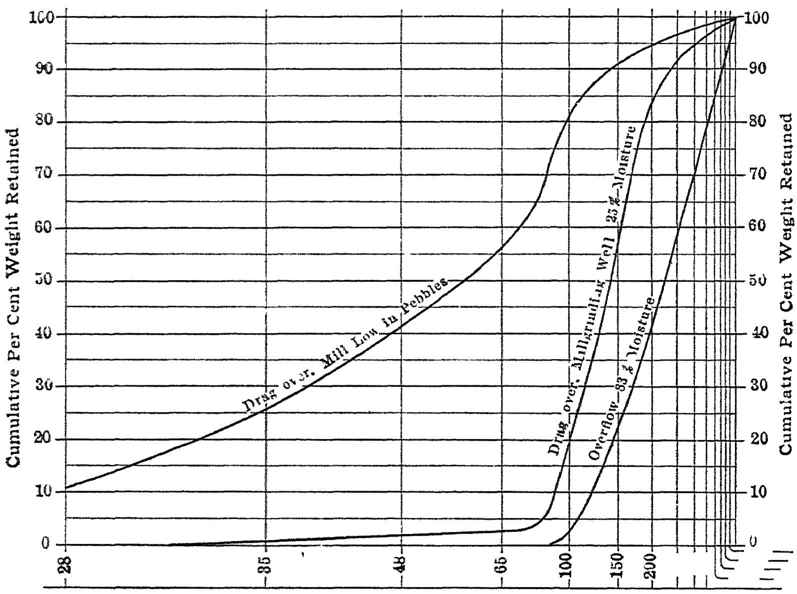



FIG. 3.—CUMULATIVE DIRECT DIAGRAM OF SCREEN ANALYSIS ON SAMPLE OF DUPLEX DORR CLASSIFIER. BALL-MILL FEED 85 TONS PER 24 HR. ALL PASS 28 MESH.

control is the specific gravity of the classifier, measured by hydrometer and recorded hourly. With an overflow of 4.5 to 5 of solution, to 1 of ore, the final grinding is maintained at about 82 per cent. through 200 mesh.

The addition of steel balls to all mills is made daily, the amount being calculated from the tonnage milled. A record is kept of the hours run and tons milled for the different parts of the mill liners, so that it is necessary to open a mill only when this schedule shows some part approaching exhaustion.

As stated before, the duplex classifiers are followed and aided by two 8-ft. Callow cones. These cones are equipped with the usual gooseneck,

the discharge, a 1½-in. (3.8-cm.) pipe, being 30 in. (76.2 cm.) below the overflow of the cone. After considerable experimentation, this size of outlet and head for the discharge proved to be best suited to our needs. Each cone feed or classifier overflow contains some 213 tons of solids and 970 tons of solution. The underflow carries back 73 tons of solids and 86 tons of solution, leaving a cone overflow of 140 tons of solids and 884 tons of solution. The screen tests, Fig. 4, show the results obtained. This arrangement also acts as a trap for any coarse gold, so that a considerable increase in value of mill heads can be accommodated without the necessity of cutting tonnage. The spigot product is returned to the Dorr classifier by No. 3 Abbe Frenier spiral sand pumps.

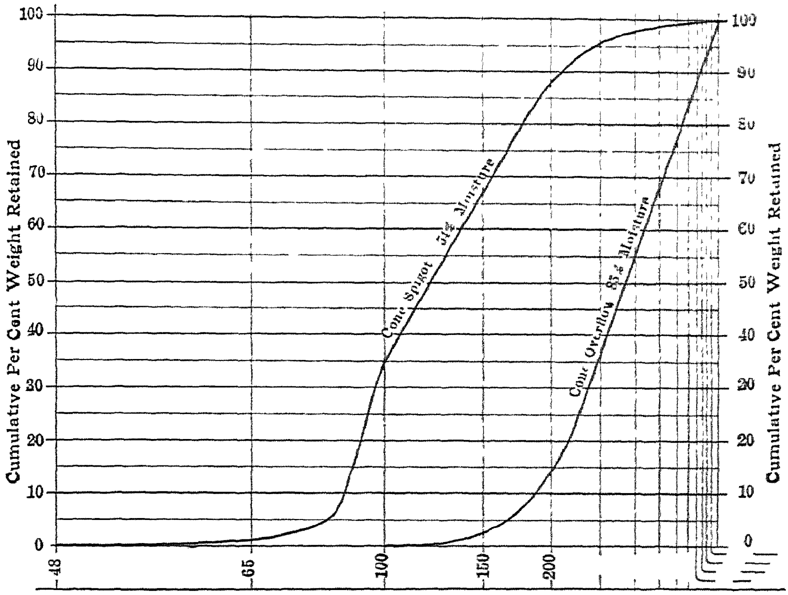



FIG. 4.—CUMULATIVE DIRECT DIAGRAM OF SCREEN ANALYSIS ON SAMPLE OF CALLOW CONE PRODUCTS. ALL PASS 48 MESH.

In passing from the grinding end of the plant it is of interest to note that comparative data in the district show that two-stage ball-milling is

TABLE 7.—*Fine Grinding Costs Per Ton of Ore*

| | Operating Labor | Repair Labor | Supplies | Power | Total |
|---|---|---|---|---|---|
| 1917........ | $0.0553 | $0 0328 | $0.1862 | $0.1899 | $0.4642 |
| 1918.......... | 0.0678 | 0.0171 | 0 2393 | 0.2055 | 0.5297 |
| Average.......... | 0.0618 | 0.0246 | 0 2140 | 0.1981 | 0.4985 |

TABLE 8.—*Total Steel Consumption, Including Scrap*

| Allis-Chalmers 5 by 6-ft Ball Peb Mills | Lb Per Hr | Lb Per Ton Milled | Lb Per Ton Reground | Per Cent Lost as Scrap | Cost Per Ton Ore Milled |
|---|---|---|---|---|---|
| 1½-in. chrome balls ........ | 11 90 | 2 120 | 3 180 | None | $0 1662 |
| 2-in. chrome balls ........ .. | 11 25 | 2 000 | 3 000 | None | 0 1568 |
| 1½-in. cast-iron balls........ .. | 16 30 | 2 900 | 4.350 | None | 0 1592 |
| Feed-end chrome steel liners . . . | 0 220 | 0 040 | 0 060 | 52 | 0.0072 |
| Throat chrome steel liner. ...... | 0 037 | 0 007 | 0 010 | 40 | 0 0012 |
| Shell chrome steel liners.. .... | 0 630 | 0.115 | 0 173 | 28 | 0.0218 |
| Discharge grates, tool steel . .. . | 0 029 | 0 005 | 0 008 | 40 | 0.0014 |
| Discharge wedges, chrome steel. | 0 093 | 0 017 | 0 026 | 41 | 0 0031 |
| Total liner consumption.... .... | 1 009 | 0 184 | 0 277 | 45 | 0 0347 |

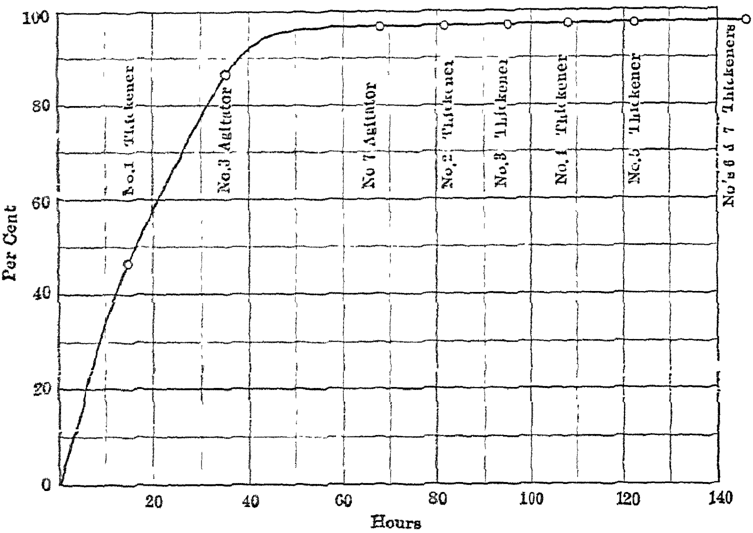



FIG. 5.—CURVE OF DISSOLUTION, 280 TONS BASIS.

about 15 per cent. cheaper than the same work accomplished with stamps and tube-mills. At the same time, indications are that single-stage ball-milling shows the same supply and labor costs but some 20 per cent. greater power consumption.

*Agitation.*—From the fine-grinding department, the pulp flows by gravity to No. 1 Dorr thickener, arriving with a gravity of 1.12 and a dilution of 1 part of ore to 4.5 parts of solution. The thickened underflow is pumped by a duplex, Campbell & Kelly No. 4 diaphragm pump to the first of a series of seven 14 by 24-ft. (4.26 by 7.31-m.) diam. Dorr agitators, which are arranged to handle the continuous flow of pulp by gravity. The flow of pulp is 280 tons per day with a specific gravity of 1.40. The period of agitation is about 62 hr., after which dissolving of values is almost negligible, as shown by the curve of dissolution, Fig. 5. Agita-

tion is carried on by air at 30 lb. (13.6 kg.) pressure. Compression is accomplished by two belt-driven Ingersoll-Rand compressors, Class E. R. I., size 9 by 8 in. (22.86 by 20.32 cm.) using 15 kw. each.

From the agitators the pulp flows through five 40-ft. (12.19-m.) Dorr thickeners arranged for straight counter-current work. The barren solution, 3.76 tons per ton of ore, is introduced in No. 4 thickener, and the wash water, about 0.82 tons per ton of ore, is introduced in No. 6 thickener. The flow at No. 6 thickener is split and about one-third is sent to No. 7 thickener, a 30 by 8-ft. (13.6 by 3.6-m.) tank. The discharge of No. 6 and No. 7 thickeners flow together to the tailings pond at a moisture content of 0.82 ton of solution to 1 ton of ore.

The seven agitators, seven thickeners, and six duplex diaphragm pumps are all driven by one line shaft and consume an average of 6 kw. or just over 8 hp. The solutions are handled by direct-connected centrifugal pumps controlled by automatic float switches.

The primary thickener taking the dilute overflow of the cones, 280 tons of ore to 1768 tons of solution, has a settling area of 4.37 sq. ft. (0.407 sq. m.) per ton of ore, and a settling ratio of 38 cu. ft. (1.076 cu. m.) per ton of ore. It is necessary, however, to operate this thickener with a low mud line to prevent colloidal material being drawn over into the gold tanks. About 870 tons of solution are precipitated daily and the balance, about 600 tons, is returned to the mill storage.

The regulation of all thickeners is maintained by varying the speed or the length of stroke of the diaphragm pumps. The gravity of the discharge and depth of the mud line below the overflow are determined and recorded every 4 hr. These data are plotted on cross-section paper and form part of the permanent records. The gravities are determined by weighing a milk can holding about 1 gal. (3.7 l.) of pulp on a beam scale, the beam of which is graduated to read in specific gravity. The mud line is determined by use of a weighted bottle, which will sink in solution but float on the mud. The maximum gravity is obtained with the highest mud line consistent with a clear overflow, other factors of alkalinity, temperature, and density of feed remaining constant.

Titrations are made and recorded every 2 hr. All additions of cyanide are made to No. 1 agitator, where the cyanide is built up to 2.7 lb. (1.22 kg.) KCN per ton of ore. Lime is added dry to the Marcy mill feed.

TABLE 9.—*Cyaniding Costs Per Ton of Ore*

| | Operating Labor | Repair Labor | Supplies | Power | Miscellaneous | Total |
|---|---|---|---|---|---|---|
| 1917 | $0.0972 | $0.0237 | $0.4575 | $0.0884 | ....... | $0.6668 |
| 1918 | 0.0867 | 0.0257 | 0.3553 | 0.1153 | $0 0018 | 0.5848 |
| Average | 0.0917 | 0.0248 | 0.4042 | 0.1025 | 0.0010 | 0.6242 |

The reduction in cost price of cyanide during 1918 more than offsets the adverse factors of power and labor in cyanide costs.

The counter-current flow sheet affords an interesting comparison of theoretical and actual results. Doubtless the main factor contributing to the deviation of these figures is the dissolution of values in the thickening zone; this is naturally most apparent in No. 1 thickener.

*Milling Conditions.*—Each day 263 tons are crushed in cyanide solution, $20.66 gold is dissolved as follows: $10 end No. 1 thickener, $10.30 in agitators; 20 c. No. 2 thickener; 0.07 c. No. 3 thickener; 0.04 c.

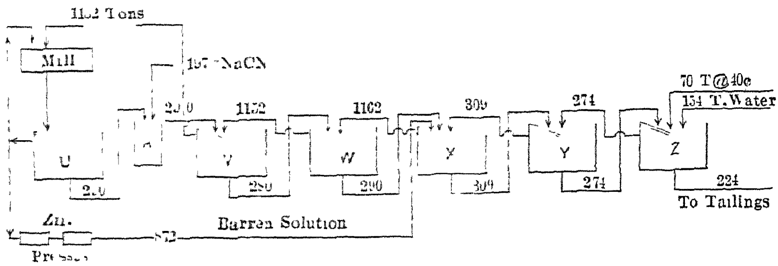



FIG. 6.—UNITED EASTERN FLOW SHEET COUNTER-CURRENT DECANTATION.

No. 4 thickener; 0.02 c. No. 5 thickener; 0.03 c. No. 6 thickener. No. 1 thickener discharge 52.5 per cent. moisture; No. 2 thickener, 51.5 per cent.; No. 3 thickener, 52.5 per cent.; No. 4 thickener, 54 per cent.; No. 5 thickener, 51 per cent.; No. 6 thickener, 46 per cent. Eight hundred seventy-two tons from No. 1 thickener precipitated to 2.5 c. Agitation, 1 part of ore to 1½ part of solution at 2.6 lb. KCN. Seventy tons wash solution at 40 c. returned from dam to No. 6 thickener. Cyanide strength No. 1 thickener 1.9 lb. KCN, add 0.75 lb. NaCN per ton ore = (0.94 lb. KCN) to No. 3 agitator.

TABLE 10.—*Values of Solutions in Various Thickeners*

| | Solution Values | | Cyanide Strength (KCN) Pounds per Ton | |
|---|---|---|---|---|
| | Theoretical | Actual | Theoretical | Actual |
| U | $6 24 | $6.22 | 1.90 | 1.90 |
| V | 3 97 | 5 85 | 1.9819 | 2 00 |
| W | 1.04 | 1.78 | 1.7881 | 1.80 |
| X | 0 29 | 0.55 | 1.7407 | 1.70 |
| Y | 0 27 | 0 40 | 1.244 | 1.20 |
| Z | 0.20 | 0 26 | 0.685 | 0.80 |

The theoretical dissolved loss per ton of ore is $0.1704, the actual loss is $0.215. The theoretical cyanide loss per ton of ore is 0.4675 lb. of NaCN, the actual loss is 0.525 lb. NaCN.

*Precipitation.*—The pregnant solution from the primary thickener flows by gravity to the No. 1 gold tank from which it is pumped by a 2½-in. Krogh centrifugal pump through a Merrill central-sluicing clarifying filter consisting of 28 frames, 3½ ft. square. This filter delivers the clear solution to the No. 2 gold tank, or steady-head tank of the Crowe vacuum-treatment system. Zinc dust is added to the solution as it leaves the vacuum-treatment system. Control of precipitation is largely mechanical and is based on past experience. The shift foreman checks the rate of zinc feed by weighing a 5-min. run, which is normally about 110 gm. but varies from 100 to 120 gm., according to the assay value of the solution being precipitated. When a new press is cut in, 20 lb. (9.07 kg.) of zinc dust is added at once and the rate of feed is increased for 6 hr. to 220 gm. per 5 min. The solution discharged from the press during the first 15-min. run is returned to the gold tank. This is with a precipitation rate of some 36 tons of solution per hour.

The cost of precipitation showed a decided reduction in 1918, due partly to the lower cost price of zinc dust but mainly to the use of the Crowe vacuum treatment, which effected a saving of nearly 50 per cent. in the amount of dust used.

TABLE 11.—*Precipitation Cost per Ton of Ore*

| | Operating Labor | Repair Labor | Supplies | Power | Total |
|---|---|---|---|---|---|
| 1917......... | $0 0003 | $0 0032 | $0.1274 | $0 0243 | $0 1552 |
| 1918............ | 0.0002 | 0 0024 | 0.0946 | 0 0178 | 0.1150 |
| Average. ....... | 0.0002 | 0 0028 | 0.1103 | 0 0209 | 0.1342 |

*Vacuum Treatment.*—Vacuum treatment is carried out in a 4 by 10-ft. (1.2 by 3.01-m.) vacuum receiver or drum set on end, the 6-in. (0.15-m.) inlet of which is in the center of the top and 23.4 ft. (7.13 m.) above the solution level in the steady-head tank, and is poured down over a series of perforated trays. During this process, the occluded air is liberated and removed by the vacuum pump. The solution level in the vacuum receiver is maintained at a height of about 30 in. (76 cm.) by a float operating a butterfly-valve in the intake pipe. This float rod is fitted with an electric contact, which sounds a "Klaxon" alarm if for any reason the solution level gets dangerously low. The discharge of the vacuum receiver is from the center of the bottom. It feeds by gravity to a 7 by 8-in. (17.7 by 20.3-cm.) triplex pump set so that the pump glands are 32.8 ft. (9.99 m.) below the solution level in the receiver. This keeps a slight pressure on the pump glands and prevents any intake of air at that point. The zinc dust is fed to this pump, where it is mixed with vacuum-treated solution. It is then pumped to the refinery and the two 32-frame, 36-in. Merrill precipitating presses. The barren solution

### SURFACE PLANT

1. Shaft collar. 3-compartment shaft 5 ft 0 in. by 5 ft. 0 in in clear
2 Head-frame 32 ft 0 in by 36 ft. 6 in. base 76 ft. 0 in to sheaves
3 Hoisting sheaves 7 ft 0 in diam 4$\frac{7}{16}$ in. bore 1 in cable
4 Skip-dumping frame
5 Allis-Chalmers double-drum electric hoist, parallel motion post brakes 150-hp motor, direct connected through herringbone gears. Rope speed 800 ft per min 1 in rope
6 Mine compressors, Ingersoll Rand, 9 in by 12 in by 16 in., belt-driven 888 cu. ft free air per min to 100 in press
7 150-hp motor 695 r p m 24 in by 20 in. pulley
8 Hoist house
9 Three concrete walls 12 in thick, supporting bin
10 Concrete wall 14 in top, batter 1 in per ft. Bin support and retaining wall above crusher floor.
11 Coarse ore bin 16 ft 6 in by 16 ft 6 in. by 19 ft 6 in. deep. Discharge 150 tons gravity.

### COARSE-CRUSHING DEPARTMENT

12 42-in link belt pan conveyor, 1½ ft per min. 25 tons per hr. 3-hp motor
13. No. 6 Telsmith breaker 6 to 1 reduction 46 in. by 12 in. pulley
14. 50-hp. Allis-Chalmers motor, 8 in. by 12 in. pulley. 860 r.p m.
15. Crusher house
16. 18-in. trough conveyor. Rise 20° weighted takeup
17. Merrick weightometer.
18 10-hp. Allis-Chalmers motor, 9 in by 7 in. pulley, 575 r p m.

### GRINDING AND CYANIDE DEPARTMENT

19 Mill bin 24 ft diam 24 ft high, 3 in. staves, 4-in. bottom, discharge 400 tons gravity.
20 Mill building
21. 18-in link belt pan conveyor, 1½ ft per min 8½ tons per hour.
22 3-hp motor
23 Two No. 64½ Marcy mills. Each, 300 tons per 24 hours from 1 in to 20 mesh 25 r p m
24 Two 100-hp motors, direct connected, 435 r p m
25 Two Simplex Dorr classifiers, drag-over returned to Marcy mill
26 Mechanical distributor
27. Three Duplex Dorr classifiers in closed circuit with ball-peb mills
28 Three 5 ft. by 6 ft Allis-Chalmers ball-peb mills with 42 in scoops, 28 r p m
29 Three 75-hp Allis-Chalmers motor, direct connected, 435 r p m.
30 Two 8-ft callow cones, spigot discharge to classifiers
31. Frenier sand pump, belt driven, 22 r p m.
32 No 1 thickener 40 ft by 12 ft Dorr m'ch'y, $\frac{1}{10}$ r p m of arms-settling area, 1227 sq ft Cap 96,000 gal, 480 tons
33 4-in pipe, pulp from thickener to pump
34. 4-in. pipe, clear solution from thickener to sol tank.
35 No 1 diaphragm pump
36 No 1 agitator tank. 24 ft by 14 ft Cap 39,100 gal -196 tons Dorr machinery, 2 to 3 r p m of arms
37 No. 2 agitator (same)
38 No 3 agitator (same).
39 No 4 agitator (same).
40 No 5 agitator (same).
41 No 6 agitator (same).
42. No 7 agitator (same).

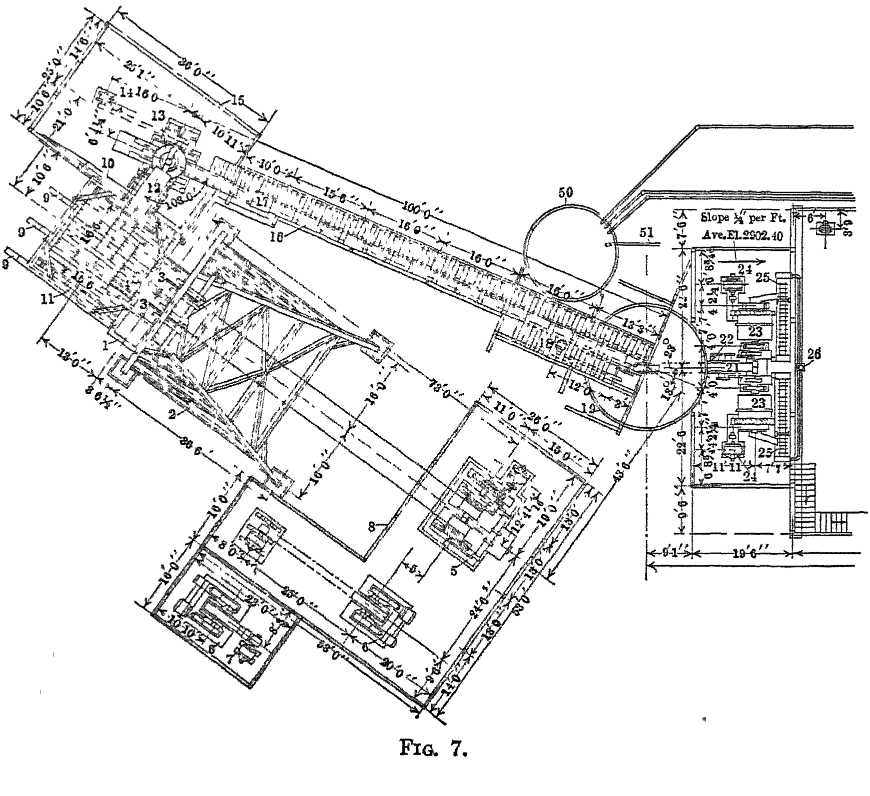



FIG. 7.

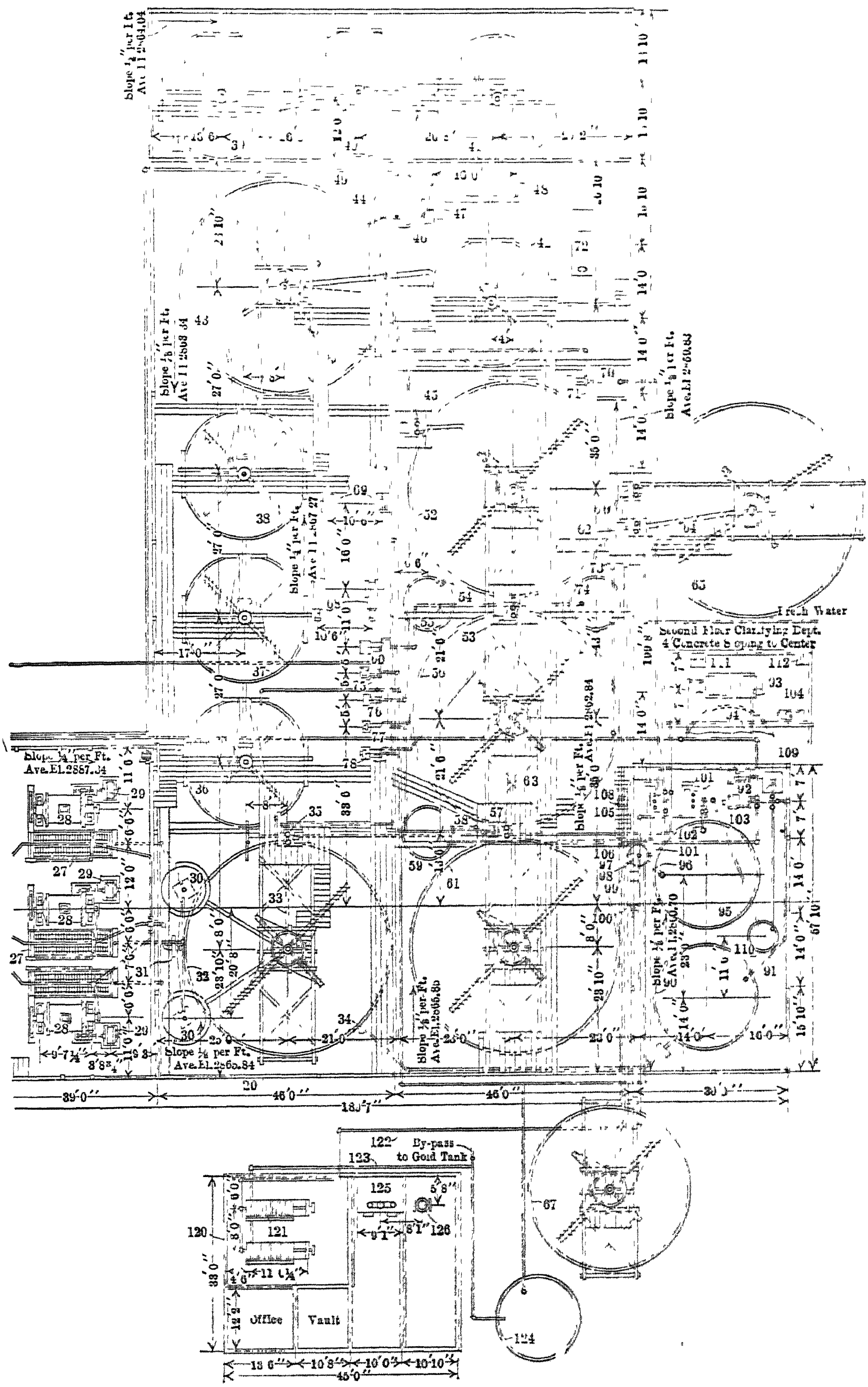



FIG. 7.—(*Continued*)

FIG 7—(*Continued*)

43 No. 2 thickener (like No 1).
44 Overflow No. 2 thickener
45 No 2 diaphragm pump
46. No. 2 sump tank 10 ft. by 6 ft Cap 381 cu ft-2850 gal.-11 9 tons Automatic float controlling pump switch.
47. Two belt-driven centrifugal pumps Suction from No 2 sump tank, discharge to mill-solution tank.
48 15-hp. motor.
49. 3-in pipe to mill-solution tank.
50 Mill-solution tank 18 ft by 16 ft Cap 27,700 gal-116 tons.
51. 3-in pipe to mill.
52. No. 3 thickener (like No 1).
53. No. 3 diaphragm pump
54 Overflow No 3 thickener
55 No. 3 sump tank (like No 2)
56. No. 4 thickener (like No 1)
57. No. 4 diaphragm pump
58. Overflow No 4 thickener
59. No. 4 sump tank (like No 2)
60. 2½-in Krogh centrifugal pump d c. to 2-hp motor, 1700 r p m. discharge to No 2 thickener.
61 No. 5 thickener (like No 1)
62 No. 5 diaphragm pump. Pulp from No. 5 thickener to launder No 64.
63. Overflow No. 5 thickener to No 4 thickener.
64. Launder delivering ⅔ of pulp from No 5 thickener to No. 6 thickener and ⅓ of same to No 7 thickener
65 No 6 thickener (like No 1)
66 No. 6 diaphragm pump (tailings) from No 6 and No 7 thickeners
67 No 7 thickener (like No 1, except size, 8 ft. by 30 ft).
68. Two I-R. compressors 9 in. by 8 in 17 b hp.
69. Two 20-hp motors 12 in. by 8½ in. pulleys, 1160 r p m
70 Cy department floor sump, 6 ft by 6 ft by 6 ft.
71. 2-in Krogh cent pump d. c to 2-hp motor 1700 r p m Delivers to No 2 thickener.
72 Heating boiler, oil-fired 80 b hp. Steam coils in agitators No. 1, No. 2 and No. 3 Sump tank No. 3 and clear solution gold tank.
73 Air lift for No. 7 thickener feed
74 No 6 sump tank (like No. 2)
75 2½-in. Krogh cent pump d c. to 7½ hp. motor 1740 r p m. drawing from sump tank No 2, delivers to mill-solution tank
76. 2-in. Krogh cent pump d. c. to 3-hp motor 1720 r.p m. Drawing from No 6 sump tank, delivering to No. 1 agitator or No. 5 thickener
77. 2-in. Krogh cent. pump d c. to 2-hp. motor 1700 r.p.m. drawing from No. 4 sump tank, delivering to No. 7 agitator
78. 2½-in. Krogh cent pump d. c. to 7½ hp. motor 1740 r.p m. Drawing from barren-solution tank and delivering to No 4 thickener.

CLARIFYING DEPARTMENT

90 Press-solution tank 20 ft by 10 ft Cap 87 tons
91. 4-in suction line to 2-in centrifugal pumps
92. Two interchangeable 2½-in centrifugal pumps (Krogh) direct connected to 7½ hp motors 1740 r p m Drawing from press-solution tank, sluicing-water tank or clarifying department sump and discharging to clarifying press, No 4 or No 3 thickener or mill-solution tank.
93. Merrill clarification press, 3½ ft by 3½ ft, 28 frame.
94. Launder taking press discharge to gold tank.
95. Gold tank, 20 ft. by 10 ft. Cap 87 5 tons
96. 6-in suction from gold tank to vacuum tank.
97. Vacuum treatment tank, 4 ft by 10 ft, removing air from pregnant solution
98. 1-in pipe to vacuum pump
99. Vacuum pump
100 Vacuum pump motor, 2 hp.
101 6-in discharge from vacuum tank to No 1, Triplex pump
102 Auxiliary suction, gold tank to No. 1, Triplex pump.
103 No 1, 7 in by 8 in. Platt Iron Works Triplex pump delivering pregnant solution to refinery
104 Zinc-dust feeder and emulsifier, delivering zinc to suction of No 1 Triplex pump
105. No 2, 7 in. by 8 in. Platt Iron Works Triplex pump, suction connected to draw from barren-solution tank or cyanide department sump.
106 Emergency connection, enabling No. 2 Triplex to draw any agitator or thickener.
107. No 2 Triplex discharge to No. 1 thickener or No 1 agitator or to sluicing jet for clarifying press.
108 15-hp motor driving No 1 and No. 2 Triplex pumps through countershaft
109 Clarifying department floor sump
110 Sluicing-water sump tank, 6 ft. by 8 ft
111. 1-hp motor driving clarifying press.
112 1-hp. motor driving zinc-dust feeder.

REFINERY

120. Refinery bldg concrete block walls, steel roof.
121 Two Merrill precipitation presses, 36 in., 32 frame.
122. 4-in. pipe bringing pregnant solution to presses
123 3-in. pipe from presses to barren-solution tank.
124 Barren-solution tank, 16 ft. by 16 ft
125 Double-muffle Case roasting furnace Use discontinued.
126 Case tilting crucible furnace.

from the presses is returned to the No. 4 thickener, part of it being used as cooling water for the air compressors on the way back.

*Clarification.*—The clarifying filter is dressed with No. 10 duck. One set of cloths lasts about 6 wk. The feed pressure on this filter varies from 8 lb. (3.62 kg.) with a clean press, up to 33 lb. (14.96 kg.) just before sluicing. The press is normally sluiced every 6 hr., using barren solution under 50 lb. (22.68 kg.) pressure. It is very essential that the sluicing bar and nozzles be maintained at a high state of efficiency; a spare bar is always kept in readiness. The cloths become coated with a lime and alumina deposit so that acid treatment is necessary every three days.

A hot solution of 0.9 per cent. HCl is used with a contact of 60 min., the acid being circulated through the press by a 1½-in. centrifugal pump. After treatment, the press and acid-circulating system are drained and washed out, and the acid solution is stored, for future use, in a redwood sump tank.

The sludge from the ordinary sluicing of the filter is returned to No. 3 thickener, as it always contains gold values and cyanide. Ordinary sluicing takes 10 min. and the acid treatment about 90 min. so that the filter normally operates 96 per cent. of the day. Redressing requires nearly 5 hr. This filter can clarify 1000 tons of solution per day under these working conditions.

In both precipitation and clarifying costs, the only operating labor charged is that of changing cloths.

TABLE 12.—*Clarification Cost per Ton of Ore*

| | Operating Labor | Repair Labor | Supplies | Power | Total |
|---|---|---|---|---|---|
| 1917 | $0.0010 | $0.0017 | $0 0197 | $0.0073 | $0 0297 |
| 1918 | 0.0004 | 0.0020 | 0 0337 | 0.0100 | 0.0461 |
| Average | 0.0007 | 0 0018 | 0.0270 | 0 0087 | 0.0382 |

*Clean-up.*—The precipitation presses are dressed with four thicknesses of sheeting. The outside cloth is removed at each clean-up and a new cloth is added on the bottom. By this method no duck or canvas is required and the condition of the cloths as regards "liming up" is maintained nearly constant. When it is desired to clean a press, the solution is cut off and the press is drained and blown with compressed air from 60 to 90 min.; this dries the precipitate to about 45 per cent. moisture. The press is then opened and the precipitate is scraped into the precipitate wagon. The outside cloths, with some adhering precipitate, are removed and burned in a special furnace and the resulting ashes are returned to the undried precipitate.

After determining the moisture content, the undried raw precipitate

is fluxed with 11 per cent. borax glass, 11½ per cent. sodium bicarbonate, 6 per cent. manganese dioxide, 3.3 per cent. ground bottle glass, and at least 10 per cent. of old slag shells from former melts, the percentage being in terms of the calculated weight of the dry precipitate.

A precipitate press ordinarily runs from 5 to 6 days, and yields about 130 lb. (58.96 kg.) dry precipitate. In resuming precipitation after a final clean-up, one press is given the entire flow. When this builds up a pressure of 35 to 40 lb. (15.87 to 18.14 kg.), the second press is opened just enough to maintain the pressure of the first press below 45 lb. (20.41 kg.). When the second press reaches 20 lb. pressure, the entire flow is turned into it and the first press is cleaned. This method is carried on until the end of the month when a final clean-up is made. The solution is metered by a revolution counter on the triplex pump, which is calibrated at intervals with a known tonnage of solution. Solution samples are taken each shift; the heads are a dip sample every hour and the tails a drip sample from the barren flow. The tonnage and the average solution assays give the ounces of gold precipitated daily; this is checked monthly against the bullion sold.

*Melting*.—The fluxed wet precipitate is placed in No. 5 paper bags and fed to an oil-fired No. 150 Case tilting furnace, using a No. 100 long-lipped, graphite pot. When ready for pouring, the pot contains fifteen sacks of precipitate and yields a 600- or 700-oz. button and some 40 to 50 lb. of slag. This charge is poured into a conical mold and allowed to set a few minutes. The slag is then tapped through a hole about 2 in. (5.08 cm.) above the gold button, and run into cold water for granulation. The cold skull of shell left in the mold, which contains most of the shot, is put back with a subsequent charge. The granulated slag, which carries some 25 oz. of gold per ton and as much silver is ground in a small-ball mill and concentrated with a laboratory-sized Diester table. The resulting slag tails from each month's run, of which we usually have less than 400 lb. (181.4 kg.) carries a total value of about $50. This is shipped to the smelter once a year.

An average month's run will show: Crude dry precipitate, 21,000 oz. troy or 1440 lb. avoirdupois; 71 per cent. bullion yield, 14,936 oz. troy or 1024 lb. avoirdupois; 39.7 per cent. gold yield, 8538 oz. troy or 573 lb. avoirdupois; 21.4 per cent. silver yield, 4497 oz. troy or 308 lb. avoirdupois. The crude precipitate contains from 6 per cent. to 7 per cent. of zinc and about 10 per cent. of lead, the latter coming from a lead acetate drip added as the solution leaves the clarifying filter.

The bullion buttons are remelted and cast into bars weighing about 150 lb. (68.03 kg.) each. A dip sample is taken with a 10-gm. clay crucible just before pouring. This sample is granulated in cold water and is sent to the assayer. The bullion, as shipped, has an approximate fineness of 560 in gold and 301 in silver.

The Case furnace consumes 3 gal (11.3 l.) of "27 plus" fuel oil per hour and smelts an average of 14 lb. (6.35 kg.) of precipitate in the same time. The pots, which are heated intermittently, have an average life of about 30 hr. At least once a year the entire furnace lining is removed, crushed, and concentrated, the tailing being fed back to the main mill. The old crucibles are likewise crushed and concentrated, the tailing being sacked and stored. All solution, wash water, etc. from the refinery are filtered and used in the mill. The workmen's clothes are purchased by the company and are burned monthly. Canvas gloves are used daily and are burned with the clothes. The costs of clean-up and refinery like those of precipitation were materially reduced in 1918 by the use of the vacuum precipitation. The zinc content of the crude precipitate was reduced from 30 or 35 per cent. to 6 or 7 per cent., while the bullion content was raised from 54½ to 71 per cent. This increase in metal content allowed the substitution of the No. 100 crucible for the No. 150, as the smaller pot easily yielded a button from 600 to 700 oz., which is about as large as can be conveniently charged into a red-hot furnace without danger of cracking the crucible.

TABLE 13.—*Refining Cost per Ton of Ore*

| | Operating Labor | Repair Labor | Supplies | Power | Miscellaneous | Total |
|---|---|---|---|---|---|---|
| 1917 . . . . | $0 0437 | $0 0051 | $0 0710 | $0.0004 | ...... | $0 1202 |
| 1918 . . . | 0 0444 | 0 0026 | 0 0366 | 0 0008 | $0.0017 | 0 0861 |
| Average . . | 0 0441 | 0 0038 | 0 0530 | 0.0006 | 0 0009 | 0.1024 |

For convenience of comparison, the costs of clarifying, precipitating and refining for 1918 are also given per ounce of gold, and per ton of solution precipitated:

TABLE 14.—1918 *Costs per Ton of Solution Precipitated*

| | Operating Labor | Repair Labor | Supplies | Power | Miscellaneous | Total |
|---|---|---|---|---|---|---|
| Clarification.... | $0 0001 | $0.0006 | $0.0098 | $0.0029 | ....... | $0.0134 |
| Precipitating... | 0 0001 | 0 0007 | 0.0274 | 0.0052 | ..... .. | 0.0334 |
| Refining....... | 0 0129 | 0.0008 | 0.0106 | 0.0002 | $0.0005 | 0.0250 |

TABLE 15.—1918 *Costs Per Ounce Fine Gold Recovered*

| | Operating Labor | Repair Labor | Supplies | Power | Miscellaneous | Total |
|---|---|---|---|---|---|---|
| Clarification.... | $0.0003 | $0.0019 | $0.0318 | $0.0095 | ....... | $0.0435 |
| Precipitating... | 0 0002 | 0.0023 | 0.0892 | 0.0168 | ....... | 0.1085 |
| Refining. ..... | 0.0419 | 0 0025 | 0 0345 | 0.0007 | $0 0016 | 0.0812 |

*Sampling.*—The value of the mill heads is arrived at by the usual method of bullion recovered plus tailing loss. A check head-sample is taken every half hour by dip from the Marcy discharge; this sample is dried and assayed with all its solution. As the Marcy is crushing in solution, which carries considerable value, and as the return circuit might also tend to vitiate the accuracy of such a sample, it can at best be considered only a guide. A drip sample of the storage solution is taken and the gravity of the mill discharge is recorded hourly. From these data, the value of the Marcy discharge sample is corrected and this corrected value is used as the daily mill heads; it checks within 4 per cent. of the heads as determined by tailing plus bullion, and is certainly more accurate than a grab or cut of the coarse mill feed. Dip samples are taken hourly of the various agitator and thickener pulps and solutions. In the pulp sample buckets, 10 cc. of a 10 per cent. solution of sodium sulfide is added each shift; this stops the dissolving action of the cyanide so that the washed pulp shows the actual amount of dissolving completed at the various points. Potassium permanganate, often used for this purpose, was unavailable on account of the war prices. The final or tailing sample is, of course, dried and assayed with all solution present.

The samples are prepared for assay by the solution man on the afternoon shift. The washing is done on a pressure-filter made of 10-in. (25.4 cm.) pipe. The cake is dried on an electric hot plate so that the samples leave the mill all prepared for the assayer. The mill assay sheet is generally completed by 11 A. M., but results on the vital solution-tail samples are obtained at least two hours earlier.

TABLE 16.—*Cost of Sampling per Ton of Ore*

| | Operating Labor | Supplies | Power | Total |
|---|---|---|---|---|
| 1917 | $0 0011 | $0.0034 | $0.0005 | $0 0050 |
| 1918 | 0 0004 | 0.0018 | 0 0009 | 0.0031 |
| Average | 0.0007 | 0.0025 | 0.0007 | 0 0040 |

TABLE 17.—*Cost of Tailing Disposal Per Ton of Ore*

| | Labor | Supplies | Total |
|---|---|---|---|
| 1917 | $0.0225 | $0.0071 | $0.0296 |
| 1918 | 0.0213 | 0.0008 | 0.0221 |
| Average | 0.0218 | 0.0038 | 0.0257 |

*Tailing Disposal.*—The mill tailing, with a moisture content of 46 per cent., runs by gravity to the ponds or dams, through 6-in. slip-joint pipes set with a minimum grade of three degrees. Two men are em-

TABLE 18.—*Cost of Solution Heating Per Ton of Ore*

| | Repair Labor | Supplies | Power | Total |
|---|---|---|---|---|
| 1917 | $0.0005 | $0.0149 | $0.0002 | $0.0156 |
| 1918 | 0.0018 | 0.0416 | 0.0005 | 0.0439 |
| Average | 0.0012 | 0.0288 | 0.0004 | 0.0304 |

TABLE 19.—*General Expense*

| | Operating Labor | Supplies | Power | Miscellaneous | Total |
|---|---|---|---|---|---|
| 1917 | $0.1220 | $0.0308 | $0.0017 | $0.0152 | $0.1697 |
| 1918 | 0.1191 | 0.0470 | 0.0036 | 0.0132 | 0.1829 |
| Average | 0.1205 | 0.0393 | 0.0027 | 0.0142 | 0.1767 |

TABLE 20.—*Lighting*

| | Operating Labor | Supplies | Power | Miscellaneous | Total |
|---|---|---|---|---|---|
| 1917 | $0.0015 | $0.0021 | $0.0043 | ... | $0.0080 |
| 1918 | 0.0025 | 0.0019 | 0.0038 | $0.0005 | 0.0084 |
| Average | 0.0020 | 0.0020 | 0.0041 | 0.0003 | 0.0082 |

TABLE 21.—*Assaying*

| | Operating Labor | Supplies | Power | Miscellaneous | Total |
|---|---|---|---|---|---|
| 1917 | $0.0225 | $0.0115 | $0.0004 | $0.0006 | $0.0350 |
| 1918 | 0.0217 | 0.0094 | 0.0011 | ...... | 0.0322 |
| Average | 0.0221 | 0.0104 | 0.0008 | 0.0003 | 0.0336 |

TABLE 22.—*Metallurgical Report*

| | 1917 | | | 1918 | | |
|---|---|---|---|---|---|---|
| | Tons | Gold, Ounces | Silver, Ounces | Tons | Gold, Ounces | Silver, Ounces |
| Ore milled | 84,548 | 89,163.878 | 45,607.06 | 92,339 | 100,903.079 | 54,137.84 |
| Bullion recovered | ...... | 85,918.424 | 43,989.44 | ...... | 97,827.459 | 52,485.38 |
| Tailing loss | ...... | 3,245.454 | 1,617.62 | ...... | 3,075.620 | 1,652.46 |
| Recovery, 96.36 per cent. | | | | Recovery, 96.95 per cent. | | |

ployed on impounding tailings, part of their time, however, being used to scrape "caliche" or higher grade crust from the surface of the dams, and part to clean presses in the refinery. The items given in Tables 19 to 22 complete the cost analysis per ton of ore milled.

*Solution Heating.*—For about 5 months in the year, the mill solutions are heated by steam coils in the various sumps. The heating plant consists of a 75-hp. Marion horizontal tubular boiler, burning 7 gal. per hr. (26.49 l.) of heavy fuel oil. No special attendance is required other than to blow flues when necessary. Steam is generated at 25 to 30 lb. (11.3 to 13.6 kg.) pressure and the coils return their condensation to the hot well. The costs of heating are distributed over the entire year. Those for 1917 are low as no heating was used during the first half.

## DISCUSSION

LUTHER B. EAMES, Pueblo, Colo. (written discussion*).—In reading Mr. North's interesting paper, several points have been noted that appear to warrant discussion. With a feed all through 20 mesh, I feel inclined to question the use of steel balls as large as 2 in. in a tube-mill. I have seen balls as small as ¾ in. used for the dry grinding of cement clinker, and surely many more points of contact would result from smaller balls This would seem to be desirable in a mill as short as 6 ft. The heavy load in the classifiers would also indicate a high proportion of returned pulp. I hope that some discussion by Mr. North and others may develop this point.

The flow sheet shows the solution returning from the dam valued at 40 c. per ton being added to the last decantation tank. Theoretically, this solution should not be added to that of a lower value. As I calculate it, if this solution were added to tank X the solution finally discharged from tank Z would be reduced in value by nearly 4 c. per ton.

I think Mr. North's explanation of the difference between the actual and the theoretical columns in Table 10 is undoubtedly correct; namely, that dissolving is still taking place very rapidly in the underflow of tank U. It has occurred to me that with so finely ground a pulp as this, it might be possible to connect two of the seven agitators in between the Callow cones and tank U. This would require these two tanks to handle a very thin pulp but would, if successful, tend toward a much closer resemblance between the actual and theoretical columns. It would also give the remaining five agitators the advantage of doing their work in a lower grade of solution.

*Received Aug. 20, 1919.

G. M Taylor,* Colorado Springs, Colo.—The ore treated at the United Eastern is primarily a gold one and is cyanided—the counter-current decantation method being employed. One of our employees visited this plant a few months ago and, on his return, reported that the moisture in the tailing was carrying 20 c. per ton, which would appear to be somewhat high. As this type of plant has been advocated by one of the recognized cyaniding engineering firms in the country, and a number of these plants are in successful operation, it does not seem to me that the dissolved loss could be so high, and I would like to know if any one can give any definite information on this point?

J. V. N. Dorr, New York, N. Y.—I am very glad that Mr. Taylor has raised this question. When I first read Mr. North's paper and saw his statement, that they had an actual dissolved loss of 21.5 c. per ton, I was very much puzzled at this high loss which appeared avoidable by another decantation or the addition of a rotary filter. Reading it again, however, I noted that the actual dissolved loss of 21.5 c. given was evidently obtained by multiplying the assay of solution discharged from *Z* thickener by the tonnage. On the basis of 283 tons milled per day, this means $56.54 going *to* the dam. He states, however, that he returns *from* the dam into the system 70 tons daily assaying 40 c. per ton or $28 per day, so it would seem reasonable, unless I have overlooked something to regard the actual dissolved loss as $56.54 less $28 or $28.54, equaling 10.8 c. per ton. Even this loss, however, is much higher than good practice elsewhere, and can be accounted for largely by the tonnage being handled, which I believe is 30 per cent. above the capacity planned.

The fact that all the thickeners but the last step discharge at about 52 per cent. moisture, instead of 45 per cent. as calculated originally, makes a great difference in the washing efficiency. The dissolution of 36 c. additional in the decantation system after the agitators have made all the extraction they can, has also contributed to raise the dissolved loss. It emphasizes, I think, the statement that has been made, that in many cases the additional amount dissolved in counter-current decantation will more than pay the cost of operating and the entire dissolved loss, so that if instead of this system the agitators were followed by filters that removed 100 per cent. of the dissolved gold at no cost, the net earnings would be less than now.

L. D. Mills, San Francisco, Calif.—I should like to ask Mr. Dorr if he can tell us, in referring to the diagram on page 558, why the solution is returned to the tank marked *X* instead of corresponding to the usual practice and being returned to the tank marked *Y*.

* Manager, Milling Dept., Portland Gold Mining Co.

J. V. N. DORR.—The addition of barren solution to $X$ instead of to $Y$ should theoretically reduce the mechanical loss of cyanide about 30 per cent. On the other hand, the diluting effect of the barren solution in tank $Y$ is lost, and thus the dissolved loss in gold is higher. It is safe to assume that the management has studied the balance carefully and is using the barren solution where it will produce the best results. The return of the 40 c. solution from the dump to $Z$ must have a bad effect on the dissolved loss, but must have been considered and it undoubtedly represents the best practice in recovering the gold with the equipment available.

# Graphic Metallurgical Control

BY H. M MERRY,* HURLEY, N. MEX

(Chicago Meeting, September, 1919)

THE graphic methods and records described in this article have been developed, with satisfactory results, for the use of executives of the Chino Copper Co., in Hurley, N. Mex. Particular attention is directed to the use of large wall-charts, the quick reference display of large charts, the scale notation of metallurgical charts shown in Fig. 3, and the inclusion of mesh extraction on screen analysis diagrams.

To properly comprehend the operation and metallurgical results from a concentrating plant, a daily record of essential data is kept, usually involving the dry weight of ore milled or treated, dry weight of concentrate or metal produced, assay values of heading, assay values of tailing, assay values of concentrate, and percentage of extraction or recovery. In addition it is often necessary to record moisture, pulp density, or dilution, percentage of weight of oils and reagents employed; screen analyses of grinding conditions, oversize, etc.; assay values of non-valuable or interfering substances; cost data; and such special information as may be required.

## Graphic Presentation

Close analysis is necessary of a daily operation involving such a number of factors, the relation and influence of which may be of great importance in the treatment scheme. When the analysis is made from a numerical tabulation, the chance for one or more obscure, but perhaps important, factors to escape notice is considerable, therefore the need for an accurate, readily digestible presentation of metallurgical data is obvious.

Graphic records of milling operations are frequently used, but owing to the wide range of magnitude and values, and the number of factors involved, the customary practice of assigning different values to the same ordinates, as in Fig. 1, results in a complex and intricate record which makes little appeal to a busy operating official. The methods shown in Figs. 2 and 3 are devised to display such data in a clear and legible manner, without confusion.

* Metallurgical Statistician, Chino Copper Co.

*Semi-logarithmic Method.*—This method, Fig. 2, depends on the use of cross-section paper with logarithmic horizontal ruling and arithmetic vertical ruling. Time is laid off on the vertical lines and values are assigned to the horizontal logarithmic scale. Three complete logarithmic scales are required to embrace the range of percentage and tonnage con-

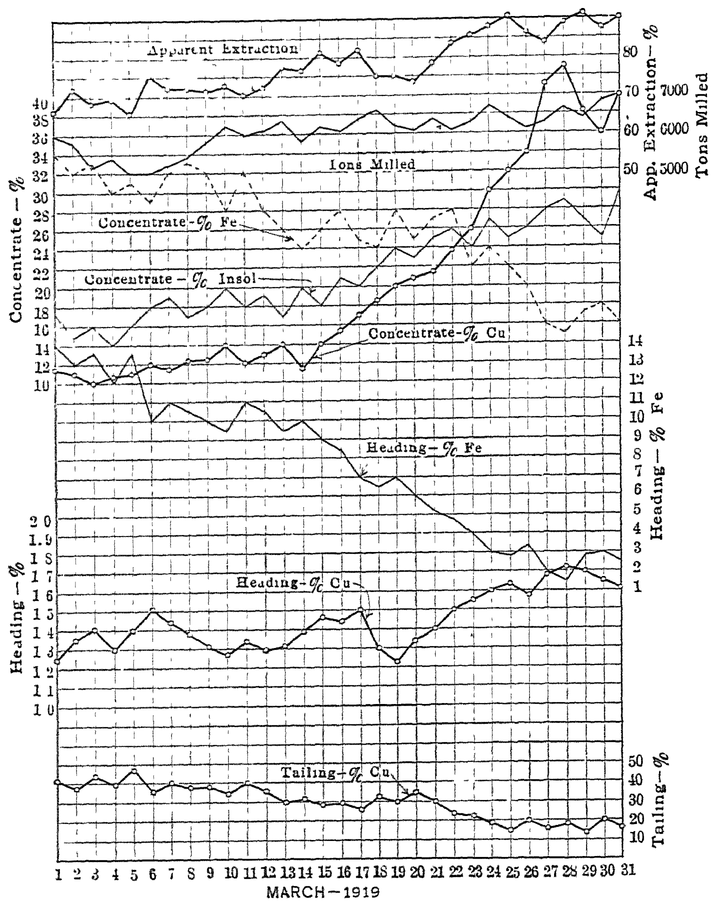



FIG. 1.—DAILY METALLURGICAL RECORD, TYPE I.

sidered in the average milling operation. Four scales are preferable, but such ruling is not available unless hand-drawn.

The graduation of the scales on the record, shown in Fig. 2, is self-explanatory and any one familiar with the slide rule can grasp the method. By this scheme all factors are shown in their proper relative position and tendencies may be closely observed and compared. The only disadvantage in this method lies in the fact that high points are depressed and low points exaggerated, which sometimes leads to erroneous conclusions.

For this reason it is more applicable to the needs of the engineer than to general use The great advantage obtained is the presentation of wide variations of magnitude in a small space, a range of from 0.1 to 100 in direct sequence being possible, and without confusion, the range may be increased to 10,000 in the space of a letter-size sheet.

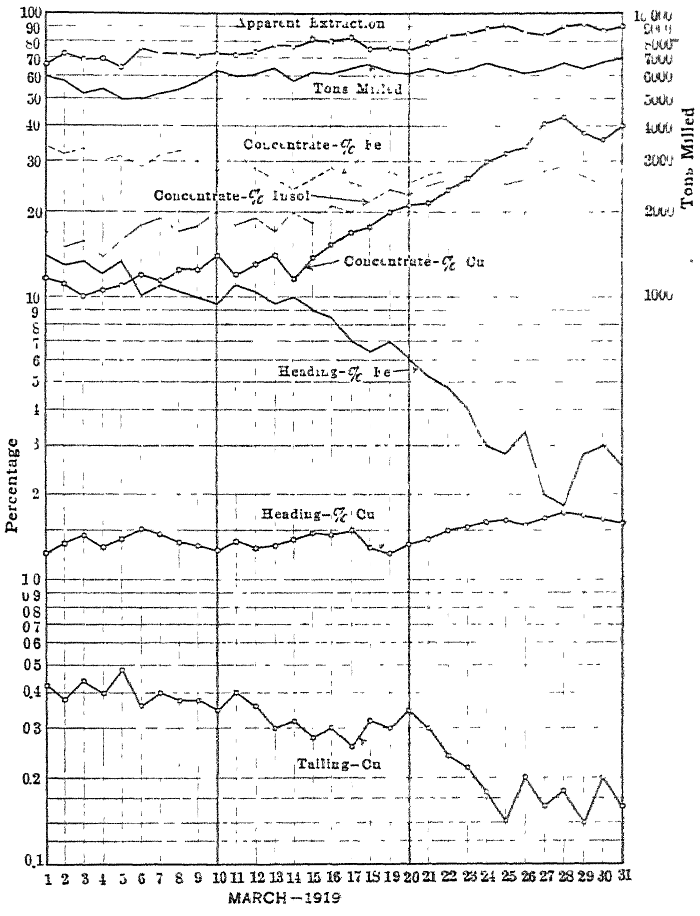



FIG. 2.—DAILY METALLURGICAL RECORD, TYPE II.

*Special Arithmetic Method.*—Millimetric ruled cross-section paper, available in 50-yd. (45-m.) continuous rolls 22 in. (55.8 cm.) wide, is used preferably for this method, Fig. 3, which might be described as logarithmic notation on an arithmetic scale. This notation is original and is of marked advantage in plotting metallurgical data. Sheets 4 ft. (1.2 m.) in length, of full width, are recommended. The notation of the scale should

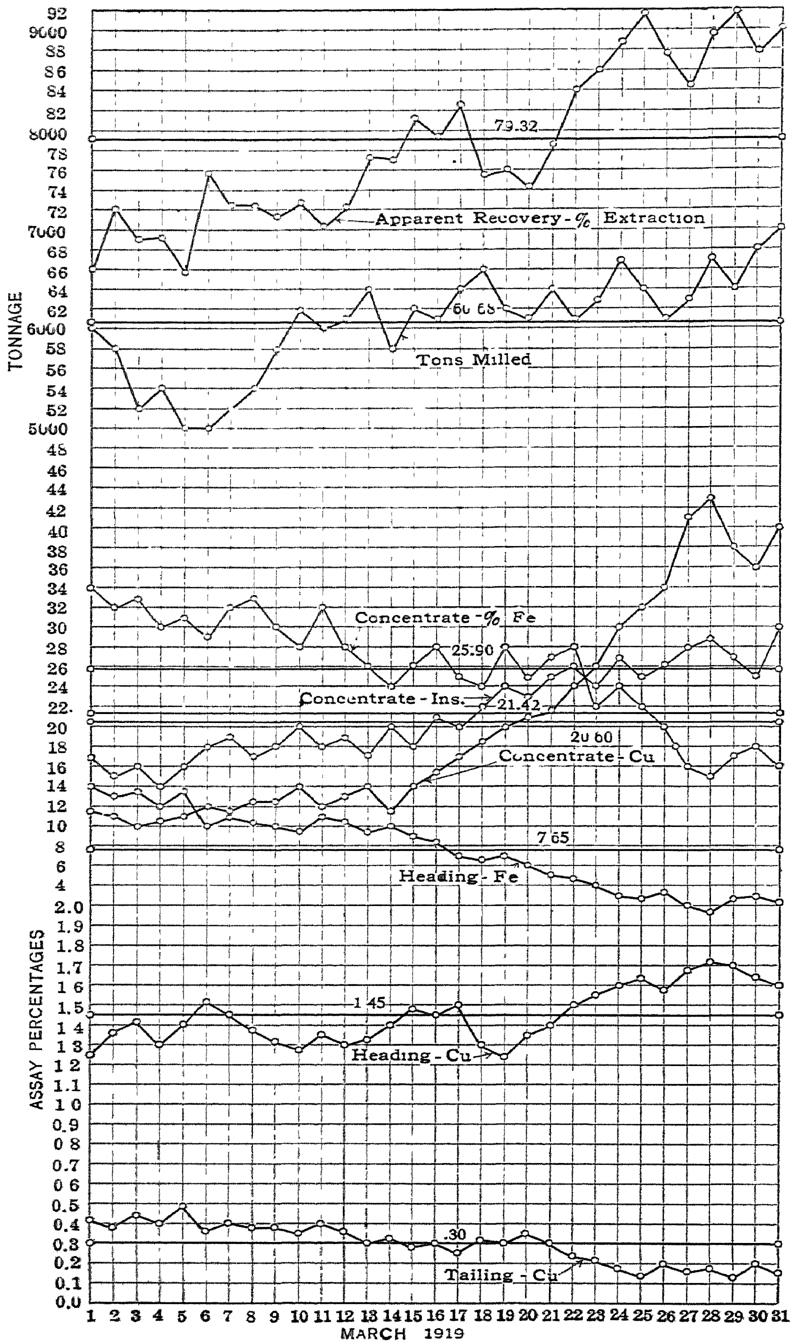



FIG. 3.—DAILY METALLURGICAL RECORD, TYPE III.

be so arranged as to display the various factors with due prominence. Assay values of heading and tailing which are the minimum and also the governing factors, should be exaggerated. Beyond the maximum range of heading assays of valuable metal, the scale may be reduced, the concentrate assay values, dilutions, extraction, etc., falling principally in this range. The tonnage scale is devised to meet the maximum and minimum variation and may include concentrate production. It is permissible to overlap the tonnage and percentage scales, preferably in multiples of ten, employing lines of different weight or color to distinguish them in conflicting areas. Abnormally high or low points that fall outside the limits of the particular scale should be plotted on the projection of the scale in which they originate, since breaking scale destroys the proportion of the relationship with other lines.

Under date of Mar. 28, Fig. 3, the position of all points except that representing per cent. of iron in heading needs no explanation, as they are referred directly to the notation on the scale. The iron, however, falls below 2 per cent., at which the descending scale breaks from a rate of 2 per cent. per centimeter to 0.1 per cent. per centimeter. If the point is plotted to the scale notation it will be out of proportion to all other similar points, so the 2 per cent. per centimeter scale is projected downward to 1.5 per cent. and the point plotted. On the scale notation it appears at 1.95 per cent., but by referring to the origin of the scale in which all other live points are found, a wrong inference should not occur.

The average line drawn through each curve is the average "per day" for the month, and not the average for the month compounded daily to date, which would appear as a median line on the chart. The three types of metallurgical presentation here shown have been plotted from the same fictitious data, for comparison.

A record of departmental milling costs and general statistics is kept on separate similar sheets, changing the scale from logarithmic notation to direct arithmetic, covering the necessary range of magnitudes.

## Mining Control

Upon sheets of the same size, a monthly graphic record of tonnage of ore mined and yardage of waste stripped is kept with incidental costs of various operations. Corresponding deep-mining data can be added or substituted with facility. This form is not illustrated as it follows conventional lines and is simple in construction.

## Wall-charts

Charts of the type shown in Fig. 3 are particularly adapted to large-scale wall-charts for permanent record. Very neat "month by days"

or "3-year by months" records may be kept: time is laid off on the width in days or months, one unit to the centimeter, leaving ample space for marginal notation. Such records, reinforced at the top and bottom with thin clamping strips of wood, screwed together, are easily handled.

For current daily record they are also well adapted to ordinary pin-posting: mounted on a cord and strawboard backing, in which the colored pins may be easily inserted, they are convenient to work upon and may be accurate to a fine degree. Colored connecting cord, to match the glass-headed pins used to represent the different values, adds to the legibility of the chart. When the period covered by the record is ended, the pin-heads are encircled by a pencil mark before the chart is removed from the pin-board. The connecting lines are inked in colors and the record transferred to a cabinet, in which it may be shifted, making it possible to display a number of charts at the same time for comparison.

## Display of Large Charts

The principal reason for the use of graphic records, after their value in analysis and comparison, lies in the rapidity with which the information displayed may be assimilated. This makes them exceptionally useful for reference. Therefore, when an elaborate and comprehensive graphic record is rolled up and filed away, safe from harm and service, a large share of the value is lost. After preparation, such charts must be accessible to yield full usefulness. A visible record cabinet now in use illustrates one way of achieving this result.

This wall cabinet is of wood. Its height is determined by the length of the record sheets, its length by the number of records it is proposed to display simultaneously, and its depth by the number of records it will eventually contain. A pin-board with cork composition backing is built in at one end of the cabinet for the posting of current sheets. An overhead monorail system of the requisite number of tracks is built in at the top of the cabinet, the rails extending its full length. This system is improvised from sliding-door equipment, the track and ball-bearing roller trucks being used. Two trucks in tandem are divided by a spacing bar to the width of the record sheets. The wooden clamps reinforcing the top of the sheets are attached to the spacing bar by a simple lug and thumb latch of brass. The latches on the spacing bars and the lugs on the clamps are laid out to jigs, so all sheets are interchangeable on any track. One sheet only is mounted on each rail. A curtain suspended from a spring roller hangs in front of the cabinet, concealing the contents when desirable. The large sheets are well adapted to photostatic reproduction, and may be reduced to letter size without losing legibility.

## Periodic Reports

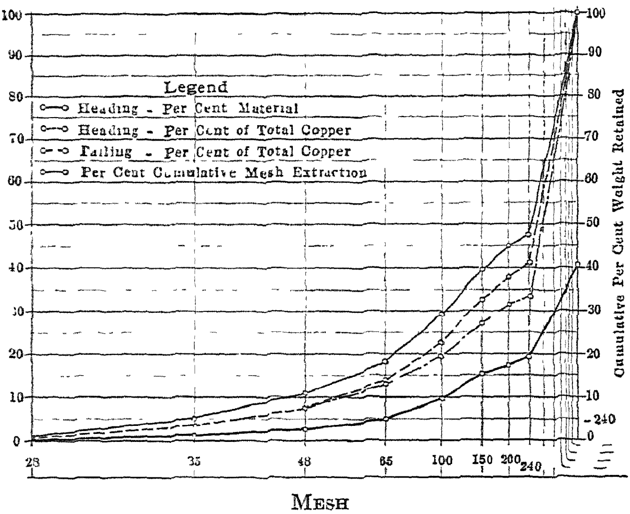
A useful chart for periodic reports is the especially designed form shown in Fig. 4. The cross-section is engraved on thin letter-size tracing paper, on the lower half of the sheet, leaving the upper part for typewritten tabulation. The tracing of the ordinates corresponds to type-

| Month | 1915 | 1916 |
|---|---|---|
| January. | 13.641 | 24 008 |
| February | 14.304 | 26 440 |
| March | 14.787 | 26 310 |
| April | 16.811 | 27 895 |
| May... | 18.506 | 28 625 |
| June | 19 477 | 26 601 |
| July | 18 796 | 23.865 |
| August | 16 941 | 26 120 |
| September | 17 502 | 26 855 |
| October | 17 686 | 27.193 |
| November | 18 627 | 30 625 |
| December | 20.133 | 31 890 |
| Year | 17 275 | 27 202 |


Fig. 4.—Copper market in 1915–16, New York, cents per pound (*E. & M. J.*)

writer spacing, and also to that of the multigraph, so standard forms may be skeletonized in large lots at small expense.

Essential data from the tabulation are plotted on the cross-section. This form is of special use in the presentation of progressive reports, in which summarized periodic figures are carried from month to month for the year, etc. Data for the current month appear in the tabulation, while essential material for past months is plotted. Twelve or twenty-four months may be carried on this form. Many special reports are adequately presented in this manner.




| Mesh | Heading | | | | | Tailing | | | | | Extraction | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Assay | Per Cent. Weight | | Per Cent. Tot. Cu. | | Assay | Per Cent. Weight | | Per Cent Tot. Cu. | | Ind | Cum. |
| | Per Cent Cu | Ind | Cum. | Ind | Cum. | Per Cent Cu. | Ind. | Cum | Ind. | Cum. | Per Cent. | Per Cent |
| Original | 1 52 | | | | | | | | | | | |
| On 20 | | | | | | | | | | | | |
| 28 | 1.10 | 0 7 | . . | 0.5 | | 0 72 | 0 7 | . . | 0 5 | .. | 0 2 | ... |
| 35 | 1 09 | 4.3 | 5 0 | 3 1 | 3 6 | 0 88 | 3 5 | 4 2 | 3 1 | 3 6 | 1.1 | 1.3 |
| 48 | 1.05 | 5.8 | 10 8 | 4.0 | 7 6 | 0 77 | 5.3 | 9 5 | 4 1 | 7.7 | 1 2 | 2.5 |
| 65 | 1.19 | 7 9 | 18 7 | 6 2 | 13.8 | 0 78 | 7 1 | 16 6 | 5.6 | 13.3 | 2.6 | 5 1 |
| 100 | 1.27 | 10.4 | 29.1 | 8 7 | 22 5 | 0 63 | 9 5 | 26 1 | 6.0 | 19.3 | 4 8 | 9 9 |
| 150 | 1 50 | 10 6 | 39 7 | 10 4 | 32 9 | 0 64 | 11 6 | 37 7 | 7.5 | 26 8 | 5.4 | 15 3 |
| 200 | 1.47 | 5 3 | 45 0 | 5 1 | 38 0 | 0 71 | 6.8 | 44 5 | 4 8 | 31 6 | 2.1 | 17 4 |
| 240 | 1 75 | 2.6 | 47 6 | 2.9 | 40.9 | 0.64 | 3.0 | 47 5 | 1 9 | 33.5 | 1 7 | 19 1 |
| Pass 240 | 1 72 | 52.4 | 100.0 | 59 1 | 100.0 | 1 26 | 52 5 | 100 0 | 65.5 | 100.0 | 21.3 | 40 4 |

FIG. 5.—SCREEN ANALYSIS—CONCENTRATION BY SLIME VANNERS.

## SCREEN ANALYSIS DIAGRAMS

These diagrams, shown in Fig. 5, are furnished on letter-size tracing paper, by the W. S. Tyler Co., who devised the ratio plotting scale. Com-

ment upon their use is unnecessary, particularly in the analysis of crushing and grinding problems. In the study of performance of various stages and methods of concentration, the value of these diagrams has been increased by adding the mesh extraction. The percentages by weight of material and copper on each mesh in heading and tailing are delineated. The mesh extraction is computed from the actual weight of copper in the material retained on each mesh in heading and tailing, from the form:

$$\frac{\text{Heading} - \text{Tailing}}{\text{Heading}}$$

The work done by the process on various sized material is thus clearly and quickly shown. If desired, the individual mesh extraction may be plotted, either with or without the cumulative total extraction.

Monthly records of stage concentration have been kept in this manner over a considerable time and are a valuable index to operating conditions. In the comparison of qualities of the many new types of concentrating equipment frequently tested in a large plant, this scheme is especially useful.

In conclusion, it may be said that the graphic records outlined above, modified or amplified to suit special conditions, have facilitated both reference to past performance and analytic study of current operations at this plant, well justifying their adoption.

# The Contract Wage System for Mines

BY A. K. KNICKERBOCKER, CUYUNA, MINN.

(New York Meeting, February, 1920)

PRACTICALLY all underground work on the Minnesota iron ranges is done by miners working on a so-called contract wage system. This system, while it has certain advantages over the straight day's pay or company-account method of wage payment, has serious and vital defects. In view of the scarcity and demands of labor, it is important that employers of labor scrutinize their systems of wage payments carefully, not only to save losses and trouble, due to labor disturbances, but also to secure the greatest efficiency of their workmen and reduce their operating costs. It is admitted that an adequate, just, and equitable wage is the prime factor in bringing about these benefits.

The wage of labor and its method of application and payment have been, and always will be, the main factors in the unrest and dissatisfaction of labor. Not only does labor want and need an adequate wage, assuming that work is adequately performed, but it wants and is entitled to a just and equitable wage. Labor must have respect for the fairness of the wage and for the fairness of the management in the application of the wage It is conceded that the Minnesota iron miners are now, and have been, in the main paid adequately, but the inherent defects of the so-called contract system prevent any just or equitable application of this wage to the various units of the labor body.

The defects of the contract system, with its obvious inequalities, favoritisms, and injustices, are cancers in the brain of labor. Every thinking miner realizes these defects and knows just what they are; the unthinking do not realize the details of the subject, but each month they are forcibly reminded of these defects through the pay envelope. As practiced on the iron ranges, however, the so-called "contract" system is not a contract and the name is a misnomer. No true contractual obligation exists.

As usually practised, the contract system involves the setting of a price per foot in development work, or per foot or per car, usually the latter, in slicing or stoping, by the mining captain, and assumes its acceptance, though often under protest, by the men. A price is set for each working place, in which there are usually two men per shift. This price includes labor and supplies, such as explosives, fuse and caps, shovels, carbide, etc., and the men divide the total earnings less the amount expended for supplies. The cost of the supplies charged to the men varies

in different mines; in some cases supplies are furnished free. In general, the companies furnish machine drills, hose, steel, picks, rails, cars, timbers, boards, lagging, perform some of the transportation and all of the hoisting, and may do some of the track work. The miners do all of the shoveling, drilling, blasting, timbering, etc. and some of the transportation. Supplies chargeable to the men are usually furnished at cost, plus 10 per cent. for handling.

In general the rate per foot or per car, or the contract price, is set at the first of each month and governs the month. Earnings are frequently calculated and checked by the mining captain, and many companies also figure the rates on the fifteenth of each month. Some of the contracts will show large earnings, others average earnings, and others small earnings. When the earnings are small, the rate has been insufficient, in which case it is raised, and a new rate governs the remainder of the month; or the contract has encountered unusual and unexpected difficulties, in which case the contract may be closed and the men paid off at the company-account rate and a new contract started, or they are allowed footage or cars not actually earned; or the men are poor workers or unskilled miners, in which case they are paid off at the company-account rate and discharged, new men are hired, and a new contract started.

## Difficulties of Applying Contract Wage System

The difficulties of the system arise from:

1. The setting of the price, often under protest from the men, by one individual, the mining captain. This price is set to the best of his ability and is based on his mining experience, his knowledge of the men and of the ground, and his opinion of the probable conditions that will present themselves through the month. It is impossible for one individual to set prices equitably throughout the mine. Hard or soft ground, heavy conditions or the reverse, and other difficulties, due entirely to the nature of the work, may be encountered so that the price set by the mining captain is a mere guess.

The human-nature problem enters also, oftentimes unconsciously. For instance, if two gangs of miners, one composed of good men and the other of extra good men, are working under exactly similar conditions, the captain will be tempted to give a slightly lower price per car to the better gang to prevent its "running away" or securing an abnormally high wage, thus penalizing the better gang or subsidizing the poorer, depending on the way in which the matter is regarded. The captain excuses this practice, to himself and the miners, by fancying some difference in conditions in the two places. Where he is unable to so fancy a difference, he must pay the same rate in both cases; but in that case the rate will

be reduced, thereby reducing the earnings of both contracts simply that one shall not make an abnormally high rate.

Furthermore, favoritism develops another result of the human-nature element. Certain favored, though no doubt hard-working and efficient, miners will receive the best places and get the highest wages month after month. In addition, the rule of seniority is not often practiced, and is as often overlooked as not.

2. The difficulty in foreseeing conditions underground 30 days in advance and the multiplicity of operations entering into the contract price make the off-hand setting of this price by the captain a mere guessing matter. The ground may change; it may get heavy, be soft or hard and vary within short distances; water may increase; poor timber or heavy ground may cause extra timbering; no two tramming distances are exactly alike; the ore may be heavy or sticky or vice versa; and numerous other factors enter to influence earnings. Do all these factors influence the price, each in its proper proportion? They surely do not; but they should, if labor is to receive pay for work performed. That they do not is no reflection upon the mining captain; he does the best he can, and nobody attempting to make contracts in this guesswork fashion could do any better.

3. Certain other difficulties, by no means unimportant, arise from the nature of the piece-work system itself, and are common to the system as applied in any industry. In many manufacturing plants, the operations of which are standardized and are performed in a routine manner day by day, the same objections apply. The principal disagreeable factor is rate cutting, after a rate has been set and earnings have been found to be abnormally high. In the mining business, it works somewhat as follows, assuming that a certain gang or contract has started a new place and that the rate has been set by the captain: The rate will represent his best judgment of what the gang will produce as against his idea of the wage he would like to pay them. This rate may not necessarily coincide with the miners' conception of a fair rate. They are often as experienced and possessed of as good judgment in this matter as the captain. The captain's mere assumption that they will be able to make normal earnings is not necessarily conclusive to them and what they think of the situation, not what the captain thinks, will most probably govern their actions.

After the captain has set the rate, one of three things is bound to happen.

1. The miners consider the rate satisfactory and go ahead and do a good month's work.

(*a*) If the rate happens to be satisfactory, they receive a good month's pay and everybody is satisfied. The company has received good production and the miners have had proper pay; this is the ideal condition.

(*b*) If the rate happens to be high, the miners will make an abnormal

wage. This is a frequent occurrence and has a pernicious effect; here, the company loses money on the men. In the first place, the miners who have done a good month's work but who have received an average or below an average pay, do not consider the receipt by this gang of an abnormal wage as at all just; therefore a certain proportion of the force is immediately dissatisfied, with the usual results of demands for higher rates and the uncalled-for quitting of men. Also, as a rule, the rate of the high-wage gang is cut for the succeeding month with the result that these men are dissatisfied and quit or work in a half-hearted fashion.

(*c*) If the rate happens to be low, the miners will make a low wage. Here, the company makes money on its men, if it looks no farther than the earnings of the men and the tons of ore produced by them. In reality, it is as much of a losing proposition for the company as either of the others. The men consider that they have worked as hard through the month as any of the other men, so the result is a dissatisfied gang.

2. The miners consider the rate too high, regardless of whether it is or not and so will not do their best work. They think that to do so will mean that their earnings will prove abnormal and thus lead to rate cutting at the month's end. They, therefore, fail to produce a normal output until such time as the captain discovers this fact and attempts a remedy by either discharging men or setting a new and lower rate.

3. The miners consider the rate too low and either quit or do not give a full measure of work. The less work they do, provided they are not discharged, and to some extent even if they are, makes it more certain that sooner or later the captain will raise the rate. Miners cannot be forced to work, and the captain must avoid low production. If his gangs refuse to work and give him production, he has two remedies. He can discharge the men or he can stimulate them by raising the rates. What he will do in any instance depends on how widely the dissatisfaction is spread through the mine and on the character and supply in the labor market.

On the Minnesota ranges, conditions in one mine are not radically different from those in others, yet there is a constant migration of men from mine to mine—an economic loss to both the workers and the employers. Not only is the system economically poor in its effects on labor and on the employing companies, but the influences of the injustices of the system on the mind of the worker make him a fruitful field for the agitator. The system is a constant trouble breeder and mischief maker. It allows a continual heckling and disturbing of labor and it will continue to be the prime factor in labor unrest.

## Suggested Improvements in Contract System

As a matter of fact, the average miner does not want the system abolished. He prefers it over the day's pay system, for it allows him to earn,

by the expenditure of effort, a wage greater than the company-account wage. He knows that to go back to the day's pay system would be, in effect, a general reduction in the level of wages. What the miner wants is a contract system based on something more tangible than the autocratic judgment of one individual.

The keynote of the improved plan is standardization, based upon accurate and sympathetically carried out time studies. No piece-work wage payments can be made satisfactorily unless each operation is standardized and the time and cost of doing it are definitely known. In other words, we must have, instead of rates set upon individual autocratic judgment, rates set scientifically and based upon long-continued and accurate time studies.

The business of mining iron ore is not simple, and to handle the matter we will have to divide it into simple operations capable of definite time and cost ascertainment, and classify these operations into two classes—fixed operations and variable operations. The fixed operations are: Shoveling, drilling and blasting, tramming, timbering, boarding up and blasting down rooms, tracklaying. These are all fixed determinable factors that, by means of accurate and long-continued time studies, based upon the work of average miners, would give an easy determination of the proper pay. These time investigations would prove, for instance, that the average miner can shovel a certain size of car of ore for $X$ cents, which amount should then be the price paid for this work all through the mine, until such time as the general level of wages is changed.

Likewise with drilling and blasting; although experience would point the way as to the proper method of handling this question. Whether this work should be paid for at so much per foot or per car or ton, or per foot of hole drilled, is a matter that could be worked out at the proper time. This item is also a semi-variable, in that the several kinds of ground must be considered. To handle this successfully, the number of kinds must be kept at a minimum. Five grades are suggested, very soft, soft, medium, hard, and very hard. This is the only item where the judgment of the mining captain is brought into play, or the only chance there is for any disagreement or argument between the men and the captain. Surely the captain should be able to classify any face into one or the other of these five grades, especially as the grades will have been well delineated by the preliminary time study and investigation work. The system should be worked so that the rate on this item can be changed during the month as often and as quickly as the ground changes from one class to the other.

Similarly, tramming should be paid for at so many cents per various sized cars trammed 100 ft. or fraction thereof. Timbering would be paid for at so many cents per set, based on a certain length post and cap. The

rate would raise or lower with a change in the post or cap length, or both, above or below the base lengths. Boarding up rooms would be paid for at so many cents per square foot or hundred square feet boarded, and blasting down of rooms would be paid for per 100 or 1000 cu. ft. of room blasted down. Tracklaying would be paid for at so many cents per linear foot for straight track, so many cents per foot for curved track, a lump sum for switches, frogs, tracks into slices, etc. Since in most mines a good share of the track work is done by a special track man this item would not be very important.

The variables entering into the problem are, unhappily, numerous, but the crux of the whole problem lies in the treatment of these variables. The variables include all those items that cause a loss of time to the miner and prevent him from using his time in one or the other of the above six fixed operations. These delays should be classified into two well-defined classes: those for which the miner is responsible and which could have been avoided by ordinary mining skill or foresight on his part, and those that are in no way under his control and are due to neglect of the company or accidents due to the nature of the work, such as unforeseen caves or falls of ground. For the first class, the miner should receive no pay. For the second, he should automatically go upon company-account pay the minute that he starts to suffer delay and be paid for all the time he is so delayed. This company-account rate should be made low so that advantage will not be taken of it, and also because there is presumably little or no expenditure of effort in most of the cases coming under this classification.

The contract sheet at the end of the month would then look something like this:

| | |
|---|---|
| To shoveling $A$ cars, $A$ size, at $a$ cents per car.......... | ——— |
| To tramming $A$ cars, $A$ size, 300 ft., at $b$ cents per car per 100 ft.......................................... | ——— |
| To drilling and blasting, soft ground, at $d$ cents per car (or foot of drift)........................................ | ——— |
| To timbering, $E$ sets, 7-ft. post, 6-ft. cap, at $e$ cents per set.. | ——— |
| To boarding up rooms, $F$ sq. ft at $f$ cents per 1000 sq. ft.... | ——— |
| To blasting down rooms, $G$ cu. ft. at $g$ cents per 1000 cu. ft. | ——— |
| To $H$ hours delay, waiting for full chute, at $x$ cents per hour | ——— |
| To $I$ hours delay, blasted down sets, no pay. | |
| To $J$ hours delay, no air, at $x$ cents per hr................. | ——— |
| To putting in $K$ props, at $k$ cents per prop................. | ——— |

Supplies could be handled in the usual way and figured in the standard rates, or supply standards could be adopted, supplies furnished free, and savings made by efficiency on the part of the men divided between the company and the men, on an equitable basis.

The plan as outlined is still a straight piece-work system. Whether

it would be advisable to develop a standard day's work and standard day's pay and encourage efficiency by the installation of some sort of bonus payments is a matter that would develop as the system was worked out and put into practice. The introduction of bonus payments would be a complication to avoid until such time as the improved piece-work method has received a good trial, become stabilized, and found favor. It is believed, however, that an attendance bonus, or the granting of a shift or half-shift per month for full or nearly full attendance during the month, would help to keep the working force full each day and is an application of the bonus system that might be instituted immediately upon the adoption of the improved piece-work system.

A little more clerical work would permit of the calculation of the earnings of each contract each day, with a daily posting of earnings in the change house, where they could be observed by the men. As a stimulation to a man to put forth increased efforts, there is nothing like an idea of what his present efforts are actually bringing him and an immediate knowledge of his earnings each day. Whether such a step is advisable or will pay for itself is open to experiment after the system is in proper working order.

## Disadvantages of Suggested System

The disadvantages of the suggested system are, of course, items of cost, due to the extra supervision and time keeping entailed and the necessity for frequent check-ups by the management. In a small mine, the details could be looked after by the shift boss but a mine of any size would require men doing nothing else but checking time, etc., and looking after the operation of the system. It is believed that the extra labor would earn its money, as the operation of the system would be, in effect, a continuation of the preliminary time and cost studies, and it is felt that the system would present facts of operation in such a manner as to quickly uncover the weak spots of an operation and suggest means for the application of suitable remedies. For instance, the daily records would show that some gangs were above the average in certain lines of work and below the average in others; steps could then be taken to discover the gang's difficulties and by education improve the production and earnings of the gang and lessen the cost to the company. Certainly it would mean a detailed and accurate cost system applied where it should be applied, to the man actually producing the ore, and it is inconceivable that the system would not furnish to the captains and superintendents a ready and up-to-date means of detecting weak spots in their operations and determining quickly and accurately the means of correcting them, and thus lead to a general reduction of costs to the limits toward which all are striving.

Another disadvantage is that the system could not be put into operation without some initial and preliminary time studies and investigations performed on average men and under widely varying, though typical, conditions, and under supervision of men sympathetic to improvements. The longer this work is continued prior to the adoption of the system, the more accurate and fool-proof it will be. These investigations can best be made by the large companies, having various mines and with large organizations equipped for the work. The cost would be prohibitive to a small company. This preliminary investigation would no doubt take two or three years' time and cost some thousands of dollars, but the savings made on the millions of tons of ore mined and shipped each year would repay the cost many times over, due to the knowledge obtained even should the work stop with the preliminary investigations.

### Advantages of Suggested System

The advantages are numerous and have previously been indicated, but for convenience will be here summarized briefly: (1) Prevention of disputes; the basic rates for each operation would be standard throughout the district. (2) Less dissatisfaction among the men. (3) Less opportunity for the strike agitator. (4) Better work by the men, due to the fact that they would see that unfairness and injustice had been abolished and would feel that they were getting paid fairly for work performed. (5) A smaller labor turnover. (6) A reduction of operating costs to the companies. (7) Opportunity for increasing the efficiency of the men and the improvement of operating conditions and methods, due to detailed costs presented quickly and knowledge obtained from them.

The suggested system may not be the best, and it is not expected that any fair system can be derived without actual study and experimental trial, but in the interests of both the men and their employers, some steps should be taken to improve the present system.

## DISCUSSION

Arthur Thacher, St. Louis, Mo. (written discussion*).—A careful study of the paper will show that there is a fundamental objection to all piece work. Perhaps the greatest objection is the impossibility of fairly measuring human effort, and without fairness no system can succeed. Success is only possible with justice. Perfect justice is impossible by human minds; but the nearer we can approach justice, the greater will be our success. It is often thought that paying for what a man does will bring justice; it would if we were able to measure accurately what he does but, as the paper indicates, such accuracy is unattainable and thus brings about dissatisfaction of the men. In the simpler

* Received Dec. 29, 1919.

operation, the defects of the system are less apparent and the results seem to be better. All systems where the rates must be constantly changing are bad, for the men soon see that they will not be allowed to make more than a certain amount and there is no incentive to keep up the greatest effort.

In the second part of the paper an attempt is made to improve the system, in the particular case under consideration, by simplifying the operations. While this may be an improvement for this particular instance, it is open to the objections cited in the first half of the paper. This is particularly noticeable in the discussion of drilling, which is divided into five grades, the divisions naturally being dependent on the judgment of the foreman. An attempt is made to determine scientifically what can be done and to arrange the work accordingly. This is a common effort at the present time, as we are too apt to think we can regulate human effort scientifically or by an adding machine. Human nature is too variable, even in the same individual, for it is largely dependent on his mental and physical condition at the time and varies so from day to day. The piece system is as old as the wage system and if it was materially better, it would have displaced the latter.

Success can only be attained by justice and the nearer we can approach to this, in both principle and practice, the greater will be our success. It is not enough that the principles should be just; the applications must be fairly put in a manner to show clearly that the aim is justice, even though individual cases may fail. Full compensation should be given for services rendered, but this can be done by paying full wages and not attempting the impossible task of measuring exactly what each man has done. The same effort and time spent in trying to measure the work done would be better employed in giving the men a better understanding of their work. This may seem impossible, but we have all seen many cases where there is a cordial relation between the men and the employers and invariably this relationship is founded on a sense of justice. That this relationship fails in many instances is more often the fault of the employer than of the men. An employer whose main aim is to get all he can shows this fact to the men, and they meet him on the same ground and try to do as little as they can. To bring out the best for coöperative work, it is necessary to believe fully in justice; this will bring a response from the men, and justice on the part of the employer will bring justice on the part of the men. This may seem ideal, but we must remember that human nature is weak and all good is of slow growth. With the example of the great war and the splendid work of our American men, we should not be discouraged for the future. If our principles are right we are advancing, although we can never attain perfection.

Lucien Eaton,* Ishpeming, Mich. (written discussion†).—Mr Knickerbocker's paper gives a good description of the way this system is handled at many mines. He points out many of the defects in the system and concludes that it is the cause of much dissatisfaction and labor unrest. The history of the contract wage system on the Marquette Range does not bear out this conclusion. I think I am right in saying that this system of payment has been in use on the Marquette Range for a longer period than on any of the other iron ranges, and the labor turnover at the older mines, where the system has been used longest, is as low as anywhere in the Lake Superior region.

It is true that at many mines the term "contract system" is a misnomer, but this is not always the case. At the mines where the system is most successful the procedure is about as follows: On the first of the month the captain goes through the mine and "sets the contracts." He has a small book with him, in which the price per car or foot, the contract number, the names of one or more of the men, the location, and other data are recorded. Usually one page is allotted to each contract. When he goes into a working-place, one of the men will ask what price will be paid for that month and, the captain will tell him. If the men are not satisfied, they ask for more money and the price is settled right there and entered in the book. Sometimes the captain first asks the men what price they want, and then states his views. If the contract is by the foot, the drift or raise is measured twice a month and the footage is entered in the book. If the contract is by the car, the record is made daily from the tally of the trammers on the main level. These figures, together with the price set by the captain, supplies used, etc., are entered in a large "contract book" in the clerk's office, where they can be inspected by the men at any time. The men are paid extra for extra timbering, such as putting in props, lining sets, etc., for which the rate is uniform throughout the mine. If repairing is necessary, for which a contract cannot be let, the men are paid company-account wages while they are engaged on that particular piece of work. When administered as outlined above, this is really a contract-wage system and the men get just what they earn. The application of the system varies, however, from the method just described to practically a company-account system, where no price is set at the first of the month, but the captain decides what wage each gang ought to receive and then makes his calculation backwards to see what price per car or per foot he must pay to reach that wage. This method, of course, is unsatisfactory.

The defects noted by Mr. Knickerbocker are most conspicuous when a new mine is being opened and much development work is being done. After the ore has been properly opened, there should be no difficulty in

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† Received Feb. 16, 1920.

arriving at proper prices per car for stoping, provided that the captain is a capable man and a good judge of ground. After all, the real answer to Mr. Knickerbocker's objections is just one thing, the capability of the mining captain. If the captain is a good man and his records are properly kept, he knows from his experience during development about what changes of ground he can expect and how many cars a gang ought to put out in a shift, and he has a daily record of output before him for comparison with his estimate.

In paragraph 1 under the heading "Difficulties of Applying Contract Wage System," Mr. Knickerbocker says, "It is impossible for one individual to set prices equitably throughout the mine." This is a pretty broad statement; my judgment, based on long experience, is that one person can set prices more equitably than two or three. Further on, in the same paragraph, he deplores the likelihood of favoritism in setting prices and the overlooking of the "rule of seniority." As far as favoritism goes, if there is any, it results in the favorite's getting a company-account job, and the only rule of seniority that holds in contract mining is as to who can put out the most dirt in a month.

In his suggestions for improvement in the contract system, Mr. Knickerbocker recommends a standard price for filling a car and for tramming per 100 ft., various prices per foot for drilling according to the hardness of the ground, and special prices for laying track, timbering, blasting in rooms, etc. Instead of helping the situation these changes would complicate it tremendously and would cause endless disagreements. There is as much variation in the labor required in shoveling as there is in breaking ground; and there is as much variation in the "tightness" of ground as there is in the hardness. In mining, a great many of the factors counterbalance one another. If the ground is hard, less timber is required; if soft, less drilling and usually less powder. Other factors are fairly constant, such as covering down, blasting in rooms, laying and taking up track, etc. It is much easier to size up the situation and set a satisfactory price per car or per foot that will cover all these different factors and duties at once, than to try to separate the work into its different details, and the chances for disagreement are infinitely less.

In considering the plan of paying men company-account wages for unavoidable delays underground, questions immediately arise such as "Who is to say what is an unavoidable delay and what is not? How much time are the men to be allowed? Is the miners' word to be taken for it, or must the shift-boss be present? If the compressor stops, and one gang is drilling, another is timbering, and another is shoveling, are all three gangs to be paid for the delay or only one? Who is to say whether or not the men drilling might not do something else, while there is no air?" And so on *ad infinitum*. No, the only basis for contract mining

is tons of ore produced or feet of drift or raise driven, standards of timbering, etc. being, of course, maintained.

Much has been accomplished and much remains to be accomplished in the standardization of equipment and of methods underground. When methods have all been standardized and the men have learned the standard method, it may be possible to pay standard prices for the separate kinds of work incident to mining, but before that time comes there won't be any labor turnover, for the men will have to stick on the job to learn the standard methods, and our problem will be solved.

Charles E. Locke, Cambridge, Mass. (written discussion*).—I read the paper with much interest and was surprised at the ideas put forth. We have been taught that the piece-work, or contract, system is most satisfactory because the company gets what it pays for and a workman is able to earn according to his ability, so that a good man gets higher wages than a poor man.

The paper presents this question from another angle and shows that, from the workman's viewpoint, the contract system may not be entirely satisfactory. Whether or not the defects pointed out are sufficient to warrant the elimination of the system and the substitution of the day-pay basis is questionable. There is a possibility that many of the defects enumerated may be overcome by a somewhat different method of applying the contract system. For instance, some form of bidding system may be used. In other words, should a company desire to have a tunnel driven, it would submit specifications and ask for bids on the basis of, let us say, one month's duration. This procedure would eliminate any charges of favoritism and of arbitrary setting of rates. The men would bid for the contract and carry it through in good faith. Undoubtedly some difficulties might be met with but they would not be too hard to overcome. For example, if one crew had a good contract one month, the chances are that the next month a lot of other men would bid against it and if the original contractors were underbid they might be dissatisfied. Possibly if the original contractors were given a handicap, say, of 5 per cent., whenever a contract was being reassigned, at the end of each month, this particular trouble might be solved. Other questions that would come up with the introduction of a new form of bidding could be as satisfactorily settled.

A. K. Knickerbocker.—It is true that the suggested system is still a piece-work system and has the disadvantages of all piece-work systems. The probability of attaining the ideal of a mutual interest between employer and employee as a solution of the wage problem is, of course, in the minds of everyone. But this, at best, can be but a process of

* Received Mar. 19, 1920.

slow growth, and doubly so when dealing with large bodies of foreigners who have not been thoroughly assimilated into our social body or versed in our American ideals, and are, perhaps, suspicious or resentful due to injustices dealt them in this or their own countries. The problem for us is to take care of the situation as it is, investigate our methods carefully, and see if there are not things that can be done to correct the trouble. Taking out even a few of the injustices will be a step toward attaining our ideal.

The bulwark of a successful enterprise is a contented worker. Several things go for contentment, but a just and adequate wage is not the least. A just wage contemplates the proper relation between work and wages, embodied in the phrase, "An honest day's pay for an honest day's work." It is apparent that the contract system, as at present administered, does not in all cases properly provide this relation.

Any employer of underground labor on the Minnesota iron ranges can prove to himself that the present system of wage payments is not entirely satisfactory and is costing him money by investigating the tons per miner or the total tonnage hoisted during the last week or the last half of the month as compared to that of the first week or the first half of the month, respectively. The fact that there are contracts in nearly every mine which each month fail to make the company-account wage and are paid off at that wage further indicates that something is not altogether right about the system.

Under the present system there is more or less wrangling around the average mine, which the suggested system would help to remedy. The men on each shift think that the men on the opposite shift do not do their share of the work or that they so arrange things that they do the easiest work and the part that shows most, such as loading cars, etc. Inasmuch as cars produced are the things which the captain watches and on which men are often judged, there is a tendency for men to become known as car producers, even at the expense of their partners. The new system would tend to remedy this feature, as it would give the men and the captain a record of exactly what each shift had done, so that there could be no argument in the matter, and the matter could be carried even further and each shift paid only for work done by it.

It has been suggested that a card, of not undue size, could be used for each man or each contract per day, with the operations listed thereon in such a manner that the recording of them could be done by means of a punch. This appears to be a neat, accurate, and time-saving way of making and keeping the records.

# Method of Curtailing Forces at the Copper Queen

BY CHARLES F. WILLIS,* E. M., BISBEE, ARIZ.

(Chicago Meeting, September, 1919)

THE problem of the curtailment of forces in large numbers does not often come to employment departments and is, therefore, a problem that many departments are not prepared to handle intelligently. Those companies that are able to measure the individual efficiency of each man would probably lay off men according to their individual ability, irrespective of dependents, citizenship, or other considerations.

During the war when labor was scarce, the business of the employment manager in industry was hiring men and carefully placing them where they were best fitted to work. The careful selection of employees had developed to a considerable extent before the war, and was becoming a recognized science. In the last few months, however, the cancelation of copper contracts, the accumulation of large stocks, the necessary curtailment of production because the copper could not be sold, and the lack of any immediate prospects of the copper market opening, have led to a universal reduction of forces, and the problem that industry now confronts is that of discharging rather than employing. The problem has been aggravated by the cancellation of contracts in many war industries with the consequent turning loose of hundreds of thousands of men, a great amount of unemployment, and the return of the soldiers and sailors. The problem has now become one of reduction of working force with the least hardship and the least injustice, and yet caring for the soldiers, sailors, and marines who have been in the service of the government for nearly 2 years, and are now returning to their pre-war positions.

The Copper Queen Branch of the Phelps Dodge Corpn., operating at Bisbee, Ariz., found it necessary, like all of the other copper companies of the country, to reduce its force materially. In making this reduction it was, after due consideration, decided to classify employees with respect to dependents, length of service, and citizenship. The question of individual efficiency or workmanship was not considered. Copper produced to sell at 26 c. a lb. has been selling at under 15 c., and very little copper is being sold at all. Practically every copper producer in the country has enormous stocks on hand, and it would be profitable,

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in the sense of taking no loss, for many to close down entirely. It has been impossible to reduce wages to the basis of the price of copper, for such reduction would bring the wage scale below the cost of living. It would obviously cause hardship to stop the work altogether and would throw out of employment a large number of employees who have worked loyally for the company for a considerable time, and it would disturb the labor situation greatly by increasing the number of unemployed. It therefore became necessary to reduce the output to as low a point as possible and still maintain reasonable costs and retain in the employ of the company men who have put in years of faithful service. In many cases it has not been possible to place these men in actual operating work, so they have been put on development work, in spite of the fact that it could be done more cheaply later on, and that some of the Phelps Dodge properties have development work so far ahead of actual mine operations that this work could be stopped for a considerable period.

As one of the features of its employees' representation, the Copper Queen maintains a welfare committee, composed of nine men, elected annually by the employees, who represent various divisions of the plant and mine. For days this committee met with the superintendent, the foremen, the employment manager, and the service-record clerk, and studied individually the record of every man as to his continuity of service, his dependents, citizenship, and reliability. With the records of every man before them, the result was a careful classification of all employees.

Class 1 consists of all old employees who have been in the service of the company over 2 years, who have dependents, and who are citizens of the United States. This class also covers old employees of the company who are nearing the time when they will be eligible for a pension, without citizenship being taken into account, for it was considered that their pension had no relation to their citizenship, and that it would be an injustice to classify otherwise men whose long service had entitled them to the privilege of retiring shortly with an income.

Class 2 consists of soldiers, sailors, and marines, whether they have been in the employ of the company for 1 month or a period of years. This classification, along with the rule that all honorably discharged soldiers, sailors, and marines who worked for the company before the war should be given employment, even though the forces were being curtailed, gives distinct preference to those men and will greatly relieve the problem of the returning soldier in the Warren District.

Class 3 includes old employees who have worked continuously for over 2 years, who are without local dependents. There are many men working in mining camps who have families elsewhere, and these men are put in the above class provided they are American citizens. Class 3 also includes the younger employees who have worked continuously

since Jan. 1, 1917, and who have dependents. In this class also are placed old employees who have dependents, but who are not citizens.

Class 4 covers the men who have worked less than 18 mo. and who have dependents and citizenship. It also includes some Austrians and Germans who had made the first move toward securing citizenship, but who were deprived of the right to make their final application during the war, provided they had dependents, and had shown evidence of desiring citizenship.

Class 5 includes those employees without dependents who have come on the company payrolls during the past 18 mo. It also includes non-citizens with no local dependents; in some cases there was a question as to dependents, as well as citizenship, and these names were placed in a higher class pending investigation.

The men are laid off in the reverse order of the classes, class 5 being first, then class 4, and so on. With the announcement of the system to be used, it became apparent to the workmen that even before class 4 might be exhausted the organization would be top heavy with overhead, and while no announcement was made on the subject, men classified in any of the first four classes have felt a sense of security of employment, a feeling that as long as the copper mine was going at all, their work was assured. Previous to the announcement of the system, knowing that curtailment was to take place, there was a general feeling of unrest, of wondering whose turn would be next. The men who felt that they had been wrongly classified have been given the privilege of appealing to the employees' welfare committee for re-classification.

It would be hard to state definitely the results of this system, for it is not known how much the conditions existing in the Warren District can be attributed to it, but certain things have been very apparent. Unemployment has not been particularly visible in the district. Inasmuch as practically all of the men laid off were single men, they left the district at once for other places of employment. There has been no serious effect upon business, for the curtailment did not cut down the population of the district to any appreciable extent. With the announcement that forces were to be curtailed, the local relief association prepared for more active work, as it was felt that among the foreigners, especially, there would be great need, but this has been found not to be the case; the association has few more, if any, to care for.

Another noticeable condition is the attitude of the workmen toward the lower wage scale. The Warren District has been one of the few districts in the West in which the cutting of wages and the curtailment of forces was accepted as unavoidable, and where the men recognize and appreciate that the wages were not cut to the extent that the price of copper warranted. Had the lower wage scale and large curtailment of forces been accompanied by large numbers of idle men, the suffering of

families, and the usual accompaniments of unemployment, it is probable that busy tongues might have caused considerable discontent.

The method outlined, with its apparent justice, is undoubtedly one of the reasons for the stability of the labor situation in the Warren District, and evidences the desire of the company to give a square deal.

## DISCUSSION

The Chairman (F. K. Copeland,* Chicago, Ill.) —At this particular time conditions existing in this country, and elsewhere, make all questions of milling or smelting or mining, or anything else, absolutely secondary to the one question of labor. We have always devoted our attention largely to the efficiency of machinery, and to reductions in costs, etc., and have let this tremendous problem of our relation to labor more or less take its own chances. There has never been a time when those who are responsible for the operation of business of different kinds were as completely at a loss as they are now, so that anything that can be brought out that relates to labor is, to my mind, of utmost importance.

T. T. Read, Washington, D. C.—This formulation of a method of curtailing forces at the Copper Queen is a notable step in advance of our treatment of personnel in the mining industry. So far as I know, it is the first case where a company has bound itself to follow a definite method, although the general idea has been practised in many if not most instances. Here the company has definitely formulated a method by which it binds itself and recognizes the obligations existing when it brings a man into a district, causing him to settle there and acquire property, real or personal. Under such circumstances, the company is in part responsible for seeing that, so far as possible, he shall not suffer a loss through unemployment. The man who goes into a district to engage in a definite type of work, for which there is there no other outlet, and who works for a company a long time has established a relationship with that company which has not always been clearly recognized. If he is no longer wanted in that company, for even the most excellent reasons, a loss is incurred by him which he is not in a position to bear.

In recent years, through studying the development of this question of personnel, there has come the recognition of the desirability for the individual to be guaranteed against loss. Probably it is an outgrowth of our appreciation of the benefits of insurance. In this case, a company has put itself on record as recognizing the interests of the men to be laid off, saying that it should be the men who will suffer the least loss. This is surely a most notable advance in the right direction.

Classification is probably open to further study to determine the question of whether these classes are of equal gradations of interest,

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and whether the men included in one group may not belong with another group.

C. H. BENEDICT,* Calumet, Mich.—Our experience, without being formulated to that extent, was about along the same lines. We made three classes—the man with dependents, the single man, and the returned soldier. We took the stand that the community had a decided responsibility to the returned soldier so that we gave every soldier his position or the equivalent thereof. Very often it was hard to decide between the young man coming back from the war and some one who might not be so fortunately placed, yet we always took the stand that the soldier was entitled to the position if he wanted it. Occasionally we put the question up to the soldier, who very often would say that he would go somewhere else to look for a job. But when we laid off the men, we laid off the single man first. When it became absolutely necessary to lay off the men with dependents, an effort was always made to cause the least loss not only to the man but to the community as well. While companies may not have advertised the fact, I think there is a great deal of time and a great deal of consideration being given to this question by the management that is not appreciated by the community or possibly by the workers themselves.

CHAIRMAN COPELAND.—Did you find, what seems to have been a very common experience, that the returned soldiers in many cases did not want their old jobs back, that they wanted something different?

C. H. BENEDICT.—In general, they were very glad to get their jobs back again, although a very large percentage of them later gave up their old jobs because they were dissatisfied.

G. M. TAYLOR,† Colorado Springs, Colo.—We had about 180 men go to war from the mines; about 15 per cent. of them returned and about a third of these quit in a month.

CHAIRMAN COPELAND.—There seems to be a great difference of experience with the returned soldier. I am very much interested in that question. In almost every case in my experience the men were glad to get their old jobs back. But I know that here in Chicago there has been much trouble; a man who had been a plumber wanted to be general manager, and so on, but our experience was contrary to that, the men were glad to return to their old jobs.

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# Physical Examination Previous to Employment

BY CHARLES F. WILLIS,* M. E., BISBEE, ARIZ.

(Chicago Meeting, September, 1919)

THE time is no longer when a man can act as an independent unit; the appreciation of the interdependence of one man upon another has emphasized the importance of the social unit. Epidemics have made us recognize that even a man's health is not distinctly his own to control as he pleases.

The physical examination of workmen previous to employment started with an effort to prevent the spread of tuberculosis, for tuberculosis may be detected in its early stages and cured. But it was soon found that many other diseases were detected and the saving of lives made possible. The physical examination does not mean the elimination of the unfit—on that basis it would utterly fail—but rather the measuring of a man's physical fitness and placing the man where he can do the best for himself, his fellow worker, and the company. Industry as a whole cannot expect to live up to the standard of physical examination set for the army, nor should it expect to do so, for in all branches of industry there are types of work that do not require the same amount of endurance as does army work, and the placing of men with physical limitations in the work for which they are capable permits a higher average of physical fitness for the work requiring physical excellence.

## OBJECTS OF PHYSICAL EXAMINATION

The objects of physical examination are: The early detection of illness, particularly at a time when a full restoration of health is possible; the protection of employees from infection caused by working in contact with contagious diseases; the discovery of a man's physical limitations, in so far as his possibility for rendering good service is concerned; assistance to the safety movement by eliminating association in hazardous occupations with men whose physical condition renders them likely to accident to themselves and others; and the lessening of time lost by sickness. Statistics issued by the United States Department of Labor show that 22.54 per cent. of idleness is caused by sickness, with an average number of 7.71 weeks idle; whereas only 1.66 per cent. is caused by acci-

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dent with an average number of 8.98 weeks lost. Sickness is responsible for almost one quarter of the economic losses due to unemployment.

The physical examination, however, will not stand as a preventive of illness and accident unless followed by the most minute detail. The army does not merely get fit men into its service but does everything possible to keep them fit, and in industry unless like care is taken and every effort made to follow the advantages of the physical examination, nothing is gained. The employment of healthy men to do work in unhealthy surroundings is nothing short of criminal.

### Advantages of Illness-prevention Work

Many mine managers have recognized for some time the importance of illness-prevention work, but they have appreciated that illness cannot be prevented by the physical examination alone any more than by improvements in sanitation, sick benefits, free medical advice alone, or any one of the other factors entering into it, and that only by the combination of all of these factors can results be secured and the benefits obtained. There are so many factors and so many branches that illness-prevention work, off hand, looks to be costly, and statistics have not been available as to its necessity or even its cost to the industry. But the experience of those who have been active in illness-prevention work is that it has paid big dividends in money, satisfaction, continuity of work, and contentment. Another reason for the lack of adoption of the physical examination has been the financial unpreparedness of the average employee to meet the exigencies of sickness, which delays early detection and treatment. While without a doubt this will some day be cared for by the state with some form of insurance, there are many ways of caring for it at the present time, and the mines where physical examination is in vogue do not allow that feature to stand in the way of its success.

### Method Used at Copper Queen Mines

The Copper Queen Branch of the Phelps Dodge Corpn., operating in Bisbee, Ariz., has had the physical examination as part of its general plan for prevention of illness and accident and the improvement of the physical caliber of its workmen for a number of years. A study of the methods used, their relation to the other activities of the company for the benefit of the workmen, and the results obtained are well worth the time spent, for the methods have been very beneficial to the operators and more so to the employees.

In the early days of the Copper Queen mines, sickness was common, epidemics not infrequent, and accidents an everyday occurrence, with the result that the employees were a heterogeneous gathering of inefficient individuals who appreciated the fact that the company cared little

for their well being and who cared little themselves. Soon, however, there was a well-devised plan that not only included better health for the employee, better working conditions, and better living conditions, but also better social and recreational facilities. It included practically everything necessary to make a well-organized, substantial, permanent community. Space will not permit the giving of details of the plan, but only a reference to some of the activities relating to illness prevention, particularly the physical examination. Parts of the plan are common to many companies, but the methods of relating them to the general plan are not so common.

A contract-hospital system is a common factor in nearly all mining communities today, but the ordinary hospital of the mining camp is not aimed so specifically at illness prevention as is the one at the Copper Queen. Its policies are broad; the desire is to be as liberal as possible, not to draw the line closely as to what should be included in the hospital fee. The hospital does not, by many thousands of dollars, support itself by its fees, but the loss is more than made up by the greater satisfaction given the employees and in the knowledge of the officials of the company that the hospital is performing the best service possible and that the doctors are not using their positions to obtain money from the men on various pretexts. As is usual in contract-hospital systems, elective operations are charged for, and occasionally disputes arise as to what constitutes an elective operation. As the object is not merely to cure men but to keep them well, in the Copper Queen hospital any operation that will improve health is ordinarily considered necessary and is performed without extra charge.

The liberality of the hospital policy would not be possible if it were not for the physical examination. It would not be feasible for a company to employ men irrespective of their physical condition and allow all the same hospital privileges. The physical examination becomes to some extent the selection of risks, and because of this selection the very liberal policy is made possible. Without a physical examination, a liberal hospital policy would soon cause a camp to become a gathering place for industrial cripples seeking to be cured. While there is no doubt that there should be such a place, it should be provided by the state rather than any one organization.

The Copper Queen also maintains a beneficial association for its employees, which is nothing more nor less than a cheap form of insurance against sickness and death outside of work. During 1917, the employees paid into the association $55,362.82 and the company contributed $12,600.85. The total benefit payments amounted to $69,844. This insurance only costs the workman 1¾ per cent. of his daily wage up to a maximum of $2.19 per month; and in case of death by accident or sickness while off duty, one year's wages are paid the dependents, not exceeding

$1500. In case of the loss of time by accident, one-half wages are paid during disability, not exceeding $62.50 per month. In case of the loss of a hand or a foot, both hands or both feet, or both eyes, one year's wages are paid, not exceeding $1500; and in the case of the loss of one eye, one-half year's wages are paid, not exceeding $750.

The support of the employees' beneficial association comes from the recognition of the Copper Queen officials that the health of the workmen and their continuous employment is worth money to the company. Without a doubt it is worth more to the men themselves, but the company is willing to pay a proportion of the necessary expense, the percentage depending on the number of the employees who are members of the association. It is a cheap form of insurance, but if men were taken into this insurance company without an examination as to their fitness, the cost of such insurance would become prohibitive. The physical examination, therefore, makes possible a very low insurance, particularly when the insurance company is run without overhead expense, without the necessity of a large surplus fund, and with the company paying a considerable percentage of the premiums.

The physical examination was also made a part of the general plan for the improvement of conditions for workmen due to its relation to the safety movement. In no other industry is the safety of a man more interwoven with that of his fellow workmen than in mining. For instance, when a hoisting engineer died suddenly from heart failure, three men in the bucket dropped to the bottom of a prospect shaft and were killed. Without a physical examination, men with hernia are likely to be placed where heavy lifting is necessary and the hernia aggravated. Many instances may be related of accidents to others due to a spell of weakness, dizziness, fits, etc. of one man. Bad eyesight also is the cause of many accidents.

In line with the general improvement in health and physical fitness of their employees, the Copper Queen has installed every sanitary convenience that seemed practical. The installation of such devices, however, would mean little if the company were to employ physically unfit men to use them. It has been the desire of the company to build up its mines and its community, and to have its workmen physically the best.

## Samuel Gompers' Opinion of Physical Examinations

Samuel Gompers, president of the American Federation of Labor, is responsible for the following arguments against physical examinations: (1) The apprehension in the minds of the workers that, if their deficiencies are ascertained, they will be discharged and will have to walk the streets in idleness and thus aggravate their situation and condition. (2) The rejection of the unfit, which practically means condemnation and suffer-

ing for those depending on him for support. (3) The fact that many applicants for work have been weakened and enfeebled by long periods of unemployment, lack of proper nourishment, etc. (4) That there is a tendency on the part of the companies working in illness prevention to extend their sphere of jurisdiction into the homes. (5) That there is no provision for the care of industrial cripples.

There is no doubt that some of these objections would be tenable if the physical examination were carried out with those objects in view. But the employer who carries out the examination for the purpose of building up a physically perfect working force rather than for the benefit of the employee himself, will ultimately defeat the real purpose and object desired.

Two years later, however, as chairman of the Committee on Labor, Mr. Gompers recommended to Secretary of Labor Wilson that medical examination of applicants be made one of the functions of the government recruiting agency. This recommendation was the outcome of a conference held in New York, July 15, 1918, and embodies the consensus of opinion of physicians and public-health workers. Resolutions stated that it was the sense of the conference that the physical examination of workers is primarily a measure of health conservation, and also is essential to maximum production, a war necessity, in that the purpose of the medical examination is not to eliminate the worker from industrial service, but to adapt him to the work for which he is physically fitted.

## Method of Making Physical Examinations at the Copper Queen

The method of carrying out the physical examination by the Copper Queen answers well the arguments against such examinations. The large number of cases that come to the attention of the medical examiner are not cases of illness that require hospital or sanitorium treatment or necessitate the laying off of the man. Full preservation of the rights of the employees is maintained and the matter of following up the results of the examination by re-examination and treatment is strictly optional with the men. The examination does not increase the authority of the employer over the workman, and, except in a few cases, it has been the desire of the employee to correct and maintain good health.

To secure privacy, the examiner's room is fitted with five private rooms, all of which open directly upon the doctor's room but are not open to each other.

The attitude of the examiner is important. Dr. R. B. Durfee, who conducts the examinations of the Copper Queen, emphasizes the fact that the examination is made in order that the workman may have a better knowledge of his own condition. He is told of his condition as the examination progresses, wherein he can help himself, where treatment is

necessary, and where changes in his living conditions would improve his health. Many of the men are given information that is invaluable to them.

During the six months' period, ending June 30, 1918, the medical examiner of the Copper Queen mines examined 2342 men, 88 of whom were rejected. Of these 88, 20 were conditionally passed by the mine superintendent and put to work. Therefore, the rejections amounted to about 30 in 1000, as against several times that number in the army. It can be readily seen that the Copper Queen policy is not the rejection of the physically unfit, but rather the placing of men where they are best fitted and where the safety and health of their fellow employees are not impaired.

Three men were rejected for albumen; this is only in extreme cases, however. A man is usually put to work where conditions warrant treatment and cure. A careful examination is made of the heart, for this has its influence upon safety. A man suffering from a diseased heart is a menace to his fellow employees and is physically unable to carry on his work. He is likely to become unnerved if engaged in work that is full of quick surprises or excitement, the excitement itself saps his strength, interferes with his own safety, and adds to the risk of others. However, only seven men were rejected for this cause. Mining is arduous work and involves much heavy lifting; therefore a careful examination is made for hernia, a serious natural weakness common in every walk of life. Thousands of people have hernia without knowing it until it becomes painfully serious or is pointed out to them. A comparatively simple operation cures the disease in a short time. Out of over 2300 examined, 48 were found to have severe cases of hernia. Practically all of the 20 men conditionally passed by the mine superintendent were men having hernia, which was to be corrected by an operation later. A ruptured person mortgages his vitality and gambles with his life when he lifts heavy loads or even coughs violently, and he should not be placed in a position where he is liable to injure himself.

It is only in the case of active tuberculosis that rejection is made; in ordinary cases of lung weaknesses men are placed in out-of-door work where the danger of contagion is small and where the likelihood of cure is large. But seven men were rejected for tuberculosis. Men with this disease in an active stage are unable to do a day's work and, what is more important, they are likely to extend the disease to other workmen when working in a confined place. The detection of weak lungs offers the doctor the opportunity to give men advice as to their methods of living, eating, etc., which will correct their condition.

Six men were turned down because of their eyes. Those with ordinarily poor eyesight are not rejected—practically every case amounts almost to blindness. It is not uncommon for a man to be entirely blind in

one eye without knowing it. If eye tests had been common in industries in the workman's early life, the sight of the now useless eyes could have been saved in many cases. Occasionally cases of trachoma are found, which call for special care in order to protect other workmen from the disease.

The rejection on account of defective hearing is a matter of degree. Two men were turned down for this cause during the six months, both of whom were cases in which there would have been considerable danger to themselves.

It is not uncommon to have men making application for work who have one arm, one leg, or who have not the use of their arms or legs; and while there are occasional positions in which these men might be placed, and it is the policy to place them whenever possible, usually rejection has to be made. One man was rejected on this account.

Men having venereal diseases in an acute and contagious state are rejected, although there was but one case of this in more than 2300 men examined. The conditions of a mine where men are working in more or less confined quarters permits the spread of such diseases far more readily than in many other industries, and no man desires to be in direct contact with venereal diseases.

Drugs, alcoholism, and morphine were responsible for the rejection of four men. Such men are dangerous to others.

Bright's disease accounted for one man, fever for three, measles for one, and teeth for one. Bad teeth lead to indigestion, which in turn clogs the mental and physical powers and makes a person stupid and inefficient. They also send germs in increased quantities to the lungs and heart, tighten joints, and cause rheumatism. Rejection is only in extreme cases, however, for this reason.

It can be seen from the foregoing that the Copper Queen method is by no means the rejection of the unfit, but rather the rejection of those almost totally crippled and those who are a menace to their fellow employees. There can be no question of the advisability of the physical examination, whether it is considered from the humanitarian standpoint, or the selfish cash-conserving standpoint. It adds to the workers' physical comfort, gives them better employment for longer periods, removes, as far as possible, danger from contagion, and improves conditions in their homes and in the community. By paying attention to physical defects in time, defects that would be overlooked if such physical examinations were not made, hundreds of employees have been refitted for work who would otherwise be jobless because partly incapacitated. Moreover, the health education imbibed by the workers extends to their families.

The fear that has been expressed that the practice of applying the rule of physical fitness will rule some men out of industry is shown, by the records of the Copper Queen, to be of little weight, because very few

applicants are barred from industry because of their physical defects; and we must agree that it is good policy to quarantine out of industry a man whose physical condition carries with it a menace to his own safety and to that of those who work with him. The records show that by locating and protecting their infirmities, hundreds of defectives have been directed into safe and remunerative places in industry who otherwise would have been unable to continue at work.

## DISCUSSION

THE CHAIRMAN (F. K. COPELAND,* Chicago, Ill.).—This is an interesting and very troublesome proposition to all of us. Ten or fifteen years ago, when the old-fashioned idea prevailed that a man was responsible for his own health and safety, that if anything happened to him it was his own lookout, we got along without this problem; but with the advent of accident compensation, with the agitation that there is all the time for pensions, sick benefits, and the responsibility of a company for the health of its employees, whenever you take a man on, that phase is becoming more and more important. One cannot afford to hire a man who is blind in one eye, even though he may have perfect sight in the other eye; if he loses that one eye, if he becomes a total disability, the company is liable for the man's complete sight.

Another thing that should make us particularly interested in this problem is the fact that the unions are very much opposed to it. One of the requirements of this steel unpleasantness is that the companies abandon their physical examination of employees. It is a very difficult problem to examine 400 or 500 men as intelligently as possible and decide whether a man on this or that side of the line is accepted.

T. T. READ, Washington, D. C.—The United States Bureau of Mines has in progress at the present time a study of the effect of underground atmospheric conditions on the safety and health of the workers. Improved methods have been employed and it is an excellent and extremely valuable piece of research work.

It is a distinct disadvantage that, with the present compensation laws, the crippled man is penalized because of the fact that he has lost one leg. If he loses the other he then becomes totally incapacitated and his employers will be correpondingly responsible, therefore no company cares to employ him. He must receive his total disability compensation if he loses the other leg, which he is more likely to do than the man who has two legs with which to get out of the way. This problem has been discussed with the organizations that have charge of the vocational

* President, Sullivan Machinery Co.

re-education of returned soldiers and they are taking steps to adjust the matter.

I would like to ask if any one knows what is the real cause of opposition by labor organizations to physical examination, aside from the general policy of labor to fight anything of a compulsory nature? The arguments put forth by Mr. Gompers have very little basis of fact to rest upon.

Chairman Copeland.—I have never been able to get any real light on the subject except the general theory that all men are equal and entitled to the same wage, and that if a man is a cripple he is discriminated against. I think the same difficulty has been found in the application of the minimum wage. A minimum wage is being established by some of the states, Massachusetts for instance, particularly in the employment of women. As the employer has got to protect himself against the employment of inefficients, instead of employing a man or woman and paying what each can earn, the question at once becomes, can the applicant earn the minimum wage which tends to throw a lot of people out of employment?

# Educational Methods at the Copper Queen

BY CHARLES F. WILLIS,* M. E., BISBEE, ARIZ.

(Chicago Meeting, September, 1919)

MANY of the failures in vocational education are due to the fact that the educational methods were not designed to the capabilities, habits, and environments of those to be trained; rather they were based on the policy that education is education and lessons are lessons, whether for youngsters of fifteen or for men of forty.

Vocational education among miners involves, first, a study of the men themselves, their habits, and the use of their leisure time; second, a study of the differences between the methods of teaching required for men and for children. Certain fundamental requirements of training methods for miners are reasonably universal and may be applied with success to any ordinary mining community. The rate of labor turnover is another consideration; it would be useless to design a four-year course for men who remain in one place only a few months.

## Length of Course

The first consideration should be the length of the course. Few mining communities have a sufficiently stable population to make it possible successfully to carry on a course for over a year. While some men might complete a four-year course, the number is comparatively small compared with the number to whom a shorter course would be of service, and it is better to raise the standards of education even a little for a large number than to raise them to a high level for a few.

It is perfectly possible to attain both objectives by arranging the courses in small units; all the units, when grouped together, should comprise a fairly complete course, yet any one unit should be complete in itself and of benefit to the workman. It should not take many of these units to make a miner or a timberman out of a mucker; but to do this the course should be short, because muckers seldom stay in one place long. By grouping a number of these units together it would be possible for a miner to become a shift boss or a shift boss a foreman, and so on. As far as possible the units of the course should not be dependent on those preceding or following. The workman understands the completed job but he is not accustomed to carrying on several jobs at the same time and

* Consulting Supervisor, Department of Industrial Relations, Phelps-Dodge Corpn.

progressing in each. The mind of the schoolboy can carry geography, English, history, and other subjects continuously, but the mind of the miner, unless he is especially trained, can carry better one thing at a time. This develops what might be termed a single-track mind, and the public schools are trying to develop minds with many tracks, all of which may be in use at the same time. The school knows, however, that the process is continuous, while in industry the educational processes may be concluded whenever a man desires. The mind of the ordinary workman can better be devoted to one thing at a time; therefore, rather than trying to make his mind suit traditional educational processes it is better to make the educational processes suit existing mental conditions.

The division of the course into individual units has further advantages in that it permits a man to enter the course at any time, complete it, and go ahead with the next course. He may start at number 14 of one course and finish with number 13 of the following one. There is no reason why he should be denied the benefits of the work because he did not happen to be in the camp when the course started. This permits the course to adapt itself better to the type of men found in some camps. Such a course is self-advertising; the interest of its own participants is one of the means of bringing new attendance, and the fact that a man may start any time, or, if circumstances require, drop out and be able to pick up the work again, is worth much in the efficiency of the work. Any man is better off for having attended even a part of the work.

## Frequency of Lectures

The question as to how often lectures should be given might vary with the individual community. The number depends on the hours of labor, the fatigue of a man's daily work, the outside attractions, and other things. They should be given as often as a full and regular attendance can be secured. In some places where there are few social functions, two or even three nights a week might profitably be given over to the work, while in others there would be but one night in which a full class would attend. The course should be designed to offer the greatest attraction to the men, not merely put up to them to "take it or leave it." The fact must not be overlooked that vocational schools are dealing with mature men of more or less fixed habits, and it cannot be expected that a man can suddenly drop all of his former recreational habits and change to evening after evening of studious endeavor. He may try to, but will not keep it up, unless he is a man of extraordinary ambition and unusual force of character.

Not less than one night a week and not more than two seems best, and even two-night periods cannot be successful except in a community that is quite lacking in other entertainment. Three nights in two weeks,

once in two weeks, or any other irregular time should be avoided, because of the likelihood of forgetting the lecture hour. It requires but a short time to have the habit of attending the course on a certain night as fixed as any other habit. Care should be taken to avoid the postponing of any lecture, for this affects attendance as a habit.

The workman attending night school cannot listen and take notes as is done in college. It must also be remembered that he already knows much about the subject being discussed, and that he is being taught to correlate and use his knowledge more effectively. To avoid the taking of notes and to encourage discussion, it is advisable to supply each lecture printed, or mimeographed, the week previous to the date of delivery, so that the men will have their questions formulated in their minds. This permits the lecture period to be devoted largely to discussion, which in itself is of value, as it makes the man do some thinking on the subject. This method also requires that all lectures be carefully prepared, not the "hit or miss" exposition usual in unprepared lectures, unless given by professional and practical public speakers.

## Object of Course

A definite objective is an important consideration in a vocational educational course for miners. Is a man taking the course so that he may produce more for the company, or so that he may share in the increased production? What benefit accrues to the workman if he gives his evenings to study? His study is not of the character that makes his life more pleasant; he is not getting the pleasure that art, music, or literature might give him. What does he get? As a rule, he gets no more pay, for most mines have fixed scales for all men of one classification and not individual rates. Unless a man can see where he is to be a personal and individual gainer by devoting his time to study, he is not likely to enthuse to the extent of giving up his evening of "Kelly," his weekly picture show, or anything else for the work. An objective is necessary for a successful educational course.

## Type of Teacher

Probably the most important consideration is the type of teacher employed. He must be a man that the men feel confident can do better than they the things that he is teaching. They do not want a college professor, because they doubt his practical ability. But the man who knows the subject and can gain the confidence of the men ordinarily cannot teach, and the teacher is not usually a practical man. It is the case of the educator versus the practical man. A man may be the best timberman alive and yet be totally incapable of telling others how to do it;

and, vice versa, the most expert man in the science of holding up ground may be unable to place a stick of timber.

This difficulty may be met, however, by selecting as the head of the educational department a trained teacher who has a broad, general knowledge of the subject to be taught. The teachers may then be selected from among the employees and bosses irrespective of their teaching ability or their knowledge of the use of the English language. The use of the printed or mimeographed bulletins permits lengthy discussions between the director of the course and the lecturer; it permits the director to plan outlines, make suggestions, and revise, and virtually amounts to having the director training the teacher to teach his subject. It is surprising to note the value of this method. The lecturer is far better acquainted with his subject after he has put it in writing and is proud of the finished product and is much more of a student of the subject than previously.

## Methods Used at the Copper Queen

Coal mining has been far in advance of metal mining in vocational training largely because, in most parts of the country, it is necessary for a boss in a coal mine to have a certificate. Metal mining has not recognized this need as yet, although it is but a question of time when all men seeking positions of high responsibility in the mining industry will be required to show by examination that they have the necessary knowledge and ability at least to maintain safe and sanitary conditions. It was the safety end that led to the examination of bosses in coal mining, and it will probably be that end that will procure the same result in metal mining. Constructive thinkers are foreseeing the time when capable shift bosses will be scarce, when men with a broad view of the mining industry can be obtained only through the technical schools, unless some other provision is made, and when men who can be correct interpreters of the company policies are few and far between. For this reason the Copper Queen authorized and endorsed what is known as the Warren District Practical Mining Course, which was designed for the education of miners to shift-boss positions. The men were also told that those who satisfactorily passed the examination would be given preference in the selection of bosses. While the educational plans are primarily those of the Copper Queen, they open the way to promotion in the other branches of the Phelps-Dodge Corporation—the Copper Basin at Prescott; the Bunker Hill Mines Co., Tombstone; Moctezuma Copper Co., Nacozari, Sonora; the Burro Mountain Branch, Tyrone, New Mex.; the Stag Canyon Branch, Dawson, New Mex.; and the Morenci Branch, Morenci, Ariz.

The plan provides for one lecture a week, forty-three lectures covering a period of ten months; these lectures are given in both the afternoon

and the evening to accommodate both shifts. The subjects are so arranged that men may start at any time during the ten months. The lectures are printed and distributed among those taking the work previous to being delivered. They are perforated and contain a bibliography of the subject and possible examination questions, thus permitting a preliminary study of the subject and allowing the lecture period to be largely discussion and amplification.

For the purpose of examining and rating applicants for shift-boss positions, an examining board, consisting of the superintendent of the mining department, two underground superintendents, and two miners selected from the men who have completed the course meet and rate applicants upon the following scale: Experience, 20 points, attendance,

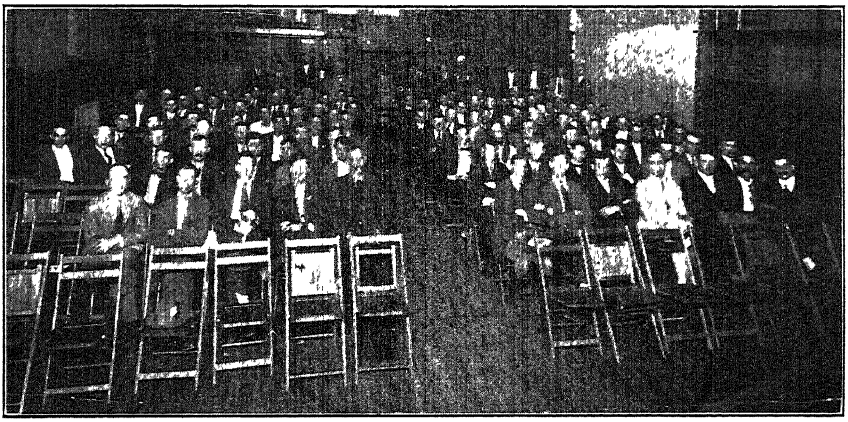

FIG. 1.—EVENING SECTION OF WARREN DISTRICT PRACTICAL MINING COURSE.

10 points, habits, personality, and ability to handle men, 20 points; examination, 50 points; passing grade, 80 per cent.

It is not necessary that applicants take the course to be put on the eligible list for shift-boss positions; they may merely take the examination and be rated. Men passing the examination will be given a certificate signed by the examining board, which is of value in any mining community, as the high standards of the Copper Queen are well known.

The course given is of state-wide importance, for it is adding greatly to the number of trained men available in the Warren District as well as in other parts of the state. The Copper Queen does not expect to hold all the men who, by better education, can improve their earnings and position—it will absorb as many as possible in its own work, but it is reasonable to suppose that men qualified by the training will have attractive offers elsewhere.

The present corps of foremen and executives of the Copper Queen act as a board of advisors for the course, adding their knowledge of Copper Queen methods and policies to the knowledge of the lecturers. With one exception, the latter have been taken from the men within the organization who have specialized in the study of the various subjects. S. C. Dickinson, formerly in charge of the Department of Safety and Welfare of the Arizona State Bureau of Mines, is director of the course, and is giving his full time to the plans for the development of efficient and sympathetic shift bosses.

## Lectures

During the spring of 1918, a number of miscellaneous lectures were given in mining subjects, in which such interest and enthusiasm were shown that the company was encouraged to go into the work on a larger scale, with a definite purpose in view, and the course, as outlined below, is the result of that work.

1. Mine tools—pick, shovel, gad, hammer, etc., their use, design and analysis. Mucking.
2. Breaking ground, general.
3. Drifting—breaking, drift rounds, etc.
4. Drifting—drift timbering, ordinary and heavy timbering, drift repairs, etc.
5. Drifting—track, track maintenance, road maintenance, switches.
6. Shaft sinking, including type rounds, timbering, etc.
7. Raises—type rounds and methods.
8. Raises—timbering, six-post and crib raises.
9. Selection of mining method—various factors governing the same, method and analysis of these factors.
10. Stopes—square-set mining.
11. Stopes—horizontal and inclined top slice.
12. Cut and fill—horizontal and incline.
13. Stopes—caving methods, Inspiration, Ray, Morenci, etc.

### *Miscellaneous Individual Subjects*

14. Sampling—various methods used, elements of inaccuracy, limits of precision necessary, etc.
15. Explosives—manufacture and use of explosives and necessary accessories—loading, firing, tamping, etc.
16. Metallurgical considerations in mining—relation between smelter and mine.
17. Fire-prevention and fire-fighting methods—methods of preventing fires, causes of fires, fighting fire risks.
18. Mine ventilation—study of needs and methods of adequate ventilation, giving both mechanical and natural devices.

19. Safety work and accident prevention—place of shift boss in accident prevention.

*Mechanical*

20. Applied mathematics—necessary mathematics to understand mechanical problems.
21. Principles of mechanics—mechanical types, etc.
22. Machinery—description, use, and care of water drifters.
23. Machinery—description, care, and use of stopers and pluggers
24. Standard machine set-ups.
25. Electrical machinery—general description of electrical machinery and electrical devices in use around mines, motors, transmission methods, etc.
26. Hoisting and haulage—general description of hoisting methods and devices, underground haulage, relative to costs, efficiency, etc.
27. Air compression and transmission—general survey of methods and efficiency of air compression and transmission. Leaks—their importance, methods of determining, and their correction.
28. Pipes and pipe fittings, hoses, etc.—general survey of various types of pipe fittings, methods of using, their relative importance, necessity of care in selection, etc.
29. Drainage.

*Geology*

30. General geology.
31. Geology of Warren District.
32. Elementary chemistry as applied to ores.
33. Ores of copper.
34. Map reading, map making, and interpretation.

*Economics*

35. Elementary economics—brief survey of theory of economics of labor and capital, and industries in general, labor unions, labor policies, etc.
36. Efficiency engineering—history and importance of efficiency work, what it means, how it is applied, its particular problems, etc.
37. Psychology of handling men.
38. Division of labor.
39. Wages, bonuses, and other methods of compensation.
40. Accounting and time keeping.
41. The cost sheet.
42. The communities of the shift boss.
43. Company policies.

## Importance of the Shift Boss

The shift boss is really one of the most important men in the economy of a mine. As one writer recently said: "He is the non-commissioned officer upon whom so largely depends the morale of the crew." He is the link between the company and its employees. A mine cannot be well managed without the aid of one or more good shift bosses. Poor ones will render futile the best efforts of a clever superintendent. The shift boss selects the men for the different tasks he tells them to do and is able to show them how to do it. Being in close touch with the men, he is the first to know if trouble is brewing, if they are dissatisfied or unfairly treated, and he may be the immediate cause of trouble by treating individuals unfairly, showing favoritism, or exhibiting bad temper. Again, the shift boss is expected to have a keen eye for ore, to see that the working places are safe, and to report any significant changes in the face of a level or stope. He sees to the safety of the working places and is the first to give aid when accidents happen.

Ignorant and loutish shift bosses there are, of course, but they are rare, because men of such characteristics rarely hold the confidence of the management or the respect of the men. Not only has the standard of shift bossing improved as mining has become more technical in method, more comprehensive in scale, but a start has been made in training technical graduates for this work of supervision, which affords a splendid training for educated young men possessed of physique and experience that are essential for such a responsibility. The innovation may succeed, but we shall continue to think of the shift bosses whom we have known, not technical graduates, but boss miners, quick witted and strong armed, keenly observant, good natured, and intensely proud of their mine.

## Success of the Copper Queen Methods

The Copper Queen course is about half completed and has been successful, if attendance can be used as a measure of success. The average attendance has been over 200, even the curtailing of forces in January and February had little effect on attendance. New faces are seen at almost every lecture. Originally the evening class was held in a small room in the Y. M. C. A. but has been since moved to the gymnasium. Again, if interest is any measure of success, the course has been successful, for little advertising other than notices, announcing the name of the speaker and the subject, has been done.

It is not yet possible, nor will it be for some time to come, to measure the success of the course in terms of shift bosses who have become correct interpreters of the company policies, but in addition to the men who will qualify and be promoted, there will be hundreds of other men trained, and with a better understanding of their work and of the company.

## DISCUSSION

G. M. Taylor,* Colorado Springs, Colo.—I do not think the plan outlined in this paper would work at Cripple Creek. Most of our men have had a pretty good education. The Cripple Creek district is a lessee district—I do not think there is one mine operating in Cripple Creek today that does not have the leasing system. We have as many men leasing on our property as we have mining and every man who is working for wages is looking for a lease—every man has mined for himself, he has been his own boss on a lease, either on our own property or on some of the other properties, and it makes a different class of employees.

The Chairman (F. K. Copeland,† Chicago, Ill.).—Do you have much labor turnover?

G. M. Taylor.—No, we do not. We are very short of men.

Chairman Copeland.—What has been your experience during the last two or three years, have the men been coming and going?

G. M. Taylor.—No, they have not. When the war broke out quite a number of our men went to the copper mines; 50 per cent. of them returned. They came back to lease and most of them took leases. Our labor turnover is practically nothing.

Chairman Copeland.—You have been very fortunate. I have known cases where it has been as high as 80 per cent. If educational effort like this can give the men an added interest in their work and an added incentive and create the feeling among them that the employer is interested in their progress and is anxious to give them an opportunity to improve their positions, I think that it is a most valuable thing. So many men, in mining and other industries, simply have the day's work before them and it is just a question of getting a day's work done as well as possible for a day's wages. That, I think, tends to crowd them into organized labor, I. W. W., or whatever it may be, so that any employer, in any line, who can make his men feel that he is coöperating with them, that they are a part of the organization, is doing valuable work toward solving some of the problems that confront us.

I was quite interested this past winter in an effort we made toward the education of foreigners. We employ many French, some Poles, and some Russians and started a night school, which after a while we opened to the women—the wives of the foreigners. The attendance was so gratifying that we opened it to all the town. It was entirely voluntary and free. This was not along the line of technical education but of the

* Manager Milling Dept., Portland Gold Mining Co.

† President, Sullivan Machinery Co.

general education of foreigners. It is very satisfying to see their interest, and the regularity with which they attend, and the progress they make.

C. H. Benedict,* Calumet, Mich.—We have good public schools with night sessions which are fairly well attended, but the pupils are almost entirely Americans rather than foreigners.

Chairman Copeland.—I think that is unfortunate, because, unless a man can read and speak the English language, it is very hard to get at him, and I think we all ought to get back of this Americanization movement and push it.

T. T. Read, Washington, D. C.—The fact should be kept in mind that Mr. Willis is discussing vocational education and not training in English, which is now more commonly spoken of as "Americanization." Mr. Willis does not make any mention of the fact, but it should be noted here, that there is a Board of Vocational Education which has a large appropriation for that work. While we have not seen any visible results of its efforts yet, this Board has a good organization and will, undoubtedly, in time do good vocational educational work, so that its coöperation will be valued in years to come. A number of courses in mining have been prepared by competent men for criticism.

Other things being equal, better results are obtained with a capable teacher who is not familiar with practice than with a practical man who does not know how to teach. Some firms have used for this purpose teachers who are volunteers; young men who are willing to give their time, just the way people give their time to teaching in Sunday School. That plan usually does not work well, and if the job is worth doing at all it is worth paying for, even though the pay may be small. You get much better results in the end. Some firms that have given vocational training, to insure that the men would complete the course, have made all persons pay a small sum at the beginning of the course, which is returned to them on a basis of the number of sessions attended, so that if a man quits half way through the course, he loses half his money.

It must always be remembered that you must adapt your methods to the class of men you are dealing with; you have got to adapt and develop your system for your own peculiar circumstances.

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* Metallurgist, Calumet & Hecla Mining Co.

# Outdoor Substations in Connection with Coal-mining Installations

BY H. W. YOUNG,* CHICAGO, ILL.

(Chicago Meeting, September, 1919)

DEVELOPMENT of high-tension outdoor substations during the past few years has been due primarily to economic reasons. The demand for power in small communities could not be met with the conventional and

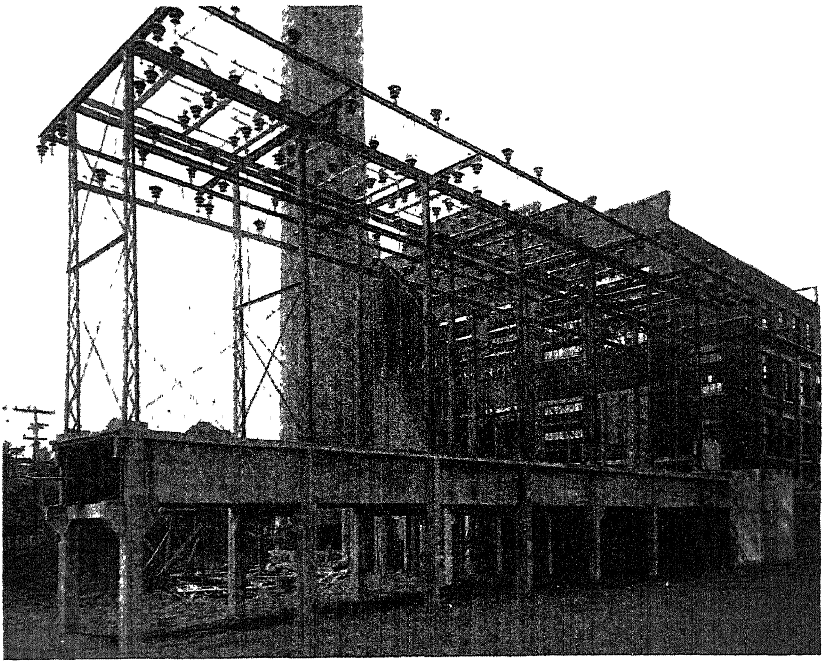

FIG. 1.—33,000-VOLT STEP-UP OUTDOOR AIR-BREAK SWITCH AND SUBSTATION INSTALLATION OUTSIDE OF A POWERHOUSE GENERATING AT 2300 VOLTS STEPPING UP TO 33,000. TWENTY-ONE COAL MINES ARE SUPPLIED FROM LINES LEADING FROM THIS INSTALLATION.

comparatively expensive indoor types unless the rate for service was materially increased. To prove that the problems incident to development have been solved successfully, it is only necessary to point to the wide adoption of outdoor substations by utility companies. Passing at once to the question as to whether outdoor substations are applicable to

* President, Delta-Star Electric Co.

the power requirements of coal mines, it is well to consider first the elements entering into their construction.

### Transformers

Self-cooled outdoor transformers are no longer an experiment. The chief difference between an indoor and an outdoor transformer is in the

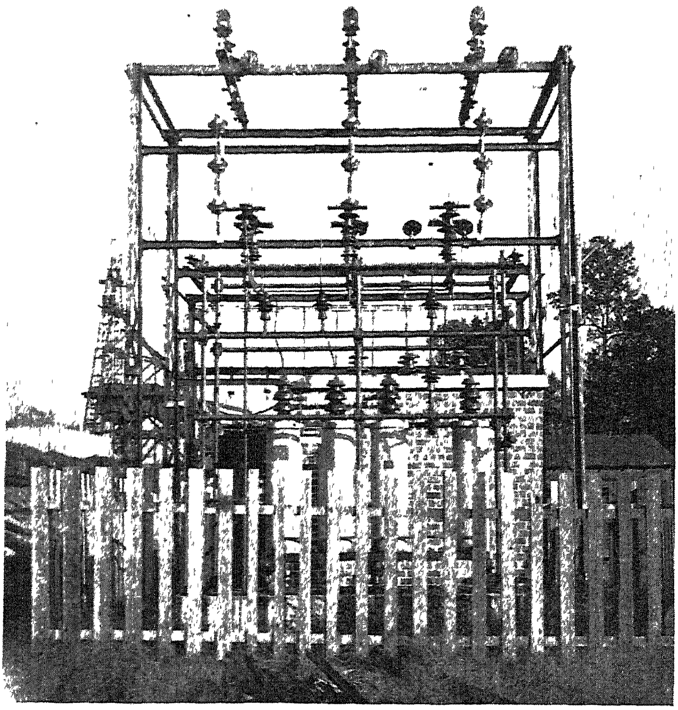

Fig. 2.—900-kw. 33,000/2300-volt three-phase substation tapping power line giving service to a coal mine.

type of bushings and cover. The outdoor type is commercially available in any desired capacity or voltage.

### High-tension Control Equipment

There are two distinct types of high-tension switches, the oil-break and the air-break forms. The oil-break switches are simply a development of the indoor types, the principal differcnce being in housings and terminals. They are thoroughly developed and in successful operation. Air-break switches are an accepted standard for transmission systems and are available in several forms. When provided with arcing horns or discharge horns, they can be used to open loaded circuits.

## Lightning Arresters, Choke Coils, Fuses, and Wiring Supports

Outdoor types of electrolytic, oxide film and horn-gap arresters with or without limiting resistance are fully developed and have good service records. Outdoor choke coils are of the same general design as the indoor type with the exception of the insulating supports. The modern high-tension fuses can be used equally well indoors or outdoors, the only difference being that the outdoor mountings have petticoat instead of pillar-type insulators. High-tension wiring supports are standard commercial devices. Like the indoor types, they are made in many forms and assembled to meet the various wiring conditions.

## Construction Requirements

If indoor equipment is to be used, a building must be erected with wall or roof bushings for the high-tension incoming lines. This building must first be designed for the particular location, hence the services of an architect and a contractor are required. In other words, to properly protect equipment, we have been obliged to incur a very considerable expense that does not materially add to the operation of the electrical equipment. If outdoor equipment is selected, it is necessary to provide a supporting structure for only part of the electrical equipment; the transformers and oil switches simply requiring a foundation slab. The supporting structure is composed of steel sections fabricated in a factory and shipped with the equipment, thus eliminating the expense of the architect and contractor, necessary with the indoor station.

The outdoor substation steel structures are of simple design and can be erected by common labor under the direction of a foreman connected with the electrical department of the mine. The steel-tower outdoor substation does not require much foundation preparation but can be erected on small concrete pillars on the side of a hill, on a piece of waste ground, or at points not suitable for other purposes.

## General Design

The general design of outdoor substations has been thoroughly standardized, the actual assembly of elements depending upon local conditions to be met. Where two sources of power, such as a double-circuit transmission line, are available proper switching arrangements must be provided. A standard design for connecting transformers to either or both of two parallel and synchronous lines is shown in Fig. 3.

At the top of the tower are mounted two sets of three-pole, double-break-per-phase, air-break switches provided with separate interlocked operating mechanisms. Each switch has its own operating shaft and a handle so located that opening and closing the switch can be accomplished

from ground level. In the high-tension side of the transformer wiring are located three single-pole, combination, choke-coil and fuse units. On the opposite side of the tower are mounted three fused disconnecting switches connected to the separately mounted three-phase lightning arresters. From the one-line diagram, the connections can be easily traced. The choke coils are so located that incoming high-voltage surges or lightning disturbances are reflected to the arresters, where they will be discharged in the usual manner.

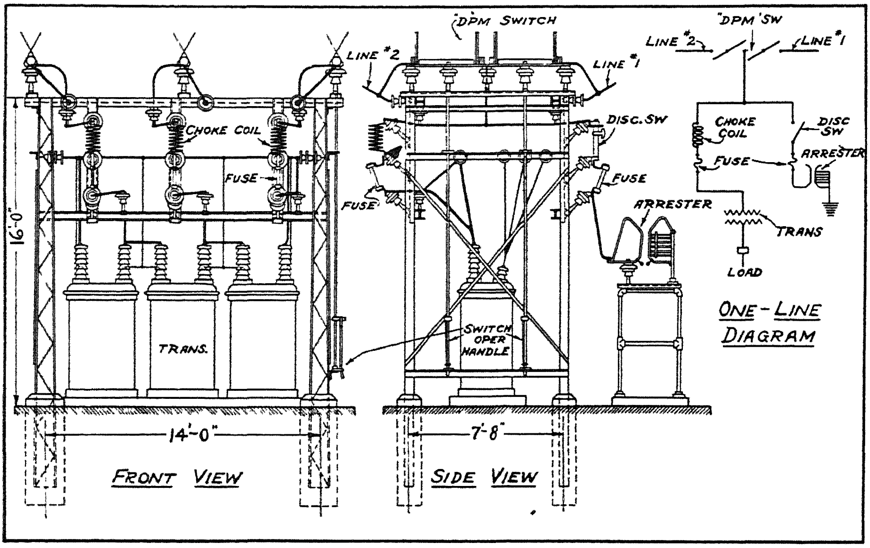


FIG. 3.—STEEL TOWER SUBSTATION WITH TRANSFER TYPE SWITCHES FOR CONNECTING TRANSFORMERS TO EITHER OR BOTH OF TWO PARALLEL AND SYNCHRONOUS LINES

## ELIMINATION OF EXPENSIVE OIL BREAKER

An important feature of this design is in the elimination of an expensive automatic oil circuit breaker on the high-tension side. That this saving can be made without jeopardizing the transformers is not generally understood by purchasers who often needlessly specify their use. It is essential that the expense be kept to the lowest possible value consistent with safety and good operation. As there is quite a difference in the cost of substations using automatic circuit breakers and air-break switches with fuses, the question immediately arises as to whether we can afford to use this lower cost equipment.

## DAMAGE CAUSED BY SHORT CIRCUITS

It is conceded as a fact that the severity of damage done to transformers during short-circuit condition is in direct proportion to the amount

of energy flowing into the circuit before the transformers are disconnected. Whether an automatic oil switch or fuse will clear the short circuit in the least time can be answered as follows: High-tension automatic oil switches of good design will open in approximately 0.16 sec. under short-circuit conditions. High-tension fuses of good design will clear short circuits in approximately 0.013 sec. or from ten to twelve times as rapidly as the automatic oil switches.

## Amount of Current Flow

High-tension fuses will, under short-circuit conditions, permit a current flow many times the full load rating. Assuming that the current flow with automatic oil switches will be no greater and by comparing the time elements of the fuses (0.013 sec.) and the time element of the automatic oil switches (0.16 sec.), it follows that the slower operating switch

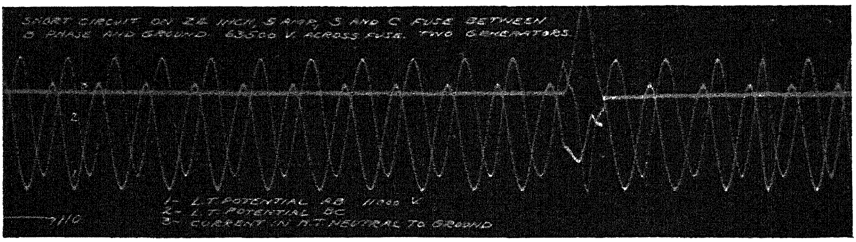


Fig. 4.—Oscillogram record of 66,000-volt short circuit. Two 9000-kv-a. generators in parallel. Line 63,500 to 110,000 volts.

will allow approximately twelve times more energy to flow into a short circuit than will flow with the faster operating high-tension fuse. In considering the above values, it should be borne in mind that the amount of reactance in the circuit, including that of the transformers, will limit the actual current flow with either device.

The oscillogram record, Fig. 4, illustrates this point. This particular test was made by connecting two 9000-kv-a. in parallel, short circuiting one phase to ground on a 63,500/110,000-volt star-connected system. The total reactance approximated 20 per cent. and the current flow with this 5 amp. fuse was 240 amperes.

## Over-fusing the Substation

The exceptionally rapid action of fuses will therefore permit overfusing the transformers theoretically twelve times the normal current and still secure protection equal to that of an automatic oil switch. In actual practice overfusing from three to ten times normal is used depending upon local conditions. For power circuits subject to wide fluctuations, such as mining loads, the high-tension circuit-opening devices should be

so rated or adjusted that they will not operate except in case of actual trouble, such as transformer failure, and such cases are rare with modern transformer designs.

Such overload protection as is desired can be secured easily and cheaply by means of a low-tension automatic oil switch installed on the secondary side of the transformer bank. In substation practice, this combination of heavy high-tension fuses on the primary side, which will only open in case of transformer failure, and a properly adjusted oil circuit breaker on the secondary side of the transformer bank is an excellent protective system often used.

## Uses of High-tension Automatic Oil Switches

The high-tension automatic oil switch has the advantage that it can be used for switching in addition to the overload feature When fuses are used, it is necessary to install an air-break switch to disconnect the transformers from the line In actual practice, however, it is often advisable and is considered good engineering practice to install disconnecting switches between the oil switch and the line. The reason for this is that the oil-switch contacts are necessarily concealed in the tanks and to be absolutely sure that a line is clear the disconnecting switches should be opened, thus giving the operator ocular evidence of disconnection.

Should the oil switch open frequently or under severe load conditions, it is advisable to inspect the contacts, oil levels, and condition of the mechanism. To enable inspection adjustments or repairs, it is often advisable to install auxiliary disconnecting switches so connected that the oil switches can be shunted and entirely disconnected from the circuit, power being temporarily delivered through the auxiliary switches.

Another point in favor of the oil switch is that after interruption it can, if in good condition, be immediately closed, thus reducing the time of service interruption. It will naturally require more time to open an air-break switch and replace fuses than it will to close the oil switch. The saving of a moment's time may in some instances warrant the additional expense of the oil switch. However, the average high-tension substation attendant, especially if not "hardened" by long experience, will after an interruption be inclined to go slow in closing the switch until he can actually view every possible part of the switch mechanism, assuring himself that it is in condition to resume service.

The fuse and air-break switch combination has, therefore, a certain advantage in that every part of the units, including contacts and condition of fuses, are in plain view of the attendant. The average man working on high-tension equipment has considerably more confidence if he can, without a shadow of a doubt, determine its condition at all times. The choice of equipment must finally rest with the purchaser but as a

guide in determining in a general way the dividing line between the use of high-tension fuses and oil breakers, the following tabulation of substation capacities on which fuses can be advantageously used is given, when automatic oil breakers are installed on the low-tension side.

| Primary Voltage | Transformer Capacity, Kv-a. | Full Load Current per Phase, Amperes | Rated Capacity of Fuses |
|---|---|---|---|
| 13,200 | 1,000 | 43 7 | 200 |
| 22,000 | 1,000 | 26 2 | 130 |
| 33,000 | 1,000 | 17 5 | 90 |
| 44,000 | 1,000 | 13 1 | 65 |
| 66,000 | 1,000 | 8 75 | 45 |

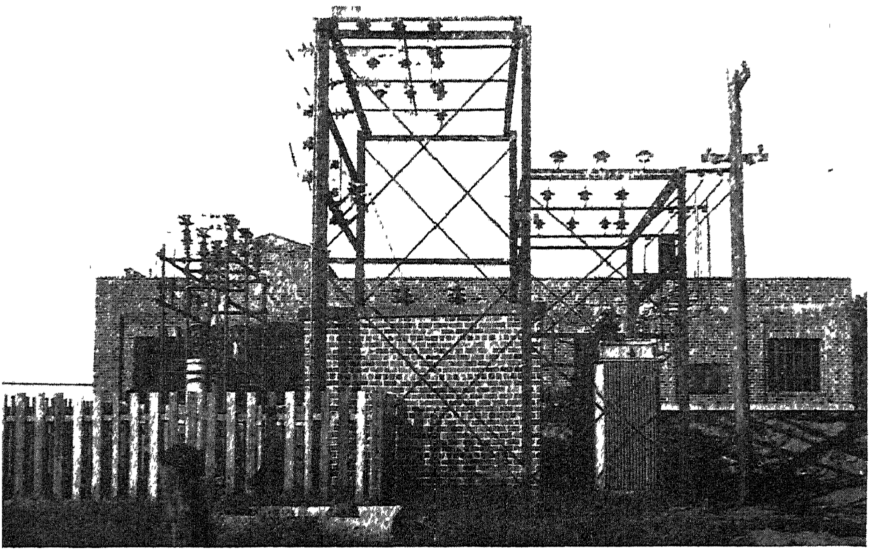

Fig. 5.—1200-kw. 33,000/2300-volt three-phase substation.
High-tension three-pole air-break switch with choke coils, disconnecting switches and limit fuses in lightning-arrester circuit, electrolytic arresters, automatic high-tension oil switches and metering equipment in brick housing. Transformers are located under steel frame carrying high- and low-tension busses.

## Single-circuit Substations

When but a single source of power is used, the substation is simplified by omitting one of the three-pole air-break switches shown in Fig. 3. A typical design having but one source of power is shown in Fig. 6.

A recent coal-mine installation made by Carl Scholz, consulting engineer, for the Valier Coal Co., Chicago, and installed at Valier, Ill., shown in Fig. 7, is of interest in that it is a complete electrical installation using transmitted power furnished by the Central Illinois Public Service company which operates an extensive interconnected system. The

incoming power line is dead ended to the steel tower and, by means of an overhead bus, taps are made to the arrester and load circuits. Three choke coils are so located that they offer a barrier to incoming lightning disturbances reflecting abnormal potentials to the arrester where they can be discharged. In the arrester circuit is a three-pole remote control disconnecting switch operated by means of a handle at ground level. Below the choke coils is installed a three-pole disconnecting switch of the same type used in the lightning-arrester circuit. Below this switch are three single-pole fuse mountings with fuses so rated that they will not blow except in case of transformer failure. The 33,000-volt primary

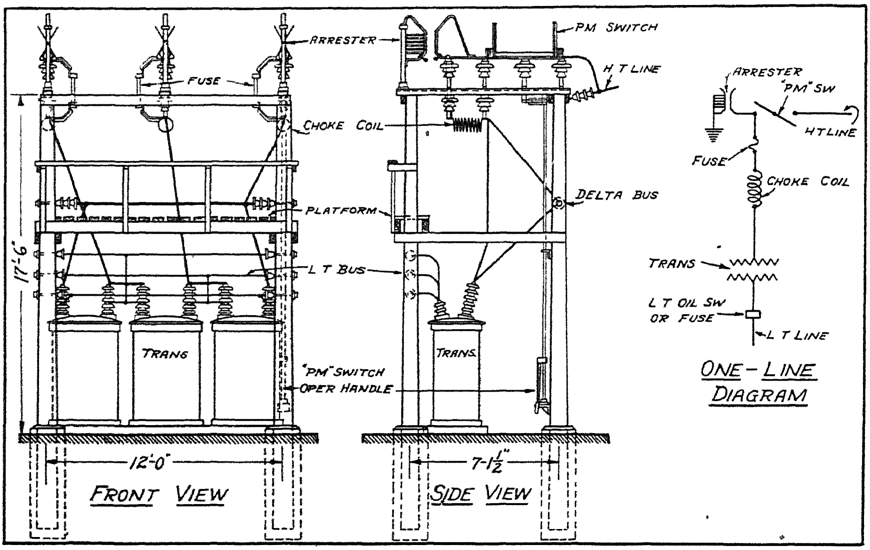


FIG. 6.—TYPICAL SINGLE CIRCUIT OUTDOOR SUBSTATION WITH LARGE CAPACITY TRANSFORMERS AT GROUND LEVEL.

metering equipment is located below the fuses, the current and potential transformers together with the watthour meter and demand indicator being a self-contained unit.

Just back of the main switching tower is located a second steel structure affording space for three 667-kv-a. 33,000/2,300-volt, 60-cycle, single-phase transformers connected in closed delta. Above the transformers are the high-tension busses with their supports. The 2300-volt circuit is carried into the building by means of lead-covered cables and connected to a five-panel switchboard.

Panel 1 is equipped with a 2300-volt check meter and an automatic main oil circuit breaker. From this panel extends a 2300-volt bus located back of the other panels permitting connections to the various loads. Panel 2 controls the main underground feeder; panel 3 controls the 1350-

horsepower main hoist motor; panel 4 controls a 250-horsepower air-shaft motor; and panel 5 controls the machine-shop and yard-lighting line. Each panel is equipped with automatic oil switches and the necessary instruments

The underground cable from panel 2 terminates at a panel on which are mounted two automatic oil switches. The cable from one switch has a tap line to which are connected twenty underground substations, each station being equipped with three 25-kv-a. transformers and an oil switch. The feeder voltage of 2300 is transformed to 220 for use with

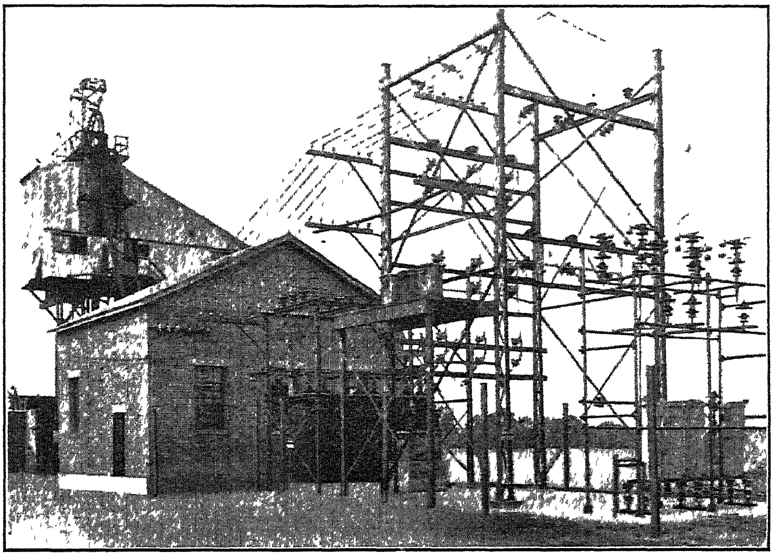

FIG. 7.—SCHOLZ SUBSTATION.

alternating-current coal-cutting machines. On the same feeder cable is connected a 300-kw. 2300/220-volt motor-generator set supplying direct current for the main haulage trolley. This motor-generator set also supplies current to twelve stations for charging the storage battery locomotives. The other switch and cable supplies power to a 100 kw. 2300/220-volt motor-generator set for the main haulage direct-current trolley and also connects to twenty underground substations for supplying 220-volt alternating current to the coal-cutting machines.

The 75-kw. underground substations are at more or less temporary locations so that equipment can be moved as the work progresses, thus avoiding long secondary runs and giving full voltage to the motors. Control of the 1350-horsepower hoist motor will be secured from the underground substation located near the base of the shaft. The operator, by means of switches and indicating devices, has positive control of the

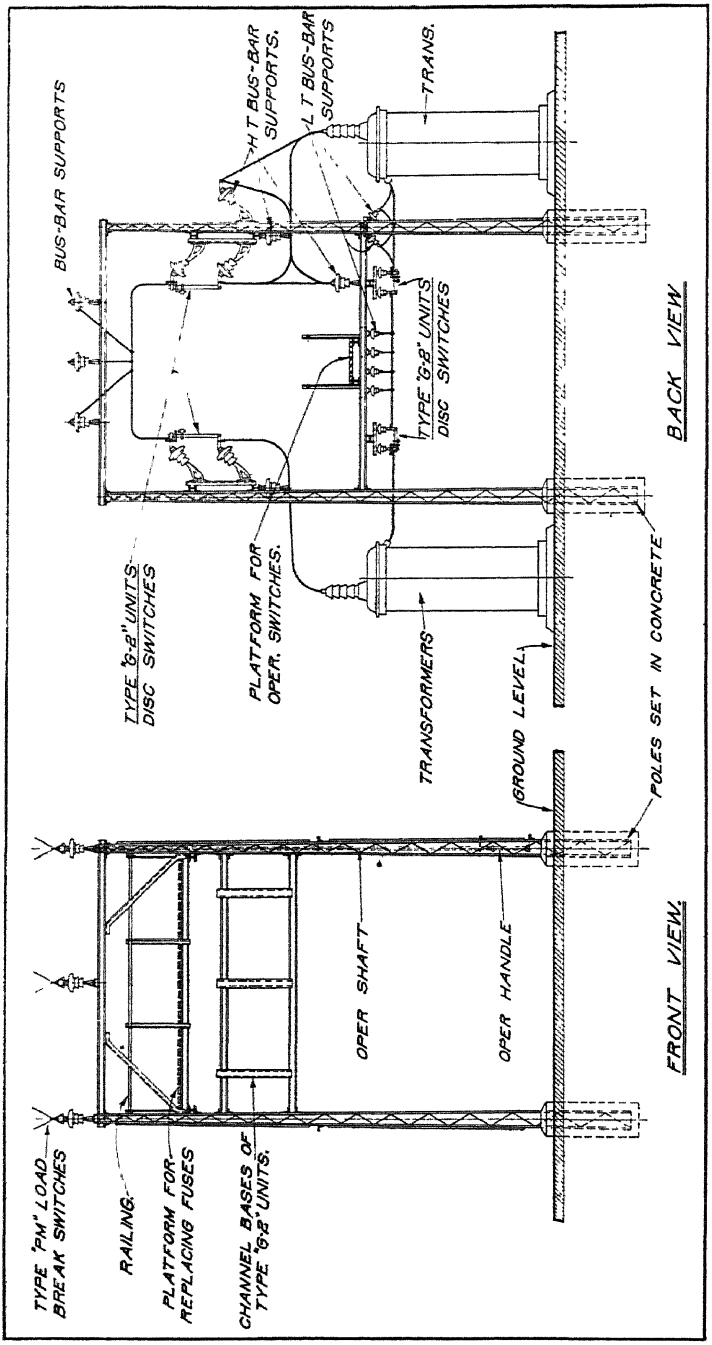


FIG. 8.—TYPICAL COAL-MINE TRANSFORMER SUBSTATION CAPACITY: 4-333 KV-A., 66,000 VOLTS.

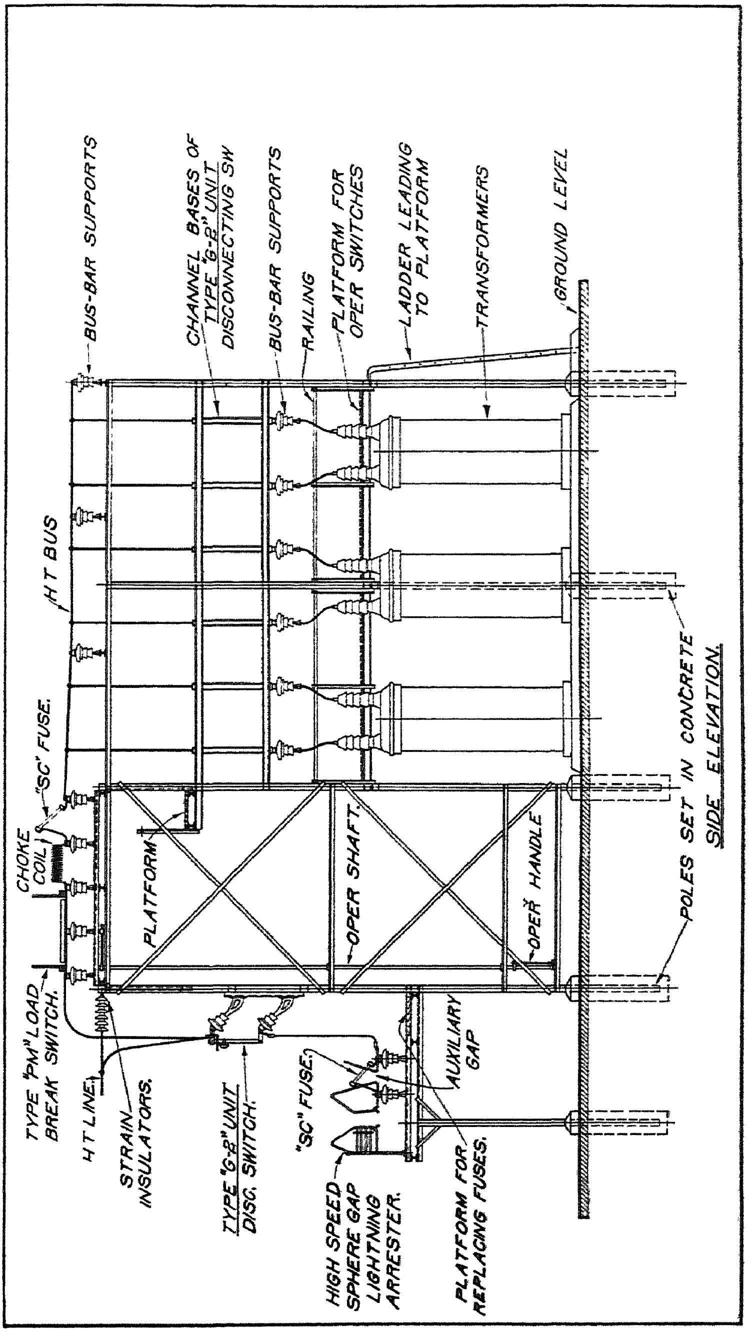


FIG. 9.—TYPICAL COAL-MINE TRANSFORMER SUBSTATION. CAPACITY: 4-333 KV-A., 66,000 VOLTS.

hoist and always knows its exact position. During March 25,000 tons of coal were mined at a power cost of approximately 6.5 c. per ton, the load factor on the station being 41.5 per cent. During this month, power was also used for construction work in progress so that the total cost of power for coal mined was 8.1 c. per ton. When the mine is in full operation and construction completed, the cost per ton will probably approximate 4 c. or less.

### 66,000-volt Substations

The tendency toward higher transmission voltages has developed a need for 66,000-volt outdoor substations and the type shown in Figs. 8 and 9 is a good example of an arrangement for coal-mining service.

A feature of this type is the use of three transformers in closed delta on one side of the station and a spare transformer at the opposite side. In case one transformer fails the spare can by means of disconnecting switches be immediately placed in service and in the correct electrical relation to the other transformers.

The high-tension side is controlled by means of a three-pole air-break switch operated from ground level. On the switch bases are also mounted choke coils and fuses for the load circuit.

The lightning arresters are of the high-speed sphere-gap graded-resistance type with auxiliary gaps shunted with limit fuses. In case of excessive flow of current to ground the limit fuses will operate in the usual manner. The arrester, however, is still connected to the line across the small auxiliary gap. This arrangement eliminates the possibility of entirely disconnecting the arresters after operation of the limit fuses.

## DISCUSSION

Graham Bright,* Pittsburgh, Pa.—It is ticklish business to meter on a high-voltage size. It not only costs considerable, but the metering equipment gets out of order and there are not many operators who like to attempt to correct troubles on the high voltages. This question of metering came up about a year ago in Philadelphia. We were putting in a plant with 66,000 volts on the primary and 6600 volts on the secondary. On account of the difficulty of metering at 66,000, it was decided to meter on the secondary side, so we obtained the transformer characteristics from the manufacturer and knew pretty well what the shape of the load curve would be; then it was comparatively simple to estimate the losses that occurred in the transformer and in the lines to the point where the metering equipment is established. It was easy to add these losses to the meter reading, so the power company did not

* Westinghouse Elec. & Mfg. Co.

have to stand the transformer losses. Though the calculation of these losses may not be quite accurate, the results would be far more accurate than would be obtained by attempting to meter on the high line.

H. W. YOUNG.—Metering on the low-tension side of outdoor substations above 33,000 volts is standard practice, as the transformer losses are well known and can be taken into consideration when making the rate. Below 33,000 volts, we also often meter on the low-tension side but we find that a good many coal mines like the idea of primary metering, as they have not arrived at a definite understanding on what they consider a fair agreement with the power company as to the values of the transformer losses.

H. N. EAVENSON,* Gary, W. Va.—It seems to me that carrying power underground is largely a question of cost. At our West Virginia mines we are putting in two 6600-volt underground substations. We are carrying the current over ground and putting it down through bore holes, because we have placed locations for the substation which can be reached by bore holes, 300 to 400 ft. deep. It is cheaper to do this than it would be to carry the current under ground in a cable. In Kentucky, where we are developing a tract of about 1600 acres, the covering is very thick, and to reach a great many of the substation locations would require bore holes 1500 ft. or more in depth. There we are using three conductor cables, which were insulated 8/32 in. thick, 30 per cent. Para rubber with double braid, weatherproof finish over the three conductors. This cable is for 6600 volts. The cable is being laid in a 3-in. and 4-in. fiber conduit, which is to be laid alongside of the heading and covered with 3 to 6 in. of concrete. Manholes will be placed at intervals of about 1500 ft., depending on the conditions and on the size of the cable used. This cable has not been installed, but work on it is just ready to start. It seems to me that the method of carrying the current depends upon the local conditions of the plant.

H. W. YOUNG.—That is a matter for consulting engineers to decide. The trouble experienced in transmission work has been largely due to the fact that it has not had the proper kind of engineering to start with. In ten years' experience, I have seen many installations where a good engineer could have saved the companies a great deal of loss and trouble. I did not know that you were using 6600 volts underground in the Kentucky fields but it is entirely practical. We are just putting in a 5000-kw. substation at Evanston, using 20,000-volt underground cable.

* U. S. Coal & Coke Co.

# Sulfur in the Coking Process

BY S. W PARR,* M S., URBANA, ILL

(Chicago Meeting, September, 1919)

From a study of sulfur with reference to its specific combination in coal, published as University of Illinois *Bulletin* No. 111, 1919, it is now possible to determine the various forms of this constituent with a good degree of accuracy. This fact suggests two questions: (1) Are the same analytic methods applicable to coke? (2) Is there any correlation between the forms of sulfur as it occurs in coal and the residual sulfur found in the finished coke? The answer to the second question might be helpful in answering a still further question as to what chemical route the sulfur travels during the carbonization process. While the special studies along these several lines are by no means complete, a few preliminary statements are herewith presented. They may, at least, prove suggestive of further possibilities from a continuation of research along these lines.

There is no reason, chemically, why the analytic methods devised for coal should not apply to coke and a fair amount of experience in connection with such use affords confirmation of this statement. The procedure in the case of coal may be briefly outlined as follows:

Combinations of sulfur as sulfate, or sulfides other than $FeS_2$, are readily dissolved without disturbing either the pyrite sulfur or the sulfur that is in organic combination by means of dilute hydrochloric acid—1 part of concentrated HCl to 10 parts of water or approximately 3 per cent. acid. Digestion at 60° C. for 24 hr. insures the complete removal of the sulfur thus combined. In freshly mined coal the amount of such sulfur is very small; weathering processes, however, may very materially increase the amount of sulfate present.

Sulfur in the form of iron pyrites, or marcasite, may be quantitatively removed without oxidation of the organic sulfur as follows: A dilute solution of nitric acid is prepared having 1 part of concentrated $HNO_3$ to 3 parts of water or with an approximate specific gravity of 1.12. The coal sample, ground to 100 mesh, is digested with this solution at room temperature for 3 or 4 days, filtered and evaporated to dryness, taken up with water, and acidified with hydrochloric acid. It is now ready for precipitation with barium chloride in the usual manner. If sulfate is present in the original sample to an appreciable amount, it should be removed by the procedure outlined above.

It is not necessary here to give the methods for determining the two

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* Professor of Applied Chemistry, University of Illinois.

forms of organic sulfur as represented by the content of that element in the resinic and in the humic constituents. It is sufficient simply to note that the organic sulfur may be taken as the difference between the total sulfur and the inorganic sulfur, as found by the two processes just described.

From this review of the methods devised for coal, it is evident that the same processes are equally applicable to coke. However, in applying them the interesting facts are developed that the sulfur in coke is not combined as sulfate nor is it present as sulfide in either the FeS or the $FeS_2$ form. Indeed it is easy to demonstrate, in the latter case, that the iron has been reduced to the metallic state, which removes the last possibility of the presence of sulfur in any inorganic form as we ordinarily understand that term. Since, in the carbonization process, the hydrocarbons have been broken down into volatile material and a so-called fixed carbon residue, it is apparent that the sulfur remaining in the coke must be in some sort of combination with carbon. Studies on the nature, properties, behavior, etc. of this compound are now in progress. From *Bulletin* 111, already referred to, the sulfur thus combined is not affected by either hydrochloric or nitric acids and is stable at 1000° C., but is discharged from its combination by nascent hydrogen.

It is evident, therefore, with the methods thus available for determining the sulfur combinations in the original coal and then in the carbonized material, we have made substantial progress toward determining the disposition of the various forms of sulfur under the application of heat. However, more data must be in hand before a final formulation can be made of the procedure followed by the sulfur in its various changes throughout the carbonization process. Tentatively, the following conclusions seem to be warranted, based mainly upon results that have accompanied our studies upon the low-temperature carbonization process. In these experiments a uniform temperature is maintained throughout the mass at all stages. The products of decomposition, therefore, at these various stages afford a good index of the changes involved. Further, it is to be noted that the highest temperature to which any of the material is subjected does not exceed 750° or 800°. Secondary reactions are thus avoided. We seem therefore to be warranted in making the following statements:

The order of procedure in the matter of liberation of sulfur from its initial combinations is, first, the sulfur organically combined in both the resinic and the humic forms is broken down, part of it remaining in the tar oils as thiophens and part going with the fixed gases as $H_2S$. Half of the sulfur of the $FeS_2$ is discharged practically at the same temperature as that for the breaking down of the organic sulfur. These reactions, for the most part, occur at temperatures below 500° C. As we approach 700° C., the final sulfur of the FeS is discharged, leaving

metallic iron and a carbon-sulfur compound, as already described. This compound is but little affected at higher temperatures, and practically all this residual sulfur from the FeS is retained in the coke, even at the high temperatures of the regular processes, viz., 1000° C.

In the decomposition of the organic matter, the sulfur passing off with the fixed gases is in the form of $H_2S$. Where $CS_2$ is formed, therefore, it must be looked upon as a secondary reaction between the $H_2S$ and the highly heated carbon of the passageways through which the gases are discharged. In the low-temperature process, therefore, where the maximum temperature of any part of the system does not exceed say 750° or 800° C. no carbon sulfide is formed in the residual gases.

Again, as the organic sulfur and the half-sulfur of the $FeS_2$ are discharged at relatively low temperatures and as the carbon-sulfur compound that represents, in the main, the disposal of the remainder of the sulfur is stable at the higher temperatures of the ordinary coking process, it would seem as though a considerable zone of temperature exists wherein the decomposition would yield fixed gases free from sulfur. This has been amply verified, as may be seen by reference to analytical results published elsewhere in the symposium.

This brief reference to some phases of the work already accomplished are suggestive of the possibility of developing further facts in these studies on sulfur, some of which may prove to be of industrial application.

## Summary

The organic sulfur in the raw coal and half of the sulfur of $FeS_2$ is for the most part discharged at relatively low temperatures. The presence of organic sulfur in the coal is not, in itself, responsible for the formation of $CS_2$ in the gases discharged from the high temperature coking process.

Sulfur in any form, either free, in gaseous combination, or solid as FeS or CaS, in contact with red-hot carbon is taken up by the latter as an independent compound, which is stable at the highest temperatures of the ordinary coking process. It is evident, therefore, that the minimum amount of sulfur available for retention by the coke is represented by one-half of the sulfur of the raw coal originally present as $FeS_2$. Under certain conditions, the coke may add small increments by coming into contact at the higher temperatures with sulfur in any other form, either gaseous or solid, but a general line of demarcation is indicated by one-half of the sulfur of the $FeS_2$ present as indicated.

## DISCUSSION

A. R. Powell, Urbana, Ill.—You will probably notice that Professor Parr's paper deals with low-temperature work, heating the coal up to 750° C. and determining the disposition of these different forms of sulfur

in the coal during coking, his conclusions being that in low-temperature coking all of the organic sulfur is driven off and one-half of the pyritic sulfur. When coal is dumped into the ordinary byproduct coke oven, the temperature of that coal will quickly go up to the ordinary coking temperature, which in the 17-in. (43-cm.) retort will be 950° or 1000°, so we have not low-temperature conditions for any very great period of time.

At the time that Professor Parr and I finished our work on the determination of forms of sulfur in coal, we immediately were interested in seeing how that behaved during the coking process. So we put some of the coal that had been analyzed for sulfur forms in a platinum crucible and subjected it to a volatile-matter determination. In one coal in which the pyritic sulfur was 0.31 per cent. and the organic sulfur 0.77 per cent., we obtained as residual sulfur 0.34 per cent. The residual sulfur was therefore greater than the total amount of pyritic sulfur. It shows that, in that case, all of the organic sulfur was not driven off. Some of the organic sulfur is retained in the coke, probably due to a secondary reaction between the sulfur that was volatilized from the organic sulfur in the coal and the red-hot coking mass.

To further prove that some of the organic sulfur of the coal is retained in the coke, we took a phenol extract of coal and evaporated off the phenol. The extracted material, which is organic material entirely and ashless, was coked in the same manner; about one-half of the sulfur of that substance was left in the coke, proving very conclusively that at least some of the sulfur in the coke must come from the organic sulfur of the coal. It might be said that these are simply theoretical results obtained in the laboratory, but one of the Koppers installations in this country gives results that indicate that, if the average of the sulfur determinations are taken for one month or several months, one-half of the coal sulfur is retained in the coke.

I have analyzed coals from this region and one-half of the sulfur in this coal is present as organic sulfur, the other half is pyritic. Therefore, it appeared that if half of the pyritic sulfur were volatilized, one-half of the organic sulfur must volatilize and one-half would remain behind in the coke. It is hard to judge how much sulfur will remain in the coke, because a secondary reaction with other constituents of the coal enters into the question.

H. C. Porter, Philadelphia, Pa.—I would like to call attention briefly to the fact that it is very necessary, in any experiments on carbonization, to take carefully into consideration the temperature conditions throughout the carbonizing mass and surrounding it. I must take exception to the statement of the last speaker, that in high-temperature industrial carbonizing processes, as in a coke oven, there is no low-temperature condi-

tion. There is a great deal of low-temperature carbonizing in a coke oven. It has been established that the coking process consists in the gradual contraction of a plastic layer or zone from the hot walls of the carbonizing chamber toward the interior, this plastic mass being at a temperature close to 400° C.; also that the plastic zone moves rather slowly. In a coke oven, any given section of the coal charge remains in this plastic condition and at this comparatively low temperature probably for 2 hr. or more. At a distance of 3 in. from the hot wall, the coal is at a temperature below 300° C. for 2 hr. after the charge is put in.

The question of the conversion of the sulfur in coal into gaseous forms or fixed forms should either be studied on a scale approximating the commercial, or the experiments should be under well-controlled and well-determined temperature conditions throughout the mass. Carbonization takes place at gradually increasing temperatures, the first decomposition occurring at a very low temperature; therefore, the decomposition of the sulfur compounds should be studied first under low-temperature conditions. The plastic mass does increase in temperature, and there is some decomposition of the sulfur compounds very probably at high temperatures, but the first breaking-down process is at the lower temperatures.

A. R. Powell.—In saying that this did not represent the low-temperature distillation conditions, perhaps I should have said that a great deal of the volatile matter given off in a coking oven comes out nearest the walls of the oven, in which case it goes into a zone of high temperatures. The sulfur that might be given off by low-temperature distillation under those conditions is probably further absorbed by the high temperature at the exterior of the coking mass.

W. H. Fulweiler, Philadelphia, Pa.—It has been always our understanding that, in coking coal, there are probably three stages in the decomposition. The first takes place at about 400° C.; the second, probably in the range of 600 to 800°; and the third begins to be more apparent at temperatures around 900°. Just what effect, of course, this may have on the sulfur-bearing compounds, I do not know, but by inference we would not expect the sulfur hydrocarbons to be affected at this temperature. It would seem to me that the man who was trying to make sulfur-free gas should study not so much the hydrosulfide as the organic sulfur compounds, which seem to vary in percentage from 3 to possibly 9 or 10. They are the compounds that are going to be difficult to remove by any method that we know at the present time and if we are ever going to make a really sulfur-free gas, they will have to be solved.

F. W. Sperr,* Jr., Pittsburgh, Pa.—Practically all the important conclusions of the paper appear to be demolished by the supplementary

* Chief Chemist, The Koppers Co.

discussion just presented. The title of the paper is misleading. The conclusions derived do not in any way apply to the coking process as any one understands the coking process. They simply apply to the low-temperature carbonization of coal and probably to a limited grade of coal which has been subjected to a somewhat peculiar treatment. As I understand Professor Parr's method, he uses superheated steam to create a uniform temperature through the mass of coal. Reactions taking place in this medium of superheated steam might be profoundly affected by this very medium and it would be altogether unwise to transfer any conclusions that might be derived from experiments of this sort to the coking process as we understand it on a large scale. The methods used consist in the separation of pyrite, marcasite, and the sulfur derived from the resinic and humic matters of the coal. Farther on, the author says that he does not find any marcasite, pyrite, or sulfur derived from resinic material or from humic material, so where are we? In what way can we understand that the methods applied to coal have been successfully applied to coke? There is certainly a lack of facts to substantiate the conclusions derived.

I question, also, the statement that there is no reason, chemically, why analytic methods devised for coal should not apply to coke. That needs a great deal of proof. Taking a simple example, the determination of nitrogen in coke is not by any means as easy as in coal.

The final conclusions are not at all general in application. This whole proposition relating to the disposal of the organic sulfur is apt to be very misleading. One would naturally, in reading this paper, apply it to the coking process. As a matter of fact, organic sulfur is not so easy to break down as it might appear. If it were true that all of the organic sulfur was dissipated in the coking process, we should have a very much greater reduction of sulfur in coke produced from washed coals than actually occurs. Furthermore, in the case of other materials, such as pitch and petroleum, which undoubtedly must contain large percentages of organic sulfur, the cokes produced have large percentages of sulfur. Cokes made from petroleum in the high-sulfur districts will run as high as 6 per cent. sulfur; cokes made from high-sulfur pitch are often quite similar. There is not enough iron or other mineral matter to combine with the sulfur so that it must be an organic combination and derived from organic sulfur.

I question the statement that all the iron goes to the metallic form. Until facts are produced to substantiate that statement, I do not think it should be accepted.

Finally, in the coking process, the secondary reactions play an extremely important part and must not be lost sight of in any attempt to produce a sulfur-free gas. These secondary actions must also be

taken into consideration when any attempt is made to figure out the original distribution of the sulfur.

A. C. FIELDNER, Pittsburgh, Pa.—Certain experiments[1] made in the Bureau of Mines laboratory several years ago showed no metallic iron in ordinary metallurgical coke made in the beehive oven. Several samples of finely pulverized coke were submitted to the Bureau, which were said to contain metallic iron that was thought to be formed in the coking process. Examination showed the metallic iron to be present in the samples, but it was suspected that the chemist for the coke company introduced the iron in his sampling process. He pulverized his samples on an iron bucking board. To investigate this possibility, some of his original lumps of coke were pulverized, at the Bureau, in a porcelain ball-mill containing flint pebbles; when this pulverized material was examined no magnetic particles nor metallic iron were found.

Ten other samples of coke from various sources were then sampled in duplicate, some in the ball-mill and the others on a chrome-steel bucking board. A large amount of magnetic particles was found in all the bucking-board samples and none was found in any of the ball-mill samples. The ash in the bucking-board samples was from 1.4 to 4.5 per cent. higher than in the ball-mill samples, due to the presence of the metallic iron.

In view of these results, I am inclined to doubt the presence of metallic iron in coke made by the ordinary method. It may be that metallic iron is formed from iron sulfide as an intermediate stage and that it is subsequently recombined at the higher temperatures that prevail near the end of the ordinary coking process.

H. C. PORTER.—I am a little afraid my previous remarks might be interpreted as a defence of low-temperature carbonization as the only method for the study of this problem. That, however, was not my intention. I merely wish to emphasize the need of simulating the true conditions of high-temperature commercial processes as closely as possible in experimental work. While in these, the carbonization begins at a low temperature, the volatile products pass through high-temperature zones, and the effect of secondary reactions at these higher temperatures must be carefully considered. The volatile products formed at the low and medium temperatures pass partly through the coke while carbonizing and probably more go by that route than close along the walls so that the effect of the hot carbonizing mass or the coke on the primary volatilized sulfur compounds is to be studied.

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[1] Arno C. Fieldner: Notes on the Sampling and Analysis of Coal. U. S. Bureau of Mines *Tech. Paper* 76 (1914) 55–57.

# Commercial Recovery of Pyrite from Coal

BY S H. DAVIS,* BAXTER SPRINGS, KANS.

(Chicago Meeting, September, 1919)

THE pyrites used in making sulfuric acid in the United States have been largely imported from Spain and Canada, the Spanish imports amounting to nearly 1,000,000 tons per annum in the pre-war period. The greatly increased use of sulfuric acid and the cutting off of these Spanish imports, incident to war conditions, brought about a threatened shortage of sulfur supplies during the war period.

The bituminous-coal mines of certain districts have, for many years, furnished a small tonnage of pyrite in the form of coal brasses. A mechanical concentrator at Danville, Ill., for a number of years, has been treating hand-picked lump pyrite and coal from the picking belt and from the mines and a small plant near Gillespie, Ill., for a few months has been recovering pyrite from washery refuse. Many mines throughout Illinois, Indiana, Western Kentucky, Ohio, and Pennsylvania have shipped an occasional car of the hand-cleaned lump pyrite. Only a very small percentage of the available pyrite has been recovered in this way, however, as usually the miners throw such lumps into the gob with slate and other impurities. It has been estimated that the western Indiana coal field could furnish more than 100,000 tons of pyrite per annum. The present production is very small. The possibility of furnishing the domestic trade with pyrites recovered as a byproduct from coal-mining operations appears attractive but there are certain features difficult to overcome.

Pyrite, to be used in acid making, must meet with certain requirements as to size and purity. Lump ore for grate burners should be under 3 in. (7.6 cm.) and over 1 in. (2.5 cm.) in diameter. Fines for use in mecnanical roasters should be under quarter mesh. The material should be high in sulfur, free from arsenic and phosphorus, and as low in carbon as possible. The pyrite obtained from coal can be made to meet all the above requirements, but it is difficult to remove all the carbon. The pyrite in coal occurs as bands and nodules of varying thickness and size and of comparative purity, but mixed with this is more or less web sulfur. The web sulfur carries admixed coal, which may make the concentrate run up to several per cent. carbon. This makes the concentrates subject to firing, causes heavy consumption of niter, and lowers the acid-plant capacity, due to dilution of gases. The hand-cleaning methods

* Illinois Pyrite Co.

and the present plants have failed to entirely overcome this, so that it may be necessary at the acid plant to mix this with other ore.

A very large tonnage of pyrite is annually being thrown in the waste in the coal mines but there are certain difficulties in making this material available. Hoisting the crude pyrite from the coal mines means, in most instances, a serious handicap to the coal-mining operations; injury to the chutes and screens are met with, which necessitates separate loading facilities; and the miner must coöperate. These difficulties have not been overcome at most of the larger mines. In treating washery refuse, this difficulty is not met, but there are few washeries at which the refuse contains sufficient pyrite or where the refuse is in large enough quantity to make it attractive.

The recovery of pyrite from coal will not meet with any great expansion, it is felt, so long as the Louisiana brimstone can be obtained at present or pre-war prices. The acid plant that uses pyrite must have a greater investment in burner and dust-settling equipment than where brimstone is used. It is true, however, that in certain locations farthest removed from the source of supply of brimstone and near the coal fields, coal pyrite can be advantageously used.

## DISCUSSION

EDWARD HART,* Easton, Pa. (written discussion†).—In 1895 I visited the chemical plant of the Messrs. Chance at Oldbury, England, under the guidance of Mr. France, the manager. In the stock house I saw a pile of coal brasses with adhering coal and said to Mr. France, "I should think you would get dark acid." "Yes," he replied, "our acid has always been rather black and our customers have grown used to it. Some time ago we ran out of brasses and could get only Spanish pyrite. Our product became white and the customers began to complain, so I put a little sugar in each carboy."

E. A. HOLBROOK,‡ Pittsburgh, Pa. (written discussion§).—Mr. Davis says, "A very large tonnage of pyrite is annually being thrown in the waste in the coal mines of this country." This quantity is much larger than is generally realized by people not intimately associated with coal mining. At the beginning of the war this country seemed threatened with a shortage of pyrite. The U. S. Bureau of Mines conducted a general investigation of the pyrite resources of the country in order to ascertain what material was commercially available. The general pyrite investigation

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‡ Superintendent of Pittsburgh Station, U. S Bureau of Mines

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was in charge of H. A. Buehler, with the subsection on coal pyrite in charge of E. A. Holbrook.

In May, a conference of State Geologists, the Bureau of Mines, and a representative of the U. S. Geological Survey, was held at Urbana. Those present included Messrs. Buehler and Holbrook of the Bureau, P. M. Smith of the U. S. Geological Survey, and representatives from the State Geological Surveys of Iowa, Missouri, Tennessee, Illinois, Indiana, Ohio, West Virginia, and Pennsylvania. It was decided that each State Survey should send out a field man in its own state and that he should visit, so far as possible, each bituminous coal mine in his state; he should actually go to the working faces underground and there, by measurement, make an estimate of the amount of pyrite available through the mining of the coal. The conference visited a coal mine near Danville, Ill., and while there adopted a common method of sampling and estimating the pyrite. Briefly, several sections were to be measured underground at the face of each mine. The average thickness of pyrite in the seam was to be noted and multiplied by three. Thus the assumption was made that the pyrite was three times as heavy as the coal, a very close approximation. If for any reason estimations of pyrite had to be made from amounts already mined, proper allowance should be made for coal adhering to pure pyrite nodules. A standard report form was adopted.

The Bureau of Mines agreed to keep a chemist at Urbana to whom the various State Geological Survey field men might frequently send samples of pyrite for sulfur analysis, to determine uniformly the grade of the various coal pyrite occurrences. It was arranged that Mr. Holbrook should visit each of the working field men and coöperate wherever possible in securing uniformity of work and quick results. The Bureau also agreed to run mill tests on ton lots of crude coal pyrite which might be submitted in anticipation of a commercial plant working on the specific pyrite tested. A list of the tests run by the Bureau on this work is given here.

### List of Coal Pyrite Tests Made and Reported on by Middle West Station, U. S. Bureau of Mines, Feb. 1–Nov. 1, 1918

| | | Tests |
|---|---|---|
| 1. | Picking belt refuse from Middle Forks Mine, Benton, Ill. | 5 |
| 2. | Washery refuse from Middle Forks Mine, Benton, Ill. | 3 |
| 3. | Washery refuse from Gillespie, Ill. | 2 |
| 4. | Crude coal pyrite from No. 3 seam, Indiana | 2 |
| 5. | Crude coal pyrite from No. 5 seam, Indiana | |
| 6. | Crude coal pyrite from No. 6 seam, Indiana | |
| 7. | Crude coal pyrite from Earlington, West Kentucky | |
| 8. | Washery refuse from Earlington, West Kentucky | |
| 9. | Washery refuse, Southern Coal & Coke Co., East St. Louis, Ill. | |
| 10. | Crude coal pyrite from Mulberry, Kans | |
| 11. | Washery refuse, Keota, Missouri | |
| 12. | Crude coal pyrite from New Philadelphia, Ohio | 2 |

| | Tests |
|---|---|
| 13. Crude coal pyrite from Mines of Bon Air Coal and Coke Corporation | |
| 14. Bon Air, Tennessee | |
| 15. Washery refuse from Saginaw, Michigan | |
| 16. Crude coal pyrite from Sunfield, Illinois | |
| 17. Washery refuse from Tyler, Pa | 2 |
| 18 Crude coal pyrite from Winburne, Pa. | |

About 70 per cent. of these tests were successful and indicated that a commercial grade of pyrite could be easily concentrated and recovered from the crude material submitted.

During the summer, H. F. Yancey, assistant chemist at the Urbana station, analyzed for sulfur several hundred samples of pyrite submitted by the field men; many of these samples were also analyzed for arsenic, carbon, and phosphorus. Results of this chemical work have been compiled and will shortly appear as an article in the technical press. The results of the general tests at Urbana just noted have in each case been sent to the men interested, and at present are being worked into a bulletin on this subject which will be accompanied by the detailed summary and estimates of the work of the field men from each state. In addition to the states already noted, some work was done by the Bureau in the Kansas, Michigan, and Kentucky coal fields. The State of West Virginia did not coöperate because the Consolidated Coal Co. of Fairmont, W. Va., had already gathered statistics on coal pyrite in that state and had erected a sulfuric-acid plant at Fairmont for coal-pyrite utilization. Consequently, we have no knowledge of the quantity or purity of the coal pyrite in West Virginia.

Following is a summary of the estimated possible early production of coal pyrite in all of the principal coal-mining states of the East. This summary is based only on pyrite that can be recovered with purity in sulfur of more than 40 per cent., and including only mines that could produce 1 per cent. or more of their coal production as coal pyrite. It would be fair to assume that this production could be obtained provided a price of 20 c. per unit on a basis of 40 per cent. sulfur could be made for this material f.o.b. at the mines.

| | Tons per Year | | Tons per Year |
|---|---|---|---|
| 1. Kansas | 125,000 | 6. Kentucky | 25,000 |
| 2. Missouri | 175,000 | 7. Tennessee | 56,000 |
| 3. Iowa | 140,000 | 8. Michigan | 12,000 |
| 4. Illinois | 238,000 | 9. Ohio | 235,000 |
| 5. Indiana | 250,000 | 10. Pennsylvania | 200,000 |

Total possible for Eastern coal fields.... .... 1,456,000 tons pyrite per year.

While the early close of the war and the unexpected developments of the sulfur fields of Louisiana and Texas stopped any general progress toward the utilization of this vast yearly possible supply, the knowledge that it is available at any time of future need makes this country independent of foreign sources of pyrite.

# Low-sulfur Coal in Illinois

BY GILBERT H. CADY,* URBANA, ILL

(Chicago Meeting, September, 1919)

EXTENSIVE sampling of coal in Illinois during the past 10 or 12 years by the State Geological Survey, in coöperation with various organizations, such as the U. S. Bureau of Mines, the University of Illinois, and the Illinois Coöperative Mining Investigations, has made possible the delineation of two areas of low-sulfur coal in the southern part of Illinois. The sulfur content is less than 1.25 per cent., so that, if otherwise suitable, these coals can be employed for metallurgical uses and for the manufacture of water-gas and retort gas. One of these areas is small and lies in Jackson County, near Murphysboro, the other is much larger and includes a large part of the famous Franklin County field.

A small area of No. 2, or Murphysboro, coal has been worked for many years near the town of Murphysboro, Jackson County, Ill. In two mines, at least, the coal has a sulfur content of less than 1.25 per cent. It is doubtful, however, whether this field will ever be a source of large tonnage as the total area underlain by low-sulfur coal in workable thickness is probably less than 15 sq. mi. (24.14 sq. km.), and a large part of it has already been worked out.

The location of the area of low-sulfur coal in the Franklin County field is shown in the accompanying map. The small area underlain by the Murphysboro low-sulfur coal is shown near the town of that name in the central part of Jackson County. The larger area lies in the west side of Franklin County, extending also about 6 mi. (9.65 km.) south into Williamson County, about 4 mi. (6.43 km.) west into northern Jackson and western Perry County, and northward an undetermined distance into Jefferson County. All but the northern limit of the area is fairly well defined by sampling in numerous mines. The inner cross-lined area is underlain by coal having less than 1 per cent. sulfur, the outer boundary surrounding the area underlain by coal having less than 1.25 per cent. sulfur.

The coal mined in the district is No. 6 or Herrin coal, commonly known as the Carterville or Franklin County coal. The bed has a thickness varying from about 8 ft. (2.4 m.) on the border of the low-sulfur area up to more than 10 ft. (3 m.) in the central portion, locally having a thick-

* Illinois State Geological Survey.

ness of 14 to 15 ft. (4.2 to 4.5 m.). The sulfur content, in general, decreases as the thickness of the coal increases.

Another peculiarity is the variation in the character of the roof that accompanies the variation in thickness. Near and beyond the border of the low-sulfur area there is a limestone cap-rock within about 25 ft. (7.6 m.) of the bed, but in the central part of the area the cap-rock is

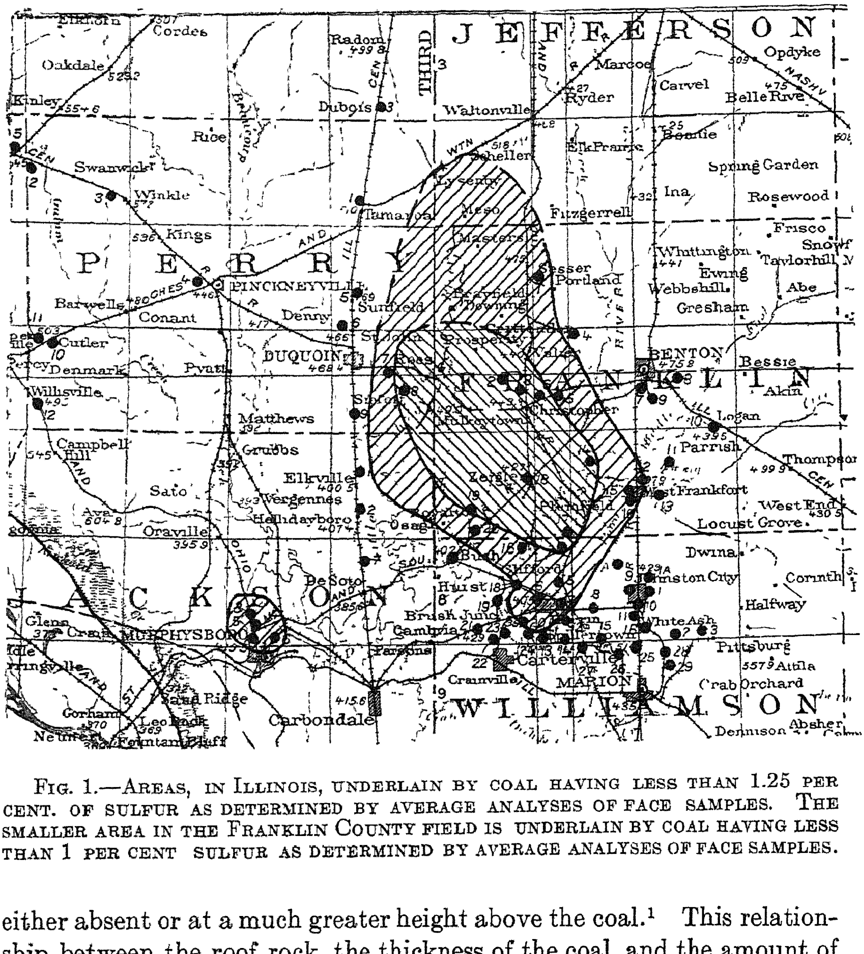


Fig. 1.—Areas, in Illinois, underlain by coal having less than 1.25 per cent. of sulfur as determined by average analyses of face samples. The smaller area in the Franklin County field is underlain by coal having less than 1 per cent sulfur as determined by average analyses of face samples.

either absent or at a much greater height above the coal.[1] This relationship between the roof rock, the thickness of the coal, and the amount of sulfur present seems to hold consistently throughout the field. There is also a decrease in the interval between No. 6 and No. 5 coals and an increase in the interval between No. 6 and No. 9 coals operating geographically across Franklin County field the same as the decrease in

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[1] G. H. Cady: Coal Resources of District VI, Ill. Coal Mining Investigations *Bull* 15 (1916) 29–47.

the sulfur content. These stratigraphic variations accompanying the chemical variation make it possible to estimate roughly the character of the coal even from drill records and to determine the approximate extension of the low-sulfur field in areas not yet mined.

## DISCUSSION

R. D. Hall,* New York, N. Y.—How does the amount of moisture in that coal compare with other Illinois coal?

F. W. DeWolf,† Urbana, Ill.—It is intermediate or a little bit below the average in moisture, but it is in no sense our lowest moisture coal. The No. 5 bed, principally in the Harrisburg district, is noticeably lower in moisture, and on the "as received" basis, will commonly run as low as 5 per cent.; I am not sure that the commercial output would run as low as that. In contrast, the Franklin-Williamson Co. coal, while varying from place to place, will average near 9 per cent., and the coal of the Northern or longwall field, up around LaSalle, will run 15 per cent.; I am speaking entirely from memory.

L. V. Rice, Chicago, Ill.—Is a distinction made between the sulfate sulfur and the pyrite in the beds in figuring the total quantity of the sulfur and the original? There is, I believe, more or less banding of pyrite. Is it believed that that comes from above?

F. W. DeWolf.—I must refer you to the author for an answer to that question. In general, that sulfur to which I referred occurs as isolated nodules or as thin bands and binders in the coal. There is also, along the vertical cleavage bands in the coal, some gypsum; in other places, some calcite. In the ordinary determination of sulfur, the result includes sulfur present as sulfate, but I think the bulk of it is pyrite or marcasite in the form of nuggets, bands, or disseminated as tiny specks in the body of the coal.

S. A. Taylor, Pittsburgh, Pa.—Has it been the experience of the men here that with the shale roof you find lower sulfur coal and with the solid rock roof you find higher sulfur coals?

C. W. Smith, Chicago, Ill.—It seems to me that the shale thickens in this district. Going up on the anticline, the roof of this coal consists of a black shale overlain with a cap. Going down the anticline, there is a gray shale that thickens. In the basin this shale is lost but there is from 80 to 90 and sometimes 110 ft. of gray shale between the limestone and the coal. In the mines along the anticline, this gray shale thickens and comes in there. The sulfur in the coal gets less as the shale thickens

* Associate Editor, *Coal Age*. † Chief, State Geological Survey.

until a certain point of the anticline is reached where there is a shale roof and no limestone in the formations above the coal, but the sulfur increases as you go east. It does not seem to bear any particular relation to the shale roof on the east side but it does on the west.

S. A. Taylor.—Do you remember the conditions of the body roof with regard to the seam?

F. W. DeWolf.—Above that shale, there is, in the area to the east, a heavy sandstone 30 or 40 ft. thick; in the area described by Mr. Cady, I believe the character of beds above the shale varies from shale and sandy shale to sandstone.

C. W. Smith.—Do you know anything about the sulfur content, whether the roof is shale or whether it is sandstone in that seam? It seems that the shale is of a more sandy nature with the sulfur content high than where it is low.

F. W. DeWolf.—I do not.

L. V. Rice.—Has the Illinois Survey come to a conclusion whether, in these continuous bands you have found in certain localities of Illinois, the metallic pyrite was deposited at the time of formation of the bed or subsequently?

F. W. DeWolf.—Unless the answer to that question is contained in the paper, I am afraid we have no answer for it. Mr. Cady has been working on this matter for the last 2 or 3 years and it has not been the function of any of the rest of us to speculate on that. My impression is that it is a radical view to hold that those bands, which are interstratified with the coal and apparently have marked persistence, have been introduced in a secondary way through a pervious roof. Unless Mr. Cady has reached such a conclusion in his paper, I should say the chances are against it.

C. W. Smith.—This seam, in the southern part, seems to bear out the statement that the sulfur varies in different parts of the seam. The sulfur here seems to be mostly in the top coal, where a large part of it seems to be in vertical seams developed on the cleavage faces in the coal. That would seem to lend some color to the theory of the secondary deposition of the sulfur in the seam, but it is alike under a lime roof or under a shale roof.

R. D. Hall.—I do not know much about the geology of Illinois, but as I have been thinking I remembered that the Big Muddy region had been subject to some faulting and thought that the resulting heat may have accounted for the low percentage of sulfur in some of the coals of that section. But knowing that the Franklin County field is quite

flat, I cannot thus account for the low percentage of sulfur found in that field. There may be earth movements of which we know nothing, and if so, we might find evidence of it in the moisture content.

F. W. DeWolf.—Was it your thought that where there had been faulting, there would be a higher moisture content?

R. D. Hall.—No; I would expect a lower percentage of moisture.

F. W. DeWolf.—The facts partly bear out the theory and partly contradict it. The most faulting is in the Saline and Gallatin Co. fields, where there are the lowest moisture and the highest sulfur contents. The area of low-sulfur coal described by Mr. Cady seems not to have been notably faulted or compressed.

C. W. Smith.—It seems that the Franklin Co. field roughly follows the Duquoin anticline, but whether it is possible to draw any parallel between that fact and Mr. Hall's theory is a question. The lowest sulfur coal seems to be the closest to the Duquoin anticline. As you go east, the sulfur gets higher. As you go up on top of the anticline, it also gets higher.

The Chairman (H. H. Stoek, Urbana, Ill.).—Is not the low-sulfur coal found practically in an elliptical space with the higher sulfur coal all around it?

C. W. Smith.—It is, but the lowest sulfur is right next to the anticline and it gradually becomes higher toward the east.

Homer T. Darlington, Natrona, Pa.—I went into Clearfield, Pa., to get some of this sulfur that is so troublesome. I said to a mine foreman, "Have you any sulfur?" He said, "Not a bit. It is all on the nail heads down there." I was interested in getting coal pyrite for the manufacture of sulfuric acid, which requires that the pyrite be of a size that can be handled by hand, which it would pay to pick from the coal. We found that the sulfur balls of a commercial character came from the seams where the lowest sulfur coal was. It is of particular interest to the acid manufacturers of the country. Pyrite from coal, as you know, in different localities, is called sulfur balls, nigger heads, and coal brasses.

Marcasite has been used probably for the last 30 years in the manufacture of sulfuric acid. I believe that the first pyrite for the manufacture of sulfuric acid came from the new Philadelphia field. It comes in a consistent band from 1 in. to 1½ in. thick, halfway up the face of the coal, not in the roof and not in the foot of the coal. That band breaks very clean from the coal and the pyrite, as shipped, probably lists less than 2 per cent. carbon. A great many thousands of tons of sulfuric acid are made from the pyrite of that particular coal. In Clearfield County, Pennsylvania, and adjoining districts, the coal pyrites shipped

for the manufacture of sulfuric acid have appeared as lenses in the coal, generally, I might say, from the bottom of the coal. In one run, 1 ft. of coal averages 50 per cent. pyrite. That would mean it would run 25 per cent. sulfur, approximately. This bottom foot of coal is not mined. The pyrite in the Illinois field is very similar, I think, to that in Clearfield County. I would say that the Illinois field I am familiar with is in the Danville district.

The pyrite in the Southern Indiana field, very thoroughly investigated by the Bureau of Mines and the State Geological Survey of Illinois, showed sulfur balls occurring not only in the coal but in the roof and in the clay or the foot of the coal. Some of these sulfur balls weigh several hundred pounds. In the Illinois field, the sulfur balls are almost pure when the coal is broken from them. In Southern Indiana, the sulfur balls from the roof often have large crystals of calcite in the center; some of those are 1 in. in length. I have found some from Pennsylvania that analyzed as high as 52 per cent. sulfur, which is practically pure pyrite.

REINHARDT THIESSEN, Pittsburgh, Pa.—When somebody asked the question why sulfur existed to a larger extent in some parts of the seam than in others, it occurred to me that it would be of a very great benefit if those interested knew the constitution of coal better. The publications prepared by the Bureau of Mines on that subject are still on the press but will be out in a relatively short time. A knowledge of the composition of coal would give a clearer understanding of the distribution of the different constituents. In general, a large part of the coal is composed of that part of the plants which at one time was wood. We can still see the woody parts in the coal. In between these strips we find an attritioned matter, a matter that has been degraded into a finely divided matter. Some coal seams are composed almost entirely of attrition matter; others almost entirely of woody matter. Some coals, like the coal from Shelbyville, for example, from Bed No. 2, are composed almost of 50 per cent. of spore-xines. Why?

C. H. MACDOWELL, Chicago, Ill.—With regard to the manufacture of sulfuric acid from coal brasses, the Illinois University did some good research work in the way of removing carbon from pyrites in coal, and the result of their work was published in pamphlet form. The Bureau of Mines took up the matter with coal producers and the Chemicals Committee of the War Industries Board prepared a list of sulfuric acid manufacturers which it sent to coal miners, endeavoring to interest the coal miners of the Middle West in the production of pyrite and marcasite. The labor situation and other perplexities did not permit much to be accomplished. It is necessary, in order to receive a full production of sulfuric acid from a plant burning coal brasses, to remove the carbon as far as possible. If this is not done, production falls off and a much larger

percentage of nitric acid is consumed. Now that peace has come, there should be a good market for a well-cleaned coal pyrite and the coal operators should give this matter thought.

Pyrites are burned both in lump and as fines. Unfortunately, at the moment, many of the Mid-Western plants are equipped to burn lump ore and the larger part of the production from coal would come as fines. At the moment much sulfuric acid is produced from sulfur. It is figured that an acid manufacturer can pay 4 to 5 cents a unit more for sulfur than for pyrites on account of the saving in labor, bigger yields, and lower niter consumption

H. T. Darlington.—The so-called coal pyrites contain from 15 to 20 per cent. carbon. In the sulfuric-acid plant, there are two kinds of ore that will pass through a 3-in. screen. There is also fine ore burned that will entirely pass through a ¾-in. screen. There are a great many lump-ore burning plants. Lump-coal pyrites from Central Pennsylvania and New Philadelphia field, Ohio, will contain less than 3 per cent carbon.

It is impossible to burn as much coal pyrites in a kiln as it is true pyrites, or pyrites free from coal, for the coal, in burning, produces so much heat that if a large charge is added to the furnace it will melt the pyrite. In the crushing of this coal pyrite, material that goes through the screen contains a large part of the coal. I have never known of any commercial use of this coal pyrite prior to the work done by the Bureau of Mines. I understand a number of coal-washing plants have since taken up the washing of coal pyrites and the reduction of coal pyrites with a low carbon content.

If one were to attempt to concentrate coal pyrites directly from the mine for sulfuric-acid manufacture, the concentrate would be rather high in carbon. If, however, the pyrite is allowed to weather, the coal will easily separate; it can be easily separated with a coal washer. Fines can be produced of a very high grade, free from carbon, and are commercially burned today in a number of sulfuric-acid plants. The practice is so rapidly changing that one cannot tell, from day to day, what the practice is going to be a month from now and a great many of the Eastern sulfuric-acid producers have not given much attention to this material. There are in Pennsylvania possibilities of the production of fines in excess of 50,000 tons a year. The possibilities of lump-coal pyrites is probably less. The coal pyrites coming on the market are largely from Illinois, Indiana, Ohio, and some from Tennessee.

C. H. MacDowell.—One of the reasons for reduced capacity in burning coal pyrite for the making of acid is the diluting effect of $CO_2$ in the plant, as well as the difficulty of temperature control and increased niter consumption.

F. W. Gray,* St. Anne de Bellevue, P. Q., Canada.—In Nova Scotia there is a large submarine coal field. The coal lies in a basin, on the edges of which the ash content and the sulfur content of the coal are very much higher than in the center. It has always seemed to me that the method of deposition of the original constituents which formed the coal had an important bearing on the percentage of sulfur. In the center of the field, where the coal seams are thick, they contain, as a rule, much less sulfur than on the sides of the basin where they are thinner. In one instance the roof coal is so high in sulfur that 6 in. of the coal is left; the bottom portion of the seam also is high in sulfur and the center of the seam is fairly good. Six or seven seams occur in a depth of possibly 1200 ft. These seams vary in thickness and also in sulfur content and there seems to be no particular reason for this. The Phalen seam is a very good coking coal and contains a low percentage of sulfur. The seam below it, which is about 4 ft. 6 in. thick, contains a much higher percentage of sulfur, and I can trace no relation between any geological position of the seams and the percentage of sulfur present.

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* Editor, *Iron and Steel of Canada.*

# Low-sulfur Coal in Pennsylvania

BY H. M AND T M. CHANCE,* PHILADELPHIA, PA.

(Chicago Meeting, September, 1919)

THE term "low-sulfur coal," as used in this discussion, is limited to coals containing less, or very little more, than 1 per cent. sulfur. For certain purposes it might be advantageous to include coals that contain as much as 1.25 per cent. sulfur, because such coals have been considered as available for making metallurgic coke of fair grade; but the same reasoning might apply to coals containing as much as 1½ per cent. sulfur, if facilities exist for utilizing such coal by admixture with coal containing less sulfur.

In Pennsylvania, it is not possible to define the exact geographic limits of the areas in which coals relatively low in sulfur exist, because variations in the percentage of sulfur occur irregularly and often abruptly. It is not uncommon to find areas, more or less circumscribed in extent, in which the coal is relatively high in sulfur although all the surrounding territory contains low-sulfur coal; and, similarly, small areas of low-sulfur coal exist in districts where the coal is normally high in sulfur. These generalizations apply, with few exceptions, to the whole area underlaid with bituminous coal in the State of Pennsylvania, and also to each of the individual beds of coal in this state.

Pennsylvania is poorly supplied with coals of the low-sulfur class. We find, however, as an offset to this condition, very large areas in which coal ranging from 1 to 2½ per cent. sulfur exists in beds of good workable thickness. This coal, by washing, can be reduced in sulfur low enough to make coke of a high grade. While a considerable tonnage of workable low-sulfur coal remains in the Pennsylvania bituminous districts, it has been necessary to turn to the areas of higher sulfur coal for the supply of metallurgical coke. This utilization of the higher sulfur areas has been made possible either by washing or by selective mining.

*Washing.*—Fine crushing of the entire feed is not generally necessary to the successful washing of Pennsylvania coals. This is due to the fact that the pyrite is rarely finely and uniformly disseminated throughout the coal but is usually confined to high-sulfur benches in the bed or to concentrations on the cleavage planes of the coal. Hence, by graded crushing and sizing it is frequently possible to at once reclaim those portions of the coal bed containing coal that is relatively free from pyrite.

* Consulting Mining Engineers.

This is especially true of coals that contain concretions of pyrite or "sulfur balls," and these are often amenable to treatment by mechanical cleaners, such as the Bradford breaker, without recourse to washing. In those cases in which a sulfur content approaching 3 per cent. is caused by finely divided pyrite uniformly disseminated throughout the coal bed, it is practically impossible to produce a coal of low-sulfur grade by washing or mechanical preparation; and these coals, when washed, must still be mixed with relatively lower sulfur coal if metallurgical coke of high grade is to be made.

A careful study should always be made of the physical occurrence and distribution of the pyrite to determine in advance the possibility of improvement by washing. The present tendency is toward doing away with close sizing unless this is essential to satisfactory reduction in ash. Many of the difficulties presented by the sludge problem can be eliminated by graded washing. There are many cases in which the washing of large-size coal, comparable to the jigging of stove and egg anthracite, may yield a product of low-sulfur grade, in which case the unwashed fine coal, or slack, containing the pyrite freed from the joint planes of the coal, may be utilized as steam coal, or may be washed separately, with or without crushing to smaller size. The adoption of this method, by avoiding crushing down of all the coal, reduces the quantity of fine coal and sludge to a minimum.

*Selective Mining.*—The intensive mining of large coal areas, in what have commonly been regarded as low-sulfur districts, has conclusively demonstrated the necessity of thorough exploration in each particular field before reliance can be placed in any estimate of average sulfur content. In localities where the coal is under considerable cover, a few low-sulfur drill cores have, at times, been the sole evidence on which the success or failure of large-scale mining operations has been predicated; and even where miles of outcrop are accessible, the exploratory work undertaken to prove the existence of low-sulfur coal has too often been confined to work too close to the outcrop to afford reliable data as to the sulfur content of the unweathered coal.

It should be remembered that a diamond-drill hole may penetrate a portion of the coal that is far below (or above) the normal in sulfur content. This is more especially true in fields in which the pyrite is concentrated in parts of the bed or in lenticular masses than in those in which a more uniform dissemination exists. In several cases operating collieries that were considered to have a long future as producers of low-sulfur coal have been forced to wash or to mechanically prepare their coal after extensive mine development or other exploration had proved the non-existence of large and continuous areas of coal of this grade. We believe that no dependence should be placed on the supposed existence of low-sulfur coal over any large area in Pennsylvania, unless there has

been a systematic exploratory campaign of sufficient scope to preclude the inadvertent inclusion of any considerable area of high-sulfur coal. Such explorations should combine, if possible, outcrop development and drilling, the latter being laid out in as regular and methodical a manner as practicable. We cannot too strongly emphasize the fact that the regional low-sulfur characteristics of many West Virginia districts find practically no parallels in the coals remaining to be mined in Pennsylvania and that, therefore, a far greater amount of precise data is required in determining the sulfur content of the Pennsylvania coals.

The low-sulfur districts that exist in the Pennsylvania coals are generally areas that contain islands of low-sulfur coal surrounded by coal of higher sulfur content. These islands are comparable to the ore chutes in metalliferous veins and may extend for several miles. They are, however, quite frequently small and of irregular outline, and it is within such areas that selective mining has generally been practised. In such a region, it may happen that one room out of every two or three will produce coal of low-sulfur content, so that the coal coming from the low-sulfur rooms can be separately shipped or utilized by coking as low-sulfur fuel. Obviously, such a mine may be classed as either a high or low-sulfur operation, depending on which grade of coal is in the preponderance.

We have discussed these correlated questions of washing and of selective mining because we believe that the future of metallurgical fuel production in Pennsylvania is inseparably bound to them. In many instances washing of the whole colliery output may be avoided by careful selection in mining, selective either as to portions of the coal bed itself or as to districts in the mine. In some cases where selective mining is practised as the method best adapted to the conditions, the output of low-sulfur coal may be increased by the washing of the high-sulfur coal, which otherwise would be sold for use as steam fuel. This discussion is not intended to include the anthracite districts, because the anthracite of Pennsylvania is almost always very low in its sulfur contents.

It may perhaps be best to commence by describing the character of the coal of the several beds of the bituminous-coal series, commencing at the base of the coal measures with the Sharon coal bed and proceeding up through the series to the Washington coal bed near the top of the upper productive coal measures, and having described the individual beds separately, it may then be of interest to take up the principal coal-mining districts and attempt to define the areas in which low-sulfur coal exists.

## Coal Beds of the Conglomerate Series

*Sharon Coal Bed.*—This coal bed, lying at the base of the conglomerate measures, is the lowest coal bed of the bituminous-coal series in Pennsyl-

vania. It is workable in Mercer and Lawrence Counties, and in a few other localities small areas of it have been found. It is almost always low in both sulfur and ash, but is not continuously workable over any large area. It is probably identical with what is known as the "Block" coal of Ohio and Indiana, where it has been largely worked under the names "Youngstown Block" and "Brazil Block." It seems to exist in pockets and is thought to have been formed in swamps. It is not available as a large source of supply.

*Mercer Coal Beds.*—These two coal beds are rarely of workable thickness but are valuable in parts of Beaver, Lawrence, Mercer, Crawford, and a few other counties. Occasionally they contain coal low in sulfur but this is not usually characteristic of coal from either of these two beds. These coal beds are within the conglomerate measures, underlying the Homewood sandstone, or top member of the Conglomerate Series in Western Pennsylvania—the old "No. XII" of the Pennsylvania Geological Survey.

## Coal Beds of the Lower Productive Series

*Brookville Coal, Coal Bed A.*—This coal is one of the most persistent coal seams of the lower productive coal measures and is of workable size over relatively large areas. It is not usually a bed of superior quality, and is often high in both ash and sulfur contents. Very few areas have been developed in which this bed produces coal low in sulfur. In some cases where low-sulfur coal is claimed for this bed, it is questionable whether the bed mined may not in fact be either the Clarion or Lower Kittanning coal bed.

*Clarion Coal, Coal Bed A'.*—This coal bed is rarely of workable thickness or quality in the counties lying along the eastern borders of the bituminous-coal district in Pennsylvania, and although coal of fair thickness and relatively good quality is thought to be mined from this coal bed, it is possible that the coal mined may in fact be the Lower Kittanning coal, coal bed B. In the western counties of Pennsylvania, especially in the counties adjacent to Clarion County (where this coal bed was first recognized and named) it is usually thin, but is persistent as a workable seam over large areas and occasionally furnishes coal low in sulfur. The areas producing coal of this latter grade are, however, circumscribed and unimportant.

*Lower Kittanning Coal, Coal Bed B.*—This is perhaps the most important coal bed of the lower productive coal measure in Pennsylvania. It is of workable thickness in parts of every county in which it lies above water level. It ranges in thickness from 3 ft. (0.9 m.) up to about 6½ or 7 ft. (1.9 to 2.1 m.) averaging perhaps 4 ft. (1.2 m.). Its quality varies greatly and in comparatively short distances. It furnishes low-sulfur coal of very high grade in parts of Cambria and Somerset Counties

and supposedly in Tioga and Bradford Counties, but the identity of the coal as bed B in these two counties is not positively proved. In Somerset and Cambria Counties (and in a part of Indiana County) it is known as the "Miller" seam, and is renowned as a "smokeless" coal of high grade, being low in both sulfur and ash.

With the exception of these districts coal from this bed is usually rather high in sulfur; and in districts even immediately adjacent to those in which the low-sulfur coal is found, the bed is often relatively high in sulfur. Increase in the quantity of sulfur is usually accompanied by a marked increase in the percentage of ash, and also in the slate and other partings that contribute to increase the percentage of ash in the coal as shipped.

*Middle Kittanning Coal, Coal Bed C.*—This coal bed, while a persistent member of the coal series, is rarely of workable thickness, and where it is found workable, the area is usually circumscribed. It was first recognized and named as a persistent member of the series in the central part of Butler County, where it is a bed of fair grade and furnishes some coal of low-sulfur content. It has also been developed to some extent in Clearfield County, notably at Morrisdale, Chest Creek, and a few other localities. It is likely to prove of greater value in the future when economic conditions will permit the mining of the thinner seams of coal

*Upper Kittanning Coal, Coal Bed C′.*—This coal bed, although a persistent member of the coal series, is of workable size only in a few localities. It has been developed in the counties bordering the Ohio line and has been mined in Butler, Jefferson, Clearfield, Elk, Cambria, and Somerset Counties. In Somerset County, a considerable area has been developed in which this bed furnishes low-sulfur coal of high grade, coal quite similar to that furnished by the "Miller" seam in Cambria County. With the exception of this area in Somerset County, this coal bed cannot become an important source of low-sulfur coal supply, because the areas furnishing coal of this grade are usually quite small.

*Lower Freeport Coal, Coal Bed D.*—This coal bed is a seam of more or less importance from the eastern edge of the bituminous-coal field westward to the Ohio line, and is found to be workable in parts of every county in which it is above water level. It usually furnishes coal of relatively high grade, but there are no large areas known that can be regarded as promising an important source of supply of low-sulfur coal. This bed reached its highest and most valuable development in Clearfield County, where in the Houtzdale-Morrisdale district it was from 4 to 6 ft. thick and furnished coal uniformly low in sulfur and low in ash. It was locally called the Moshannon Bed. This district is now practically exhausted. In parts of Somerset, Cambria, Indiana, Jefferson, Armstrong, Clarion, and perhaps Butler Counties, circumscribed areas exist in which this coal is a low-sulfur coal. This is the coal bed worked

at Punxsutawney and Reynoldsville in Jefferson County. It is of good quality but of reduced thickness in Elk County. In the immediate vicinity of Punxsutawney, this coal bed furnishes high-grade low-sulfur coal.

*Upper Freeport Coal, Coal Bed E.*—This coal bed is perhaps the most persistent of any of the seams of the lower productive coal measures, being found in practically every county from the face of the Allegheny Mountains westward to the Ohio line. It is not, however, always of workable thickness and is rarely a low-sulfur coal. The areas in which it has been found to contain a small percentage of sulfur are usually local and circumscribed in extent. A district in Butler and Allegheny Counties in which this coal lies below water level, and in which development is by boring and shafting, may perhaps furnish a considerable tonnage of low-sulfur coal from this bed.

### Coal Beds of the Lower Barren Series

Three coal beds of the lower barren measures, the Gallitzin, the Price, and the Berlin coal beds, have been found of workable thickness over small or circumscribed areas, and the developments upon these beds have in some cases shown the presence of low-sulfur coal. These areas, however, are too small to become important sources of supply of low-sulfur coal.

### Coal Beds of the Upper Productive Series

The upper productive coal measures contain five coal beds of importance and a number of other beds that, while workable in a few localities, are usually small and unimportant. These five beds, beginning at the bottom, are the Pittsburgh, Red Stone, Sewickley, Waynesburg, and Washington; they will be considered in this order.

*Pittsburgh Coal Bed.*—This coal bed is below water level in the central and western parts of Greene County and over much of Washington County and underlies large areas in Fayette, Allegheny, and Westmoreland Counties, and is found in detached areas of relatively small extent in Indiana and Somerset Counties.

As a result of diamond-drilling exploration, it has been claimed that this bed will furnish large areas of low-sulfur coal in Greene and Washington Counties. While this claim is doubtless well founded, it is also claimed that some drill holes indicate the presence of large areas in which the sulfur is too high to enable us to class the coal as a low-sulfur coal. In parts of Washington, Allegheny, Westmoreland, and Fayette Counties, where this bed has been largely developed by actual mining, it is an important source of supply of relatively low-sulfur coal. In Westmoreland and Fayette Counties, the limitation of these areas has

been well defined by mining operations, which exist in practically all of the territory known to be of this character. But large areas exist in Washington County, where the coal lies at a considerable depth beneath the surface, in which the sulfur content has not been definitely fixed. The detached basin of this coal in Somerset County has furnished coal of low sulfur content but this field contains no undeveloped territory

*Red Stone Coal Bed.*—This bed is of workable size in parts of Washington, Greene (?), and Somerset Counties, but it is not likely to be a source of any considerable tonnage of low-sulfur coal.

*Sewickley Coal.*—This bed has been found of workable size in parts of Greene and Fayette Counties. The data in hand indicate that it is usually relatively high in sulfur.

*Waynesburg Coal.*—This coal bed is of some importance in Greene and Washington Counties, having been found of workable thickness over an area of considerable size. This area may contain some coal sufficiently low in sulfur to be classed as low-sulfur coal, but it is not generally regarded as a promising source of supply of coal of this grade.

*Washington Coal.*—This coal bed is of importance mainly in Washington County, being a bed of good size in the central part of that county. It is not thought likely to prove a source of supply of low-sulfur coal.

## Districts in Which Low-sulfur Coal Exists

We will not attempt to describe or to delimit all of the many localities in which small areas of low-sulfur coal have been found, but will confine our description mainly to areas large enough to become important sources of supply for fuel of this grade. In describing these districts we will, for convenience, describe them by counties, alphabetically arranged.

*Allegheny County.*—The extreme southern part of Allegheny County contains the principal areas of low-sulfur coal existing in the Pittsburgh coal bed in this county. This land lies in the center of the great mining development upon this bed of coal. The sulfur content increases as the coal rises to the north, northwest, and west. The Freeport upper coal may be a possible source of low-sulfur coal in the northern part of the county, but extensive drilling and development work will be necessary before areas of low-sulfur coal in this or in any underlying coal bed can be accurately defined.

*Armstrong County.*—No large or important area of low-sulfur coal is known in this county. The Freeport lower coal has yielded some high-grade coal of this class, and areas of coal beds E, D, or B may perhaps be developed by drilling in those parts of the county where these beds lie below water level.

*Beaver County.*—This county may have small areas of the upper Freeport and Lower Kittanning low-sulfur coal above water level, but

these areas are not likely to prove important. In the southern part of the county adjoining Washington and Allegheny Counties, low-sulfur coal may perhaps be developed in any of the coal beds under water level, *i.e.*, in E, D, C′, or B.

*Butler County.*—In the southern and southwestern part of the county, low-sulfur coal may be developed in the Upper or Lower Freeport coal beds, where these coals lie below water level. No other area large enough to be of importance is known in which there is any reasonable probability of obtaining low-sulfur coal.

*Cambria County.*—This is one of the most important low-sulfur coal districts in Pennsylvania. It contains many more or less detached areas in which beds E, D, and C′ have yielded coal of this character, but the principal source of supply is from bed *B* (Lower Kittanning coal) which is generally known as the "Miller" seam and furnishes high-grade "smokeless fuel" of low volatile, low sulfur, and low ash contents. The low-sulfur coal areas of bed B are found principally in the southern half of the county, that is, south of Ebensburg, while most of the low-sulfur areas of beds C′, D, and E are found in the northwestern part of the county.

In addition to the true low-sulfur coal areas, this county contains large areas in which coal (especially the coal of beds B and D) with from 1.25 to 2 per cent. sulfur exists in beds of fair workable thickness and which, by washing, can be made available for byproduct coking.

*Clearfield County.*—The fine grade of low-sulfur coal yielded by the celebrated "Moshannon coal bed" (Lower Freeport coal, bed D) in the Houtzdale-Morrisdale basin, has been practically exhausted. Small workable areas of this coal bed and of the Upper Kittanning coal, bed C′, may yield some coal low in sulfur but the supply from these areas does not promise to be large and no other part of the county except some areas of small extent in the DuBois district are known to contain any coal low in sulfur.

*Fayette County.*—The low-sulfur coal areas of this county, so far as is known, are confined to coal of the Pittsburgh coal bed, none of the coal beds of the lower productive coal measures being of this grade over any considerable area. The great low-sulfur coal field of the Pittsburgh bed, extending from near Connellsville on the northeast nearly to the Cheat River on the southwest, still contains large quantities of coal of this grade and an extension of this district from Smithfield westwardly toward the Monongahela river has been thought to indicate a possible extension of the same conditions into a part of Greene County.

*Greene County.*—The Pittsburgh coal bed is the only probable source of supply of low-sulfur coal in this county. Sufficient drilling and development work has not yet been done, nor have sufficient results of drilling been published to enable us to form an opinion as to the extent of low-

sulfur coal likely to exist in this county. In many cases the recorded data are in conflict, but sufficient is known to justify the conclusion that the southeastern part of the county is not likely to furnish a large percentage of coal of low-sulfur content. This same conclusion is indicated by the data bearing upon the northwestern part of the county. It has been claimed that there is a great belt of low-sulfur coal running through the central part of this county from northeast to southwest. This claim is based mainly upon the condition of the Pittsburgh bed to the northeast of this belt in Fayette, Westmoreland, and Washington Counties, but without any evidence to the southwest of this belt to lend strength to this hypothesis, so that confirmation of this theory must depend on the results of drilling. The possibilities of this territory are very great, but the facts in hand do not justify positive statements except such as may apply to individual parts of the district that may have been proved by drilling.

*Indiana County.*—The Upper Freeport (E), the Lower Freeport (D), and the Upper Kittanning (C′) coal beds may each be expected to contribute a number of areas containing low-sulfur coal in such quantity as to be of importance. Between and surrounding and interwoven with these low-sulfur areas are areas of much greater extent containing coal that can readily be so improved by washing as to make high-grade metallurgical coke. These areas occur most frequently in the central and eastern parts of the county. In those parts of the county adjacent to its southern and western boundaries, the coals are usually higher in sulfur and are not so readily improved by washing. The Lower Kittanning coal (B), or Miller seam, is rarely a low-sulfur coal in this county.

*Jefferson County.*—The Punxsutawney district has become celebrated for the low-sulfur, low-ash coal mined from the Lower Freeport (D) coal bed. This coal field has been extensively worked for more than 30 yr., and the low-sulfur area has been well defined by mining operations extending far beyond the limits of this area. No considerable addition to the available coal of this grade is expected to be developed or proved in this coal bed in the future. Neither the Upper Freeport (E), Upper Kittanning (C′), nor Lower Kittanning (B) coal bed is likely to prove an important source of supply for low-sulfur coal in this county.

*Lawrence County.*—Small areas of low-sulfur coal in the Upper Kittanning (C′) and Gallitzin coal beds have been thought to exist in this county.

*Somerset County.*—The well-known "Jenner" smokeless coal (bed C′) and the Meyersdale basin of the Pittsburgh coal bed have been important sources of supply for coal of this type, and low-sulfur coal is also provided from bed B in the Windber district in the northeastern part of the county. Outside these districts, low-sulfur coal is rarely found in quantity, but the areas in which low-sulfur coal is known to exist are of

sufficient size to continue to be important sources of supply for many years.

*Washington County.*—The central and southeastern parts of Washington County contain important areas of the Pittsburgh coal bed of low-sulfur content. In addition thereto we find large tracts in which the coal is low enough in sulfur to be easily improved by washing for use in making metallurgical coke of high grade. Much territory that, a few years ago, was classed as low-sulfur coal has been found to require washing to reduce the sulfur to within the permissible limits. In the western and northwestern parts of the county, the coal is generally thought to be too high in sulfur to be readily reduced by washing to fit it for coke making. We believe, however, that much of the coal from these relatively high-sulfur districts can be made available for coking by washing as described in the first part of this paper.

*Westmoreland County.*—In the southern and southwestern parts of Westmoreland County important areas of the Pittsburgh beds of low-sulfur content have been developed and extensively worked. The sulfur seems to increase to the north and northeast, but this increase is not uniform and does not affect all of the coal. Much of the coal produced by the Pittsburgh bed at these several localities requires washing to reduce the sulfur; and as the sulfur usually occurs in a form capable of separation by washing, the county will continue to be a source of supply of coal for making high-grade coke. No large areas are known in this county in which coal from any of the coal beds of the lower productive coal measures is low enough in sulfur to be classed as low-sulfur coal.

## DISCUSSION

RICHARD R. HICE, Beaver, Pa. (written discussion*).—The matter of selective mining is probably of more importance in Pennsylvania than is washing, and perhaps washing would not be necessary at some plants where now practised if proper care were taken in the mining. I have in mind one plant where the sulfur content of the entire bed is more than 2 per cent., yet if the bottom 12 in. and the top 12 or 15 in. are separated, the main portion of the bed, some 6 ft., will not contain more than 1.25 per cent. sulfur; and yet in this mine the entire bed is worked together, the resultant coke, as a matter of course, not being fit for use in iron smelting.

Not sufficient notice is taken of the "Double-thick" Upper Freeport coal in Allegheny and Butler counties. This is a very valuable deposit. Other areas where there is a top member of the Upper Freeport coal are also known, which add to the value of this bed as a coking coal. This double Freeport coal is especially suited for byproduct coking.

* Received Aug. 26, 1919.

I cannot be as optimistic as regards the "low-sulfur" Pittsburgh coal in Greene County as the authors. If the term low-sulfur is to be confined to coal with not exceeding 1.25 per cent. sulfur, there is probably little of Greene County that will come under the classification. The mines now working in the county, the records of drill holes in this county and in adjoining areas of Washington County and West Virginia, do not indicate that there will be much coal in Greene County that can be classed as low-sulfur. Western and southern Washington County, from the relatively few data available, must be largely classed as carrying at least more than 1.25 per cent. sulfur. In some cases the sulfur content can be reduced by washing, but in others the reduction in sulfur will not justify the expense.

The authors have brought into a concise statement the substance of a mass of literature on Pennsylvania coals, which certainly is of value both to the coal industry and to the consumer who wants a fuel suited to his needs.

# Removal of Sulfur from Illuminating Gas

BY W. W. ODELL,* AND W. A. DUNKLEY,† URBANA, ILL.

(Chicago Meeting, September, 1919)

THE sulfur content of coal is perhaps more important in the manufacture of illuminating gas than in any other coal-using industry. Whether the gas is made by the distillation of coal in retorts or ovens or by steaming incandescent coal or coke in the water-gas process, a portion of the sulfur present in the fuel always passes into the crude gas. Practically all the sulfur that remains in the finished gas is transformed into sulfur dioxide ($SO_2$) when the gas is burned. Sulfur dioxide is a pungent suffocating gas and, if present in the air in any perceptible amount, is deleterious to health and comfort and damaging to house furnishings. So universally is this recognized that laws regulating the amount of sulfur that may be permitted to remain in the gas are very generally in force.

The capacity of a plant to purify gas frequently limits the selection of coal for gas making. This condition may arise from different causes. Some plants were originally designed to handle a certain grade of low-sulfur gas coal, and very little leeway was allowed for growth or for the use of inferior coals. Other plants, when built, had ample capacity to purify gas from relatively higher sulfur coals, but their output has grown so rapidly that their purifying capacity is now limited and only the low-sulfur coals can be handled.

There is a great quantity of coal in the United States that would satisfy the requirements of the gas manufacturer, were it not for the high sulfur content. The coals containing less than 1 per cent. sulfur are becoming scarcer year by year and, in time, the gas industry will probably be compelled to use coals that are now considered out of the question.

## SULFUR COMPOUNDS IN CRUDE GAS

Sulfur exists in unpurified illuminating gas in a variety of compounds; chief of these are hydrogen sulfide ($H_2S$) and carbon disulfide ($CS_2$). In crude coal gas $H_2S$ is usually present to the extent of 0.5 to 2.0 per cent., by volume, equivalent to about 300 to 1200 gr. per 100 cu. ft. of gas. In carbureted water gas, the amount of sulfur compounds depends not only on the sulfur content of the coal or coke used, but also upon the

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sulfur in the carbureting oil. The total amount of $H_2S$ in the unpurified carbureted water gas is usually much less than the amount present in unpurified coal gas, being usually between 50 and 200 gr. per 100 cu. ft. of gas. The amount of $CS_2$ in gas made by either process seldom exceeds 30 gr. per 100 cu. ft. In making gas from American coals, the amount of $H_2S$ produced is usually the limiting factor, from the sulfur standpoint, the amount of other sulfur compounds present being of less importance.

## Preliminary Purification

The term "purification" as applied to gas means the removal of certain gaseous constituents present in the gas as first produced, which are detrimental to its use if allowed to remain in it. In coal-gas manufacture, these impurities include carbon dioxide ($CO_2$), ammonia ($NH_3$), cyanogen $(CN)_2$, nitrogen (N), and all sulfur compounds, including $H_2S$ and $CS_2$. Nitrogen is not usually considered an impurity, for although it does not burn and only dilutes the gas, there is no commercial method known for its removal. The same impurities are present in carbureted water gas, but the amounts of ammonia and cyanogen compounds are so small as to be negligible.

The purification process begins as soon as the gas leaves the chamber in which it was produced. Certain amounts of the impurities are removed by water vapor as it condenses from the gas; also, some of the impurities react with each other, forming compounds soluble in water, such as the combination of the ammonia with part of the $CO_2$ and $H_2S$ during cooling and washing of coal gas. The following equations express some of the most important of these reactions:

$$NH_4OH + H_2S = NH_4HS + H_2O \qquad (1)$$
$$2NH_4OH + H_2S = (NH_4)_2S + 2H_2O \qquad (2)$$
$$2NH_4OH + CO_2 = (NH_4)_2CO_3 + H_2O \qquad (3)$$
$$(NH_4)_2CO_3 + CO_2 + H_2O = 2(NH_4)HCO_3 \qquad (4)$$

The amount of ammonia present, however, is much too small to remove these constituents completely. These reactions should be utilized as far as possible, for by so doing the hydrogen sulfide purifiers can be relieved of much work that would otherwise be thrown upon them. Frequently as much as 20 to 40 per cent. of the $H_2S$ can be removed from the gas before it reaches the sulfide purifiers. Some of the processes evolved for utilizing the reactions between the impurities, especially between $NH_3$ and $H_2S$, will be discussed later.

## Purification with Lime

The greater part of the $H_2S$ originally present remains in the gas after the washing and condensing processes are completed. To completely remove this constituent, the gas is passed through a series of receptacles, usually called purifying boxes, in which it is brought into intimate con-

tact with some material that combines with the $H_2S$. The material formerly used was slaked lime, $Ca(OH)_2$. This material had some chemical advantages over the hydrated oxide of iron now almost universally employed, since it removed not only $H_2S$ but also the $CO_2$ present in the gas and, when properly handled, the $CS_2$ as well. There is some difference of opinion as to the exact chemical reactions occurring in a lime purifier. When there is an excess of hydrated lime present, both $CO_2$ and $H_2S$ are removed. The following equations probably represent the reactions occurring:

$$Ca(OH)_2 + CO_2 = CaCO_3 + H_2O$$
$$Ca(OH)_2 + H_2S = Ca(OH)(HS) + H_2O$$

After a portion of the lime is sulfided, the hydrosulfide, $Ca(OH)(SH)$, can then react with $CS_2$, removing this constituent according to the following representative equation

$$2Ca(OH)(HS) + CS_2 = CaCS_3Ca(OH)_2 + H_2S$$

It is seen that $H_2S$ is liberated by this reaction. Further, as soon as the first box is completely fouled by $CO_2$ and $H_2S$, the $CO_2$ in the gas begins to liberate $H_2S$ according to the equation

$$Ca(OH)(HS) + CO_2 = CaCO_3 + H_2S$$

Hence a fresh box of slaked lime has to be placed after the partly fouled boxes to take up the $H_2S$ liberated.

Although lime removes impurities very completely, it has many disadvantages as compared with hydrated iron oxide, which has largely displaced it. It has a purifying capacity per bushel of only 6000 to 10,000 cu. ft. of gas (when the gas contains the usual amount of $CO_2$ and $H_2S$) or about one-fourth to one-sixth the purifying capacity of iron oxide and the material must be handled much oftener to purify the same amounts of gas. It is therefore much more costly per unit volume of gas purified. In order to insure complete $H_2S$ removal, either a large number of boxes is necessary or the process has to be very carefully watched. The spent lime is a most offensive product, creating a nuisance in the neighborhood of the gas works, and its disposal is difficult, especially since it has little or no commercial value.

### Purification with Iron Oxide

Hydrated oxide of iron, or iron oxide, as a gas-purifying material first received the serious attention of the gas industry in America about 1885, and so obvious were its advantages that within a few years it had practically displaced lime for purification. Iron oxide does not remove $CO_2$ from the gas, but with proper working conditions the removal of $H_2S$ is very complete. There is some evidence that foul oxide removes a slight amount of $CS_2$. The reactions by which $H_2S$ is removed may be represented by the following equations:

$$2Fe_2O_3.H_2O + 6H_2S = 2Fe_2S_3 + 8H_2O$$
$$2Fe_2O_3.H_2O + 6H_2S = 4FeS + 2S + 8H_2O$$

The sulfiding reactions are carried out in a series of purifying boxes in which the oxide is spread in layers on wooden grids, so arranged that the gas has to pass through the oxide in traveling through the box. The boxes are made in various forms and are constructed from steel plate, cast iron, or concrete. The oxide is usually mixed with some inert material, such as wood shavings, ground corn cobs, sawdust, etc., to make it more porous. The direction of flow through the oxide may be up or down, or the gas may enter between two layers and pass up through one and down through the other. Practice varies greatly in different plants.

When fouled, the iron sulfides formed are brought into contact with oxygen (air), and the iron oxide is revivified. The process of revivifying is represented by the equations:

$$2Fe_2S_3 + 3O_2 + 2H_2O = 2Fe_2O_3.H_2O + 6S$$
$$4FeS + 3O_2 + 2H_2O = 2Fe_2O_3.H_2O + 4S$$

This revivification is accomplished in one of the following ways: By spreading the foul oxide in thin layers exposed to the action of air. By shutting off the gas from one or more boxes at a time and passing a current of air through them. By admitting a small amount of air (not over 2 per cent.) with the gas to be purified, thus combining purification and revivification.

With the first method, the oxide must be removed from the purifier several times before it is unfit for further use (spent). With the other methods it is usually necessary to handle the oxide less often The oxide is usually dispensed with for purifying purposes when it contains from 50 to 60 per cent. of free sulfur.

Spent oxide, especially that from coal-gas purification, has a commercial value; it forms an important source of cyanides, and sulfur is sometimes extracted from it. The following is a typical analysis of a spent oxide from coal-gas purification: Sulfur, 61.3 per cent.; tarry matter, 1.8 per cent.; ammonium sulfocyanide $(NH_4)SCN$, 1.7 per cent.; Prussian blue, $Fe_4(Fe(CN)_6)_3$, 5.2 per cent.; moisture, 1.9 per cent.; iron oxide, iron sulfide, shavings, etc., 28.1 per cent.

## Purification Practice

To remove $H_2S$ completely from gas, it is necessary to have more than one purifier in the system. The number of boxes in a series may vary from two to six or even more. A box of oxide may be removing a very large percentage of the $H_2S$ present in the gas long after it is incapable of taking out the last traces. It is necessary, therefore, to regulate the flow of gas through the system so that comparatively fresh oxide is available to remove the last trace of $H_2S$. The connections of the purifiers are so arranged that the position of any box in the series can be changed; making it first, last, or intermediate. Frequently, when the

position of one of the purifiers is established, the others have to follow in a predetermined sequence due to the arrangement of connections.

The amount of purification being accomplished by each purifier of the series can be determined by making quantitative $H_2S$ tests of the gas entering and leaving each purifier. The difference in $H_2S$ content of the gas before and after a box indicates the amount of the $H_2S$ that it is removing. The following is the result of such a series of tests made by William Bennet.[1] The purifiers were in the sequence given in the following table and 2 per cent. of air was admitted to the gas for revivification:

| Order | Box No | $H_2S$ at Inlet | $H_2S$ Removed | Per Cent of Total $H_2S$ Removed in Each Box |
|---|---|---|---|---|
| 1 | 4 | 554 5 | 527 1 | 95.06 |
| 2 | 5 | 27 4 | 20 6 | 3.72 |
| 3 | 1 | 6 8 | 6 8 | 1 22 |
| 4 | 2 | 0 0 | 0 0 | 0 00 |

It is seen that the first box in the series absorbed more than 95 per cent. of the $H_2S$ in the gas, but the absorption was not complete until after the third box. To show that this was not due to an excessively foul conditon of the second box the order of the boxes was changed, No. 5 being placed first, and after 24 hr. another series of tests was made, with the following results:

| Order | Box No. | $H_2S$ at Inlet | $H_2S$ Removed | Per Cent of Total $H_2S$ Removed in Each Box |
|---|---|---|---|---|
| 1 | 5 | 575 7 | 517.2 | 89 84 |
| 2 | 6 | 58 5 | 49 0 | 8 51 |
| 3 | 1 | 9 5 | 9 5 | 1.65 |
| 4 | 2 | 0 0 | 0 0 | 0 00 |

In this instance No. 5 box, which when second was incapable of completely removing the small amount of $H_2S$ in the gas coming to it from the first box, is now able to absorb nearly 90 per cent. of the total $H_2S$.

The difficulty of removing traces of $H_2S$ from the gas greatly affects purification practice in different plants. Where only two boxes are installed, it is the usual custom to have the fouler box in the first place and to rely on the cleaner second box to remove the traces of $H_2S$. On the other hand, where there are several boxes in series, the practice is sometimes reversed, based on the theory that the foul box will revivify more readily if most of the $H_2S$ has previously been removed in a clean box.

Not only does the order of the boxes differ in various plants, in respect to the foulness of the contained oxide, but also the method of changing the order. For example, if a given plant has four boxes in series, numbered

[1] *Gas World* (Feb., 1916), **64**, 202.

in the following order: 1 — 2 — 3 — 4. When box No. 1 becomes foul, permitting enough $H_2S$ to go forward so that a trace appears between 3 and 4, it is time to change the order. Some operators would make the new order: 2 — 3 — 4 — 1; while others would make it 4 — 1 — 2 — 3. The latter arrangement appears to be safer, since box No. 1 is relieved of the heavy duty of removing $H_2S$ and so has an opportunity to revivify, while the comparatively fresh boxes Nos. 2 and 3 are interposed between it and the outlet of the system to remove traces of $H_2S$, which will almost inevitably pass box No. 1. By the time box No. 1 reaches the last place, it will be revivified to such an extent that it will remove any remaining $H_2S$.

## Conditions Favoring Sulfur Removal

Among the factors affecting sulfur removal are freedom of the gas from tar and oil vapors, temperature, time of contact of the oxide with the gas, hydrogen sulfide content of the gas, surface of oxide exposed, and the chemical and physical condition of the oxide. It is important to have the gas free from tar and oil vapors before it enters the purifiers. It is probably true that as much oxide is ruined by failure to observe this precaution as is consumed in efficient purification. This is due sometimes to inability to regulate the temperature of the gas and efficiently scrub it with the means available. This factor merits attention in many gas plants, especially the smaller ones.

The temperature maintained at the purifiers is an important consideration. For efficient action, a temperature at the boxes of 80° to 100° F. (27° to 38° C.) is considered most favorable. If the temperature falls much below 60° F. (17° C.) the action becomes sluggish. Where this temperature cannot be maintained by the sensible heat of the gas, artificial means are employed, such as using steam coils in the boxes or injecting steam into the gas. By many, it is considered important that the gas should be saturated with water vapor during the purification process; and where closed steam coils are used for heating, some advocate the introduction of steam into the gas.

The time of contact of the gas and oxide has a marked effect on the efficiency of the purifying process. If it is possible to pass the gas through the oxide at a slow rate, a considerably higher percentage of $H_2S$ can be removed than when a higher rate is necessary. It is very advantageous, therefore, to pass the gas through the oxide at a uniform rate. Various formulas have been worked out showing the relation of rate of gas flow to area and depth of purifying material in the purifiers.

The chemical and physical properties of oxide have an important bearing on their efficiency in removing $H_2S$. The relations of chemical composition to commercial usefulness have not been so completely worked out that it is possible to predict with certainty from a chemical

analysis how the material will behave as a purifier. The commercial supply is obtained from a variety of sources and various kinds of oxide are preferred by different operators.

## Sources of Purifying Oxide

The oxides employed for purifying gas fall in the following classes: *Natural oxides,* these include bog iron ore, and other forms of hydrated iron oxide. *Iron rust,* prepared by rusting clean iron borings, etc., in contact with wood shavings or other carriers. *Precipitated oxides,* made by precipitating iron oxide from a solution of copperas (ferrous sulfate) or of other iron salts, by the use of lime.

It has long been recognized that the purifying capacity and activity of oxides vary greatly and many efforts have been made to explain the differences experienced. While iron oxide is the material that reacts with $H_2S$ in every case, experience has shown that the iron content is not necessarily an index to the value of an oxide for this purpose. The physical condition of the oxide and perhaps the state of hydration of the oxide have an important bearing on its action. Fulweiler and Kunberger[2] expressed the view that oxide present in the form of a colloidal precipitate was the most efficient for $H_2S$ removal and that this condition was usually indicated by the ability of an oxide to carry a high percentage of combined water. They also stated that such an oxide usually has a smooth velvety texture and that its quality can be judged better by appearance under the microscope, or even by the feel of the material, than by chemical test, though they recommend an actual fouling test such as they describe.

While further study of the properties of oxide may give added information as to means of purifying efficiency, which in some plants is deplorably low, its use will always have certain disadvantages. Oxide, being a solid material, is bulky and costly to handle, even though mechanical appliances for handling it are in use in some of the larger plants; considerable labor is required in most cases. The material, when foul, is very disagreeable to handle, and it is sometimes difficult to procure laborers for this work.

In the foregoing discussion of iron oxide for sulfur removal, the use of the oxide in a dry way only has been considered. Iron oxide may be used in suspension in water to accomplish the same results; in fact, this method was contemplated in the early days of oxide purification. The wet method has some advantages as well as disadvantages. O. B. Evans[3] tells of some experiments made by the Atlantic Refining Co., in which gases from oil stills, containing as much as 8000 gr. of sulfur per 100 cu. ft., were purified by passing through two towers (in series) filled with

[2] *Proc.* Am. Gas Inst. (1913) **8,** 476. [3] *Gas Rec.* (Apr. 9, 1919) **15.**

glazed helical brick. Iron oxide suspended in water flowed down through the towers into 1500-gal. tanks. The iron sulfide, which was precipitated by the $H_2S$, was kept in suspension and was oxidized again by air agitation. From the experiments, the conclusions were drawn that the method would be applicable to usual gas practice in some cases. The chief conclusions were:

1. The method results in a serious loss of candlepower when a high-candle gas is purified at ordinary temperatures, but ordinary illuminating gas would probably lose less than 2 per cent. of its candlepower at 90° F. (33° C.).

2. A given apparatus, under the same conditions, will remove a constant percentage of the $H_2S$ present in the gas, whether the gas contains 8000 or 700 gr. per 100 cu. ft.

3. With two towers in use, so operated as to remove 95 per cent. of the sulfur in the gas, the first tower will do 93 per cent. of the work.

4. When completely fouled, the oxide may contain as much as 89 per cent. free sulfur.

Mr. Evans states that the main difficulties to be experienced are the loss of candlepower (where candlepower is required) and the revivification of the material in suspension, which does not proceed so readily as when in a dry state. The chief expense, he states, is the cost of pumping the material. About 3 per cent. of oxide in suspension was about as great a concentration as could be readily handled.

## Other Purifying Methods

Weldon mud ($MnO_2 \cdot MnO \cdot H_2O$), a hydrated peroxide of manganese, has been used for removing the last traces of $H_2S$ in the gas. It has not found extended use, however, on account of the high first cost and the fact that it readily absorbs $CO_2$ and becomes inert to $H_2S$.

A number of liquid purification methods have been worked out which, while more or less successful, are not to any great extent supplanting the less complicated iron oxide method. The method of Th. P. Petit is rather unique. In this process a 25-per cent. solution of potassium carbonate ($K_2CO_3$) is used to absorb the $H_2S$ from the gas. This solution is used in rotary washers, which are placed after the ammonia washers. The $K_2CO_3$ solution absorbs $CO_2$ and $H_2S$ forming bicarbonate and potassium hydrogen sulfide, according to the reactions:

$$K_2CO_3 + CO_2 + H_2O = 2KHCO_3$$
$$K_2CO_3 + H_2S = KHS + KHCO_3$$

The spent solution is revivified by blowing air through the solution at 105° to 120° F. (41° to 49° C.). The temperature has considerable to do with this revivification method. Since $CO_2$ helps to drive off $H_2S$ a small amount of stack gas is used in revivification.

For the same reason, an excess of $CO_2$ in the gas to be purified hinders the absorption of $H_2S$. While care must be used with this process to see that the carbonate solution is always in the right form to absorb $H_2S$ and so prevent an excessive production of $CO_2$ in the raw gas, one advantage claimed for the process is that no sulfites, sulfocyanides, or hyposulfites are formed (when the ammonia has been previously removed), which are quite troublesome in some other processes. The operation is said to be simple. It may be stated that the air used in revivification acts merely as a carrier for $H_2S$, HCN, and $CO_2$

A number of processes have been devised utilizing the reaction of ammonia with $H_2S$. Since there is never enough $NH_3$ in coal gas to remove all the sulfur present, it is evident that the ammonia solutions used have to be revivified and reused. This may be done by heating the weak ammonia liquor nearly to the boiling point when most of the $CO_2$ and $H_2S$ are driven off with but little loss of ammonia. Of course strong liquor cannot be so heated without great loss. The cooling and reheating of great volumes of liquor together with losses and uncertainty as to the efficiency of purification has, in part, accounted for the method not being generally adopted. Many other more complex ammonia reactions take place in utilizing this process, which add to the difficulty of regulation. Since the gas constituents vary in many gas plants from hour to hour, greater attention must be given to purification when employing such a process. $CO_2$ is as objectionable in these processes as with the Petit method.

Several investigators have endeavored to remove $H_2S$ from gas by the use of sulfur dioxide ($SO_2$). In effect one might expect the reaction between these gases to be as follows: $2H_2S + SO_2 = 2H_2O + 3S$. In practice, however, this combination does not take place in so simple a manner.

W. Feld experimented with the use of $SO_2$ in gas purification and ultimately developed a process by which both $NH_3$ and $H_2S$ can be removed from the gas. In this process, he made use of the peculiar characteristic of ammonium thiosulfate of combining with $SO_2$ to form ammonium polythionate. The reaction is rapid and complete.

$$2(NH_4)_2S_2O_3 + 3SO_2 = (NH_4)_2S_4O_6 + (NH_4)_2S_3O_6$$

Ammonium polythionate is an effective medium for removing both $H_2S$ and $NH_3$ from the gas.

While there are many interactions in the application of this process, the final products are shown by the following equations:

$$(NH_4)_2S_4O_6 + 3H_2S = (NH_4)_2S_2O_3 + 5S + 3H_2O$$
$$(NH_4)_2S_4O_6 + 2NH_3 + H_2O = (NH_4)_2SO_4 + (NH_4)_2S_2O_3 + S$$
$$(NH_4)_2S_4O_6 + 2NH_3 + H_2S = 2(NH_4)_2S_2O_3 + S$$

In each of these reactions ammonium thiosulfate is formed and converted to polythionate by $SO_2$.

By alternate treatment with coal gas and $SO_2$, the process is continuous with an accompanying increase in concentration of ammonium salts and sulfur. The excess ammonia is recovered, as is also the sulfur, some of which is burned for the generation of $SO_2$. The ammonia is recovered as the sulfate. This process is explained in some detail by Wagner.[4]

## Removal of Carbon Bisulfide

Various methods have been devised for the removal of $CS_2$ from the gas, although in American practice it is seldom necessary to remove this constituent. The most notable methods employ heat (reheat the gas), with or without the use of catalyzers. One such process was worked out by Dr. Charles Carpenter in conjunction with Mr. E. V. Evans. In this process the gas is preheated to a temperature of 400° C. and then passed through externally heated tubes containing fireclay balls impregnated with a nickel salt which, in the presence of hydrogen, promotes catalytic action at a temperature of 450° C. The reaction is expressed chemically as follows: $CS_2 + 2H_2 = 2H_2S + C$. In this manner the total sulfur in the purified gas (which in most cases is almost entirely $CS_2$ or organic sulfur) is reduced 80 per cent. A requirement is that $H_2S$ is removed before the $CS_2$ is transformed; therefore this method cannot be directly applied to the purification of foul gas.

The credit for working out a method (employing a preheating and a high-temperature reaction) for the decomposition of $CS_2$ that could be successfully utilized on a commercial scale is due to Hall and Papst of the Portland Gas & Coke Co., Portland, Ore. The apparatus installed at that plant consisted of two reheaters, each being a column filled with checker firebrick. These columns are alternately heated and are operated intermittently. The temperature required for this process is 700° to 900° C. The $CS_2$ is decomposed according to the well-known reaction $CS_2 + 2H_2O = CO_2 + 2H_2S$. The hydrogen sulfide produced in each of the processes just mentioned is later removed by iron oxide.

Liming of coal previous to charging the retorts is another method that has been used to decrease the sulfur in the unpurified gas. From time to time various reports have been made showing an increase in the yield of gas and ammonia, and a decrease in the per cent. of sulfur in the gas from the use of this process. For this purpose, the lime used is equal to 1½ to 2 per cent. of the weight of the coal. It is admitted with the coal before it is crushed, and steam is blown on the coal just before it is charged. This results in the coal being coated with lime, which is fixed to its surface. Results obtained at Cheltenham, England, seem to show

[4] F. H. Wagner: "Coal Gas Residuals." N. Y., 1918. McGraw-Hill.

that under the prevailing conditions more of the sulfur remained in the coke and other byproducts, with correspondingly less sulfur in the gas.

While the chemistry of this process is not thoroughly understood, it is possible that the lime, by catalytic action or otherwise, permits the formation of less $CS_2$, or rather converts the $CS_2$ more completely into $H_2S$. It was shown that the purified gas had a uniform content of 21.5 gr. of organic sulfur per 100 cu. ft. when lime was used. Liming of coal is not a common practice in this country.

A process known as the "Athion" is of importance since it is, or has been, in use in Europe. The gas after being deprived of tar, $H_2S$ and $NH_3$, is washed with a solution of potassium carbonate ($K_2CO_3$) which absorbs $CO_2$: $K_2CO_3 + CO_2 + H_2O = 2KHCO_3$. The $CO_2$ is driven off by heating the liquid and the carbonate is regenerated.

After the $CO_2$ is all removed from the gas the Athion material (which is a cellulose compound obtained by treating cellulose sulfate with soda lye) absorbs the $CS_2$. The Athion material is placed on grids in purifying boxes in the ordinary way. On combining with $CS_2$, xanthogenate of cellulose (viscose), which is utilized in the manufacture of non-inflammable celluloid, is formed. This method of purification reduced the sulfur compounds in finished gas in one instance from 30.7 to 7.8 gr. per 100 cu. ft. It is claimed that 10 tons of this special material will absorb 1¼ tons of $CS_2$, or purify 75,000,000 ft. of gas.

## Conclusion

The installation of purifying equipment is a heavy expense. Purifying apparatus of the present type requires considerable space, not only for the boxes themselves but for the storage and revivification of oxide. The cost and scarcity of labor in some communities make a less cumbersome process desirable. In view of these considerations, it seems as though the gas industry will be limited to the use of low-sulfur coals until a more convenient and cheaper method of purification has been worked out. Those plants that have excess purifying capacity are frequently able to realize good returns on their added investment by their ability to use coals somewhat higher in sulfur than customary when they are obtainable at a lower price. For example, several water-gas plants in Illinois and neighboring states are using local coals as generator fuel in place of coke and are realizing a saving, even though in some cases the sulfur to be removed has increased 50 to 100 per cent. In plants in which the purifying equipment is overloaded, the solution of the problem must rest on two considerations: first, the more efficient utilization of the present process, more attention being paid to the nature of the material used and the best ways of using it; and, second, the development of processes that will permit more economical gas purification and will perhaps make the sulfur in the coal a desirable byproduct rather than a detriment.

# DISCUSSION

Homer T. Darlington, Natrona, Pa.—John Hudson recently perfected a material, which is called ferrox. This material is very similar to brick in appearance and in physical properties. It is so strong that a person can stand on a piece of the ordinary size that is used in purification, without crushing it. Ferrox is sulfuric hydrate, which is the basis of the so-called oxide purifiers, and the binding material. After the material has been made, it is crushed to ¾ in. With material of this size and the same heat, fifty times the amount of gas will pass through the same part of the bed. In this way, a larger amount of gas can be purified than by the old method, where the so-called ordinary oxides used in gas purification were used. I would think that this material would have a very large application in the purification of producer gas. It is used to a small extent in the purification of illuminating gas.

The Chairman (H. H. Stoek, Urbana, Ill.).—Is it a patented process material?

H. T. Darlington.—I think it is, I do not know.

W. H. Fulweiler, Philadelphia, Pa.—There seems to be a tendency to make the presence of a little sulfur dioxide seem rather terrible. This is rejecting the conclusions of the Bureau of Mines after its investigations at Baltimore a number of years ago. Then the Bureau said the $SO_2$ would drive a man out of a place long before it would seriously injure his health. As a matter of fact, illuminating gas as served to the consumer today contains a smaller percentage of impurities than white granulated sugar.

I want to call attention to one thing in the line of purification. Generally the fresh box is kept in front of the sulfide removing boxes. In other words, the technical method of handling purification is to first insure that all of the $CO_2$ is removed before the $H_2S$ is taken out. Two sets of boxes are used, the so-called carboning boxes and the sulfating boxes.

In the purifying, the equations given here are those given in the textbooks ever since the industry started, but I believe, from some work that has been done, that the reactions that take place between the iron oxide, the $H_2S$ reactions, are not quite as simple as they appear. It is well known that a certain amount of ferrous sulfate is always formed and, practically always in the ordinary methods of purification, some $FeS_2$. A trace of $SO_2$ is nearly always formed in carelessly handled boxes and, therefore, I think it must be patent that the reactions are more or less complicated.

The same may be said for the revivification of this material. Some experiments rather indicate that the ferrous oxide is not first formed

from the revivification of ferrous sulfide. Very probably, apparently, ferrous sulfide is formed and the lower oxide that is formed rapidly is gradually oxidized up to the higher oxide.

The impression is given by the paper that by admitting 2 per cent. of air you intentionally revivify the oxide. That is not so. The admission of small amounts of air increases, to a considerable extent, the duty of the oxide; it will not completely revivify it. The revivification in the box is attracting a great deal of attention at the present time but that revivification, while simple in theory, is quite complicated because the heat of the reaction of the revivification appears to be rather high and the ordinary purifying materials are rather good non-conductors of heat.

There is a great tendency for the oxide to collect in lumps into which the oxygen penetrates and causes a reaction. There is not a sufficient current of air to carry off the heat generated by the reaction When the temperature rises, secondary reactions take place, which also increase the temperature, resulting in the dehydration of the ferric hydroxide, consequently spoiling it, and in the excessive formation of $FeS_2$, which, of course, does not further revivify, and therefore passes out of usefulness.

When the oxide has been revivified, it mechanically increases in volume due to the formation of the sulfur. After a batch of oxide has been revivified in boxes until it has swollen to twice the mechanical size, the swelling of the material increases the resistance of the passage of gas so much that material must be taken out.

The best and most efficient oxides we have seen were oxides that occurred as a byproduct in the manufacture of alumina in which the iron was precipitated from a carbonated solution. The physical characteristics of the iron have an important bearing on the volumes of purifying material. An iron that comes out from a strong neutral solution is never as good as an iron that comes out of an alkaline solution, like soda-carbonate solution.

In the discussion of the methods for removing hydrated carbon bisulfide, attention might have been called to a method in practical use in which, after the removal of hydrogen sulfide, the gas is passed over a very active iron oxide at a temperature of about 200° C. At that temperature, the reaction is sufficiently rapid for the greater portion of the carbon bisulfide to be converted into hydrogen sulfide. We found that the temperatures used by Carpenter on the nickel process were detrimental, apparently, to carbonated water gas. They worked all right with coal gas, but when applied to carbonated water gas with the excessive amounts of unsaturated hydrocarbons, there was a loss of candlepower and an additional poisoning of gases. If iron hydroxide properly prepared can be made to react at such a low temperature, that is 200° C., it would not have a bad effect on the candlepower. We could drop it down 90 per cent.

It is an interesting fact that no matter what number of grains of carbon bisulfide you start with, you never can get it down to 9 or 10 by these processes. It is very difficult to get a quantity of gas and hold it and keep the sulfur contents the same. We have tried a large number of materials as containers and it is difficult to get anything that will not change the sulfur content.

The summary states that the question of capital charge is one of the very important things that face the gas companies today. Money is very hard to get for public service corporations at times. They have a large amount of money invested in the old system, so the new system would have to show a large saving to warrant their scrapping what they have. High-sulfur coals do not seem to offer a great reduction of price. The most promising method is the development of the more active ferric hydroxides. The average iron hydroxide might take up 10 per cent., by weight, of hydrogen sulfide before it would be found. It might be easy to get some of the boggers to take up 35 and 40 per cent., which would mean increasing the capacity of a given box by probably three times.

It is also very easy to prepare, in the laboratory, oxides that will take up as much as the theoretical, that is 63.45; and by the admission of a little air, that can be easily run up to as much as 80 per cent. without removal. That is the direction in which we are most likely to make direct progress in view of the high capital cost involved in any other method.

How does this so-called ferrox act on revivification? It would seem that any porous material of that nature would only act on the surface after the first revivification, because the interior would almost immediately become clogged with the free sulfur set free in the revivification. A Mr. Berkhiser developed a process for converting the sulfur directly into sulfuric acid in which he used oxide supported with an inorganic binder in what appears to be identically the same method Mr. Darlington mentioned.

H. T. Darlington.—I am not sure the binder has been definitely fixed upon. Cement is the cheapest binder mixed with appropriate materials for giving strength to hold the ferric oxide together. Revivification, I understand, has been repeated often enough so that the total amount of sulfur absorbed has gone as high as 100 per cent. of the ferric oxide or ferric hydrate content. Before the material had been tested out, the fear was that the sulfur deposited by the reaction—first by the absorption of hydrogen sulfide and then by revivification—would swell and crack the mass. However, such was not the case. The material remained hard until it was thrown away. The material absorbs hydrogen sulfide to the center with great rapidity. A mass 1 in. in diameter will absorb hydrogen sulfide to the center within 2 or 3 minutes.

# Forms in which Sulfur Occurs in Coal*

BY A. R POWELL† AND S. W PARR,‡ M. S, URBANA, ILL.

(Chicago Meeting, September, 1919)

FOUR general methods have been used in the study of the decomposition of coal. The first has been directed toward the processes of coal formation, the second has been by means of microscopic studies, the third from the data of destructive distillation, and the fourth has made use of various solvents.

Applying these methods to the sulfur of coal, it has been found that the organic sulfur of plant and animal life enters into the formation of coal and remains in organic combination in the final product. The inorganic sulfur is almost entirely combined as iron pyrites and marcasite. The infiltration of hydrogen-sulfide waters is doubtless a large factor in these combinations. Sulfates are found in very small quantities except where free oxygen has entered the coal strata.

The mechanical mixing of detritus with the early coal material is another source of sulfur. In this case the sulfur is distributed in microscopic particles throughout the mass. In destructive distillation, a part of the sulfur comes off as hydrogen sulfide and various thiophen compounds, but this furnishes little information as to the nature of the sulfur compounds in the coal itself.

The use of solvents has given valuable data concerning the sulfur compounds in coal. Fremy[1] and later E. Guignet[2] noted the action of alkaline solutions on coal after treatment with nitric acid and Anderson and Roberts[3] showed the presence of large quantities of organic sulfur in the extract so obtained. In a study of the phenol extract of coal, Parr and Hadley[4] found organic sulfur in the soluble organic material. These investigations proved the presence of organic sulfur in coal in addition to the inorganic forms, such as iron pyrites and sulfates. T. M. Drown[5] attempted the analysis of the pyritic and sulfate sulfur as distinct from the organic forms by means of sodium hypobromite; no definite proof of

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* Synopsis of an investigation published as *Bull.* 111 (1919) of University of Illinois Engineering Experiment Station. By permission of the Director.

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[1] *Compt. Rend.* (1861) **52**, 114. [2] *Compt. Rend.* (1879) **88**, 590.

[3] *Jnl.* Soc. Chem. Ind. (1898) **17**, 1013.

[4] Univ. of Ill. Eng. Exp. Sta. *Bull.* 76 (1914).

[5] *Chem. News.* (1881) **43**, 89.

the accuracy of this method was presented. Ferd. Fischer[6] showed that this method gave variable results and was inaccurate. Helm[7] tried the selective action on the organic-sulfur compounds by means of organic solvents but obtained no definite results. No trustworthy method seems, therefore, to have been devised for determining the different forms of sulfur in coal.

The effect of different percentages of organic and inorganic sulfur in the coal on the amount of residual sulfur when the coal is coked was studied by M'Callum.[8] No quantitative relationship was obtained, but where the organic sulfur was high the volatile sulfur was also high.

The forms of sulfur in coke were early investigated by Bradbury.[9] Some sulfide sulfur was found to be present and also a little sulfate sulfur, but the larger part was unaffected by most reagents. This latter sulfur was thought to exist in some organic form.

The purpose of this investigation was to devise a fairly accurate and reliable method for the quantitative determination of the various forms of sulfur in coal and to utilize this method in a study of the changes occurring in the coal sulfur during the processes of oxidation and coking.

*Methods of Analysis.*—The first attempt at a method for the analysis of the different forms of sulfur in coal was made by the use of organic solvents. Previous work, by E. E. Charlton of this laboratory, had indicated the possibility of securing a selective extraction of the organic sulfur by treating the finely powdered coal for some time with phenol; this method has been studied in some detail during the present investigation. The criterion by which the accuracy of this method was judged is based largely on an assumption, the only available procedure at the time, as follows:

The total sulfur and iron in the coal were determined by the sodium-peroxide fusion method and then another sample of the coal was treated with a 3 per cent. hydrochloric-acid solution for a period of 40 hr. at a temperature of 60°. The latter treatment extracts only the sulfate sulfur leaving the pyrites and organic sulfur intact. Both the sulfur and iron in this extract were determined. The iron not extracted by the dilute hydrochloric acid was assumed to be pyritic iron and the pyritic sulfur was obtained from this by a simple stoichiometric calculation. This pyritic sulfur, together with the sulfate sulfur obtained in the dilute hydrochloric-acid extract, was considered to be the total inorganic sulfur of the coal, and the organic sulfur content was figured by subtracting the inorganic from the total sulfur. The main source of error in this procedure lies in the fact that most coals contain appreciable amounts

[6] *Zeit Angew. Chem.* (1899), 764.

[7] *Archiv. Phar.* (1882) **38.**

[8] *Chem. Eng* (1910) **11,** 27.

[9] *Chem. News* (1878) **38,** 147.

of iron silicate, which on calculating the equivalent sulfur to $FeS_2$ gives a high factor for that constituent.

In attempting to secure a direct factor for organic sulfur, the phenol soluble material was first studied. The apparatus used for the phenol extraction is shown in Fig. 1. The heating medium was an electric oven constructed as shown. The coils of nichrome wire were ¼ in. (6.35 mm.) in diameter and were wound around the iron cylinder in a spiral fashion; 50 ft. (15 m.) of No. 20 wire were used. The extraction was carried on in 50-c.c. Erlenmeyer flasks fitted with long glass tubes that extended

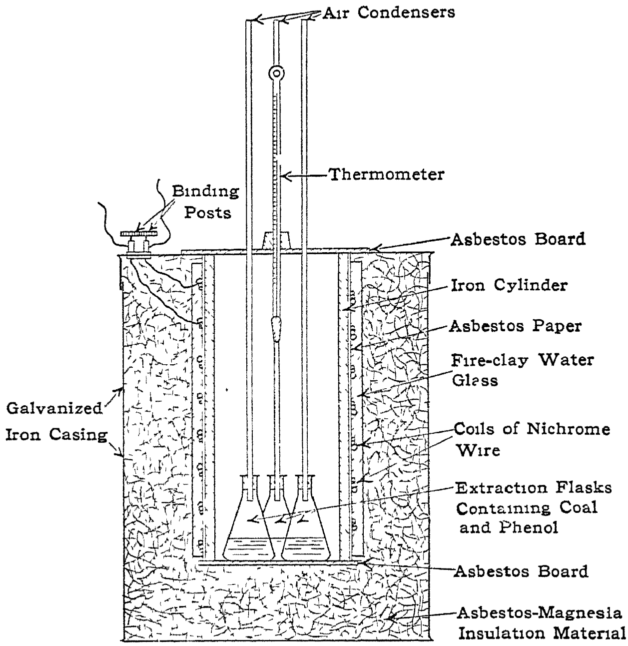


FIG. 1.—CROSS-SECTION OF ELECTRIC FURNACE USED IN PHENOL EXTRACTION OF COAL.

outside the oven to act as condensers for the phenol vapors. The method of procedure was to place ½ gm. of the coal in the flask, pour over it 25 c.c. of phenol, and maintain the oven at a temperature of 140°. The extraction was continued for 20 hr., when the contents of each flask was filtered through a Gooch crucible while still hot. Every particle of residue was rinsed out of the flask by means of alcohol and ether. The residue in the Gooch crucible was dried, mixed with sodium peroxide, and ignited in a nickel crucible. The solution obtained from this fusion was analyzed for sulfur by the regular sulfate method. By subtracting the sulfur found in this residue from the total sulfur, the phenol soluble sulfur was indicated. The results are given in Table 1.

TABLE 1.—*Sulfur as Provisionally Distributed. Based on Total Iron and Phenol Soluble Organic Material*

| Coal No | Total Sulfur | Sulfur Calculated as $FeS_2$ from Total Iron | Sulfur as Sulfate | Total Organic Sulfur, by Difference | Organic Sulfur Soluble in Phenol | Difference Between Total Organic and Phenol-soluble Sulfur |
|---|---|---|---|---|---|---|
| 1 | 2 68 | 1 39 | 0 04 | 1 25 | 0 42 | 0 83 |
| 2 | 2 18 | 1 58 | 0 17 | 0 43 | 0 15 | 0 28 |
| 3 | 0 64 | 0 17 | 0 00 | 0.47 | 0 20 | 0 27 |
| 4 | 2 14 | 0 71 | 0 05 | 1 38 | 0 34 | 1 04 |
| 5 | 1.20 | 0 40 | 0 25 | 0 55 | 0.16 | 0 39 |
| 6 | 5 00 | 2 03 | 1 31 | 1.66 | 0 77 | 0 89 |
| 7 | 3 31 | 2 06 | 0 01 | 1 24 | 0 50 | 0 74 |
| 8 | 1.02 | 0 80 | 0 01 | 0 21 | 0 10 | 0 11 |
| 9 | 1.40 | 0 75 | 0 00 | 0 65 | 0 21 | 0 44 |
| 10 | 0.94 | 0 31 | 0 02 | 0 61 | 0 12 | 0 49 |

In compiling the data for this table, it should be borne in mind that the factor for $FeS_2$, while based on an assumption with reference to the form in which all of the iron is combined, shows, in the nature of the case, a maximum value for the resulting $FeS_2$. As a result of this assumption, the value in the column for the total organic sulfur, by difference, is the minimum possible for that constituent. Nevertheless, the values remaining after subtracting the organic sulfur of the phenol-soluble material show very considerable percentages of organic sulfur remaining in the insoluble or humic residues. It is the factor for this constituent that we desire to obtain accurately. Not nearly all of the organic sulfur is extracted by the phenol.

The coals tested were for the most part from Illinois and ranged from freshly mined material to samples that had been kept in the laboratory for 2 years. In no case was there, even approximately, complete extraction of the organic sulfur by phenol, unless some heretofore unidentified inorganic forms should account for the difference. That the sulfur extracted by the phenol was organic in nature could not be doubted, since that material left no ash on ignition.

The next step was to search for a selective solvent for the iron pyrites. Nitric acid oxidizes and takes into solution iron pyrites very readily, so that a series of experiments was started to determine whether the pyrites could be dissolved without disturbing the organic sulfur. Samples of coal were treated with dilute hydrochloric acid and then subjected to extraction with concentrated nitric acid. The method used in treating the coals with the concentrated acid was very simple; 25 c.c. of concentrated nitric acid was poured over 1 gm. of coal and the mixture was allowed to stand 24 hr. before filtration and analysis. The results of

these experiments showed that less sulfur was extracted by the nitric acid than that supposed to be pyritic sulfur in the coal. Furthermore not quite all of the iron was extracted from the coal, as should be the case if all the iron were present as pyrites.

Some doubt was entertained as to whether all of the pyrites was being taken into solution, so experiments were performed to determine the effect of stronger solvents. Samples of coal were extracted with a mixture of one part concentrated hydrochloric acid to three parts of concentrated nitric acid, using gentle heat to hasten the process. The sulfur in these extracts was much higher than that obtained by the action of cold concentrated nitric acid, but the iron content was about the same. These facts clearly indicated that the nitro-hydrochloric-acid mixture not only extracted the iron pyrites but also quite a considerable amount of some other form of sulfur. Furthermore, the ratio of sulfur to iron in the extract was higher than that corresponding to $FeS_2$. Even the cold concentrated nitric acid extract gave a sulfur-iron ratio that was high.

It appeared very probable that the iron in the acid extracts represented the pyritic iron, since silicate iron would go into solution only on prolonged heating. This, therefore, might be used as a means for calculating the pyritic sulfur and it would without doubt be more accurate than the older method of using the total iron. A comparison of three methods of calculation is given in Table 2.

TABLE 2.—*Comparison of Pyritic Sulfur Determinations of Coal by Different Methods*

| Coal No | 5 | 6 | 7 | 9 |
|---|---|---|---|---|
| I. Sulfur as calculated from the total iron minus the HCl soluble iron. . . . . . . . . . . . . . . . . . . . . . . . . . . . . . . . . . . . | 0 40 | 2 03 | 2 06 | 0 75 |
| II. Sulfur as calculated from the iron soluble in HCl + $HNO_3$ minus the iron found in the dilute HCl solution | 0 30 | 1 94 | 1.72 | 0 25 |
| III. Sulfur by analysis in the HCl + $HNO_3$ solution minus the sulfur found in the dilute HCl solution . | 0 34 | 2.64 | 2 09 | 0 39 |

This table shows that both the pyritic sulfur obtained by the older method of calculation from the total iron and the pyritic sulfur obtained directly by analysis from the hydrochloric-nitric-acid extraction are higher than that obtained by calculation from the iron soluble in hydrochloric-nitric acid. The high results based on the total iron are due to the presence of quantities of silicate iron, and the high results of the direct hydrochloric-nitric-acid extraction of pyritic sulfur are due to the oxidation of a part of the organic sulfur.

From the behavior of the reagents thus far used it now seems possible to find one in which the oxidation of the organic sulfur would not occur, so that a solution could be obtained which would yield directly the pyritic sulfur (plus the sulfate sulfur). There would then be a distinct line of demarkation between the inorganic and the organic forms. In order to judge as to whether there was a selective action on the iron pyrites, the sulfur-iron ratio in the solution was used as affording the best indication of a selective solvent.

Various strengths of cold nitric acid were tried. The concentrated acid, as stated before, gave high results, but when one part of concentrated acid was mixed with three parts of water and this was allowed to act on a sample of coal, the sulfur-iron ratio of the extract was almost theoretical. From one to four days was required for complete solution but further extraction after this period was found to be without result. One gram of finely powdered coal and 80 cc. of the dilute acid were allowed to stand at room temperature and at the expiration of four days the extract was analyzed for both sulfur and iron. The results of this method, applied to various coals, are given in Table 3. A graphical representation of the results obtained is given in Fig. 2. In all cases the ratio between the sulfur and iron in dilute nitric acid extract indicates a selective action on the iron pyrites.

TABLE 3.—*Extraction of Coals with Dilute Nitric Acid; One Part of Concentrated HCl to Three Parts of Water*

| Coal No. | Time of Extraction | Iron in Acid Extract | Sulfur in Acid Extract | Pyritic Sulfur Calculated from Iron in Acid Extract as Pyritic Iron |
|---|---|---|---|---|
| 7 | 1 hour | 1.12 | 1 23 | 1.28 |
| 7 | 18 hours | 1 16 | 1 29 | 1.33 |
| 7 | 72 hours | 1.13 | 1 30 | 1 29 |
| 7 | 10 days | 1 20 | 1.36 | 1 37 |
| 6 | 4 days | 1.80 | 2.06 | 2 06 |
| 6 | 4 days | 1.80 | 2.06 | 2 06 |
| 5 | 4 days | 0 27 | 0 31 | 0.31 |
| 5 | 4 days | 0 27 | 0 31 | 0.31 |
| 9 | 4 days | 0.15 | 0 16 | 0.17 |
| 10 | 4 days | 0 08 | 0 10 | 0 09 |
| 10 | 4 days | 0 08 | 0 10 | 0 09 |

Methods had now been devised for the estimation of the sulfate sulfur, of the pyritic sulfur, and that part of the organic sulfur soluble in phenol. However, on adding these three forms together and comparing the sum with the total sulfur, it was found that only about one-half of the total sulfur had been accounted for; therefore, it became necessary to devise a method for making a direct analysis of this residue.

The procedure was as follows: One gram samples of the various coals were extracted with the dilute nitric-acid solution at room temperature for 24 hr., thus removing the pyritic and sulfate sulfur. Guignet and later Friswell[10] had both observed the effect of alkalies on coal previously treated with nitric acid. The residue, after the extraction with nitric acid, was therefore treated with 25 c.c. of concentrated aqua ammonia. This ammonia mixture was then diluted and filtered, the filtrate having a dark reddish brown color. Upon acidifying this filtrate with hydro-

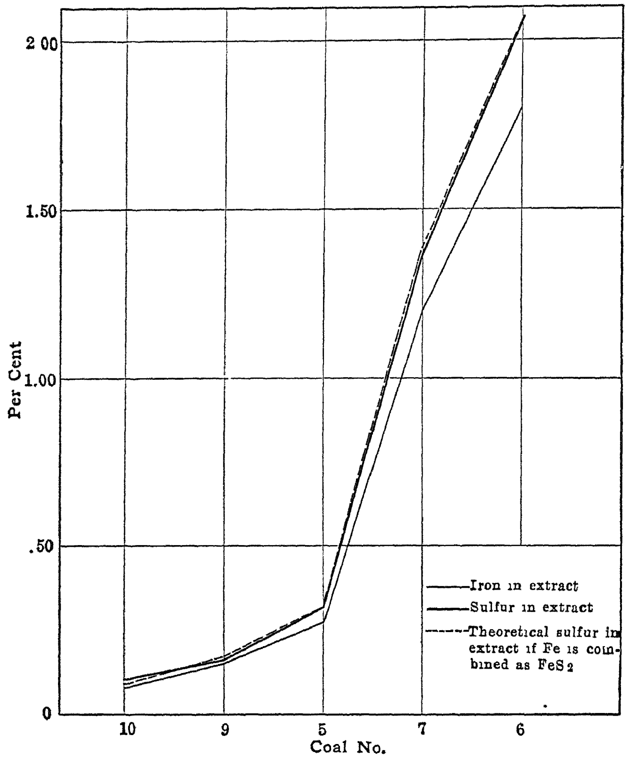


FIG. 2.—GRAPHIC RESULTS OF EXTRACTION OF COALS WITH DILUTE NITRIC ACID.

chloric acid, a brown flocculent precipitate formed. The filtrate from this precipitate was very light yellow in color and contained no sulfur. This showed that the sulfur in the residue from the nitric-acid treatment must be present in either the ammonia insoluble substance, of which there was very little compared to the original residue, or else in the flocculent brown precipitate. An analysis of the small undissolved portion of the residue, after the treatment with ammonia, gave but a trace of sulfur. Practically all of the remaining sulfur of the coal was therefore contained in the flocculent brown precipitate.

[10] *Loc. cit.*

The brown precipitate gave an ash content of only 0.35 per cent. as against an initial ash content of 9.45 per cent. in the original coal. The analysis of this residue was as follows: Silica, 0.10 per cent.; $Fe_2O_3$, 0.05 per cent.; other oxides, 0.20 per cent.; total, 0.35 per cent. The low ash content indicates quite definitely that the sulfur contained in the brown precipitate is organic in nature. In order to distinguish this form of sulfur from the other forms contained in coal, the name of humus organic sulfur has been applied to it.

It now became necessary, in order to test finally the methods just given, to determine the amount of the various forms of sulfur present in certain coals, add these together, and see how closely the total checked with the total sulfur of the coal as determined by a sodium-peroxide fusion. Extremely close checks were not looked for, since the combined errors of five determinations entered into each comparison Table 4 gives the average results obtained on five different coals. The results show that a fairly accurate method had been found for the determination of the various forms of sulfur.

TABLE 4.—*Analysis of Different Forms of Sulfur in Coals and Comparison of the Sum with the Total Sulfur*

| Coal No | 4 | 5 | 6 | 7 | 8 |
|---|---|---|---|---|---|
| Resinic sulfur | 0.34 | 0.16 | 0.77 | 0 50 | 0.10 |
| Sulfate sulfur | 0 05 | 0 25 | 1.31 | 0 31 | 0 01 |
| Pyritic sulfur | 0 85 | 0 31 | 2.06 | 1 36 | 0 29 |
| Humus sulfur | 0 87 | 0 51 | 0.70 | 0 95 | 0.45 |
| Total | 2 11 | 1 23 | 4 84 | 3 12 | 0 85 |
| Total sulfur by $Na_2O_2$ fusion | 2 14 | 1 20 | 5.00 | 3 31 | 1 02 |
| Difference between sum of the four sulfur forms and the total sulfur | 0 03 | 0 03 | 0.16 | 0 19 | 0.17 |

For more convenient reference the method of procedure for each form is outlined below.

(1) The sulfate sulfur is determined by extraction of the coal with dilute hydrochloric acid. Five grams of the finely powdered coal are treated with 300 c.c. of a 3-per cent. solution of hydrochloric acid for 40 hr. at 60° C. This is then filtered and the filtrate analyzed for sulfur by the usual barium chloride precipitation process.

(2) The pyrite sulfur is determined by extraction of the coal with dilute nitric acid after a preliminary extraction with dilute hydrochloric acid as under (1) to remove the sulfate form. As a matter of fact, the residue from the sulfate determination may be used for the pyritic sulfur determination, but the quantity is rather large for this purpose and it is more convenient to use 1-gm. samples. The coal residue so obtained, or

1 gm. of the original coal is treated with about 80 c.c. of dilute nitric acid. (1 part nitric acid, sp. gr. 1.42, to 3 parts water, approximate specific gravity of the mixture is 1.12). This is allowed to stand at room temperature for 24 hr. and is then filtered. The filtrate is evaporated to dryness, to get rid of the nitric acid, then taken up in a little hydrochloric acid and the sulfur determined by barium chloride precipitation.

The organic sulfur may be taken as the difference between the total sulfur and the sum of the two inorganic forms as found under (1) and (2). However, if direct determinations are desired of the two organic forms the procedure is indicated under (3) and (4) as follows:

(3) The resinic sulfur is determined by extraction of the coal with phenol. The method of procedure for this phenol extraction has been given in some detail earlier in this paper.

(4) The humus sulfur is determined by the difference between the sum of the other three forms and the total sulfur in the coal. Or, a direct determination may be made by taking the residue from the nitric-acid extraction for pyritic sulfur and adding 25 c.c. of ammonium hydroxide (sp. gr. 0.90). This is poured over the residue and allowed to stand for several hours. It is then diluted, passed through a large filter, and the filtrate evaporated to dryness. By fusing the dry residue with sodium peroxide, its sulfur content may be determined in the usual manner

## DISCUSSION

W. H. Fulweiler, Philadelphia, Pa.—Was any attempt made to determine the nature of this ammonium soluble extract?

A. R. Powell.—We could not get a test for any typical organic sulfur compound. This substance, dissolved in the ammonium hydroxide, seems to be closely related to humus substances found in soils; at least it has the same physical characteristics. For that reason we have called this "humus organic sulfur," a very complex, organic combination that would probably require a very long investigation to determine its constitution.

W. H. Fulweiler.—Has an ultimate analysis been made?

A. R. Powell.—No; there are three or four articles about this substance, which is soluble in ammonia; the ultimate analysis is referred to in *Bulletin* 111, which may be obtained from the Engineering Experiment Station at the University of Illinois, Urbana. Detailed data on the ultimate analysis of this substance are given in an article by Anderson and Roberts in the *Journal* of the Society of Chemical Industry, Vol. 17, p. 1013 (1898).

# Mechanical Separation of Sulfur Minerals from Coal

BY J. R CAMPBELL,* M SC., SCOTTDALE, PA

(Chicago Meeting, September, 1919)

A DOZEN years or so ago, the general superintendent of our company, now the president, Mr. W. H. Clingerman, asked me to study the coal-washing problem. This work brought me into contact with the best washery men of the country. I investigated many kinds of coal-cleaning schemes, made numerous tests, and collected voluminous data, yet I confess frankly that I have had little practical washery experience and that little has been at the Middlefork washery of the United States Fuel Co. at Benton, Franklin Co., Ill., which was put in operation last fall. While this washery installation may not be the "last word," I am quite sure it embodies some long steps in advance of the older designs and constructions.

From the washery man's standpoint, the removal of pyrites or marcasite, a disulfide of iron, is the one vital problem, and the one productive of the most beneficial results, so far as elimination of sulfur is concerned. Sulfates and so-called organic sulfur occupy quite a secondary position and are generally lost sight of in coal washing.

The elimination of sulfur in coal in the past has usually been accomplished by jigging in water, but lately concentrating tables have been successfully employed. The flotation process has been applied to fine coal, but is hardly considered practicable. Coal washing is rapidly coming into its own, due to the depletion of low-sulfur coals for coking purposes, and we predict that soon the same careful attention will be given to the design and construction of coal washeries that is now being given to byproduct plants and other allied industries. Heretofore, coal washing has been considered only an incidental and necessary evil instead of the big problem it is—one worthy of the most careful study by men well trained along technical lines and capable of delving into the fundamental principles.

## Preparation of Coal

It is usually necessary to crush the mine-run coal before sending it to the jigs, the degree of crushing depending on the occurrence of the sulfur and other impurities in the coal and the type of jig over which it is to be sent. The feldspar jig requires finer and more uniform crushing

* Chief Chemist, H. C. Frick Coke Co.

from a mechanical operation standpoint than the type of jig that makes its own bed. Coal crushed to pass a ½-in. (12.7-mm.) or ¾-in. (19-mm.) square-hole screen may be considered the maximum size for the successful operation of the feldspar jig. Much larger particles can be handled over other jigs, though ¾-in. coal will give more efficient and uniform results. The main object of crushing is to liberate the impurities in the coal.

If the coal is to be sent over concentrating tables instead of jigs, it must be crushed much finer. Our experience is that these tables will not handle material much above ⅜ in. square, perhaps ¼ in. being the most desirable size. Tests have been made on ⅛-in. to 0-in. material with gratifying results.

The matter of sizing the coal for the jigs, or concentrating tables, is a mooted question. Some of our best washery men claim that no better reduction in sulfur can be obtained by washing in sizes while others say that there is a decided advantage in jigging sized coal. We have an open mind on the subject. It appears that the nature of the coal and the impurities in it have some bearing on whether the coal ought to be sized or not. If it is found desirable to wash in sizes, perhaps two sizes would be sufficient. We have found that it is practicable to jig coal ¼ in. and even ⅛ in. to 0 in., if it is properly wetted before entering the jig. If a concentrating table is used in combination with a jigging plant, it is necessary to size in two sizes, the ¼-in. or ⅜-in. coal going to the tables and the oversize to jigs.

The type of crushers used is important, as it is desirable not to produce any more fines than necessary when jigging, because it is more difficult to handle the fines in jigs without loss of good coal, especially in washing unsized coal. It is the common belief among washery men that the roll-type crusher produces less fines. It is also highly desirable to produce the proper size in one operation; *i.e.*, without secondary crushers for handling the oversize. This is feasible if there are facilities for bypassing the undersize coal in the mine-run coal and handling only the oversize through the crushers. If double rolls are used, provision should be made for bypassing the undersize coal produced by the top rolls so that the bottom rolls handle only oversize coal. In this way uniform and properly sized coal can be produced for efficient washing.

## Type of Jigs and Tables

There is a saying among washery men that any jig will do good work if given the proper adjustment and attention, whether it is a vertical plunger or a horizontal plunger jig and whether it is a single or a double (or more) compartment jig. It is my belief that the two-compartment jig is about the limit for the best results. In fact, I am not convinced

that a single-compartment jig, properly designed, is not the thing, especially if feldspar is not used on the second compartment, or both compartments, of a double-compartment jig.

It is also highly desirable to have a jig that will produce two products, washed coal and refuse, from a given feed with minimum loss of good coal in the tails and the minimum amount of sink in the washed product. Any jig that is not able to do this is inefficient, for at this stage of the coal-washing game, a secondary or middling product is not to be tolerated when the washed coal has to be used for coking purposes.

I have had the opportunity to make a critical study of wet concentrating tables for coal-washing purposes, and most of them have done excellent work, yet I see no reason why they should supplant the use of the jig altogether, even on handling fine coal. The table may have certain advantages over the jig, such as less power and water consumption, but the tonnage handled is decidedly against it. A feldspar jig will produce just about as low sulfur, though not as low ash, as wet concentrating tables in handling the same class of feed coal; *i.e.*, $\frac{1}{4}$-in. or even $\frac{1}{8}$-in. coal. If some genius in dry concentration will devise an A-1 table for fine coal of this size, it will prove a boon to coal washing, for it would simplify the handling and drying of the fines.

I do not wish to be understood as opposing the use of wet concentrating tables for handling fine coal, for I appreciate the fact that they are rapidly and deservedly pushing their way into the coal-washing field. Unless feldspar jigs are used for fines alone, tabling the fines below $\frac{1}{4}$ in or $\frac{1}{8}$ in. is the only practical way of handling them, but the use of jigs for the over-size coal should always be considered, when practicable from an operating standpoint.

## Drying Washed Coal

There are two principal ways of drying the washed coal from the dewatering elevators: the older method of using drainage bins, or pits, and the more modern and rapid one of using mechanical dryers, or centrifuges. The first method is slow and rather inefficient, unless large drainage capacity is provided for storing the washed coal to allow ample drainage. The drainage pit is exemplified by the installation at the Cambria Steel Co.'s washer at Johnstown, Pa. I have seen this installation several times and, if my recollection is correct, the moisture is reduced to about 12 per cent. with reasonable time for drainage.

Of the mechanical dryers, there are at least two on the market and in successful use at several washery plants in the country. Mechanical dryers are successfully used at the Middlefork washery. By their use it is possible to reduce the moisture to the minimum and load the dried coal direct into railroad cars. At the washery of the Bethlehem Steel

Co., Steelton, Pa., with a low initial moisture in the raw coal, the moisture in the washed coal will average around 5 or 6 per cent.; and at our plant with a high initial moisture in the raw coal, the moisture will average around 10 per cent. in the washed coal with the pulp from the Dorr tanks added to the washed and dried coarse coal. Without this pulp addition, the moisture probably would be 3 per cent. less I will discuss this feature later under a separate head.

The main trouble with mechanical dryers is that they do not handle the fines very successfully on account of the fine coal passing through the screens of the dryers and building up the circulating wash water when operating in a closed circuit.

## Water Clarification and Sludge Recovery

This is one of the most interesting features of a washery plant and one that has been a bugaboo in the past. We took the "bull by the horns" and installed Dorr thickeners for water clarification and sludge recovery on a big scale, though the experiment was first made by the Stag Canon Fuel Co. in a modified way. The use of the Dorr thickeners has been very successful, though we experienced some trouble at first due to inexperience. The following is a typical operation of these thickeners under normal conditions:

| | Influent | Effluent | Underflow |
|---|---|---|---|
| Water, per cent. | 98 | 99.7 | 47 2 |
| Solids, coal, per cent. | 2 0 | 0 3 | 52 8 |
| Specific gravity | 1 0052 | 1 0008 | 1 1580 |
| Total, per cent. | 100 | 97 | 3 |
| Tons per hour | 500 | 485 | 15 |

The above is based on the operation of two 70-ft. Dorr tanks handling the wash water from approximately 1200 tons of washed coal in 8 hr. The overflow water contains but a small percentage of solids and is in fine condition for re-use. The underflow, or sludge, is in good condition for handling in a number of ways and, from an analytic standpoint, takes on the character of the washed coal proper.

The following are possible solutions of the sludge problem: It may be pumped by means of the diaphragm pump direct to the washed-coal elevator on top of the coal in the dewatering buckets and passed through the mechanical dryers with the coarse coal, or it may be pumped direct to the dryer, but this practice builds up the circulating system and is almost certain to cause trouble sooner or later.

The second way is to operate in an open circuit and pump the 50-50 sludge direct on top of the washed and dried coarse coal, which eliminates it from the system altogether, although this practice adds about 3 per cent. moisture to the final washed product.

The third, and perhaps the most logical method, is to put the sludge from the Dorr tanks through a continuous filter of approved type and dehydrate it to 20 per cent. water and under, after which the cake may be added to the dried coarse coal. This method would give a final washed product of minimum water content and would add less than 1 c. per ton to the cost of the washed product

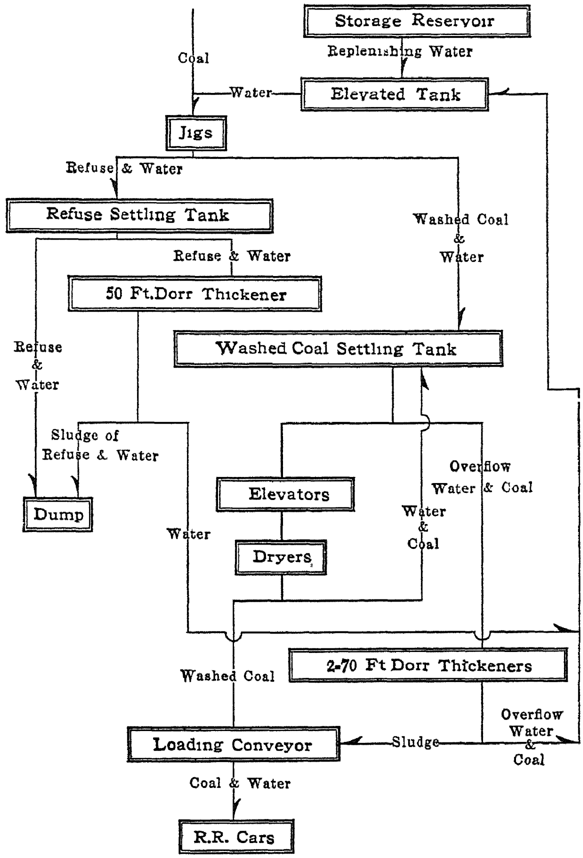


FIG. 1.—FLOW SHEET OF WATER-CLARIFICATION AND SLUDGE-RECOVERY SYSTEM.

A fourth way would be to take the cake from the continuous filter and completely dehydrate it in a direct-heat dryer, adding the powder to the washed and dried coal. This method would be more expensive and seems of doubtful value. Fig. 1 shows a flow sheet of a typical water clarification and sludge-recovery system.

## WATER CONSUMPTION

There seems to be a general belief that 1 ton of water will wash 1 ton of coal; I have seen no jig plants doing so well in practice. The tendency,

perhaps, is to use too much water, since this is the easy way to get the coal over the jigs. For the most efficient removal of sulfur and other impurities, the least possible water should be used. The feed water is an important factor in both jig and table practice. In table work it takes about three times as much feed water as dressing water and the total consumption varies from 1 ton to 2 tons of water per ton of coal handled. It is essential that the coal be wet before going through the washer. As for water consumption on jigs, I am informed that the Delamater single-cell jig uses 1½ tons of water per ton of coal. Our experience on a two-compartment jig is that it takes about 3½ tons of water per ton of coal. A good figure is from 3 to 4 tons of water per ton of coal on a two-compartment jig. The renewal water is that which is carried off by the coal and refuse, loss by evaporation, leakage, etc., and may be safely covered by 50 or 60 gal. per ton of coal. This question of water consumption and renewal is important in arid sections like "little Egypt" where our washery is located and where we have had to construct a mammoth reservoir to provide the water required.

## Mathematical Formulas

Another most interesting thing in the coal-washing game is the diversity of the mathematical problems. I wish to touch briefly on these mathematical considerations, though I appreciate the fact that every technical washery man, or table man, may have his own pet way of doing things. In the past I have always used straight arithmetic and that part of it referred to as alligation alternate, which I learned in the early days after some struggle. For instance, given the analysis of the feed coal, washed coal and refuse, it is comparatively easy to calculate the percentage of refuse, thus,

| | Feed | Washed Coal | Tails |
|---|---|---|---|
| Ash, per cent | 14 48 | 6 98 | 55 42 |
| Sulfur, per cent | 3 53 | 2.41 | 10 40 |
| Total, per cent | 18.01 | 9.39 | 65 82 |

$$18.01 \left| \begin{array}{c|c} 9\ 39 & 8.62 \\ \hline 65\ 82 & 47.81 \end{array} \right\| \begin{array}{ll} 47.81 & \dfrac{47\ 81}{56.43} \times 100 = 84.7 \text{ per cent., washed coal.} \\[2ex] \dfrac{8.62}{56.43} & \dfrac{8\ 62}{56.43} \times 100 = 15\ 3 \text{ per cent., refuse.} \end{array}$$

In a similar manner the percentage of refuse can be calculated from either the ash or the sulfur determination alone when very accurately determined and the results should agree within reasonable limits. In general, the mathematical deductions are more accurate than the actual

weights under ordinary conditions of obtaining the latter. The underlying principle of alligation alternate can readily be formulated as follows·

Let $H$ = chemical analysis, raw coal, $W$ = chemical analysis, washed coal, and $R$ = chemical analysis, refuse; then,

$$H \left| \begin{array}{cc|c|l} W & H-W & R-H & \dfrac{R-H}{R-W} \times 100 = \text{per cent washed coal.} \\ \hline R & R-H & \dfrac{H-W}{R-W} & \dfrac{H-W}{R-W} \times 100 = \text{per cent. refuse.} \end{array} \right.$$

Or, inversely, $\frac{R-W}{H-W}$ equals ratio of elimination and dividing 100 by this figure gives the percentage of refuse, and 100 per cent. − per cent. refuse equals per cent. washed coal.

In a similar manner, $\frac{(H-W)R}{(R-W)H}$ = per cent. of elimination of ash or sulfur. Thus, in the example cited before,

$$\frac{(14.48 - 6.98) \times 55.42}{(55.42 - 6.98) \times 14.48} = 59.3 \text{ per cent. elimination of ash.}$$

$$\frac{(3.53 - 2.41) \times 10.40}{(10.40 - 2.41) \times 3.53} = 41.3 \text{ per cent. elimination of sulfur.}$$

Recently, while investigating the work of the Dorr thickeners, I developed a formula for determining the percentage of solids in the influent, effluent, and underflow from the specific gravity of the various solutions, which seemed the most expeditious way of reaching conclusions rapidly without the tedious and laboratory way of filtering and weighing the solids, though this process should be followed at frequent intervals as a check.

Let $A$ = specific gravity of water, $B$ = specific gravity of coal (say 1.35), and $X$ = specific gravity of solution in question; then,

$$\frac{B(X-A)}{X(B-A)} \times 100 = \text{per cent. solids in solution.}$$

To illustrate, we will use the test on Dorr thickeners, the underflow having a specific gravity of 1.158.

$$\frac{(1.158 - 1.000) \times 1.350}{(1.350 - 1.000) \times 1.158} \times 100 = 52.6 \text{ per cent. solids.}$$

The weighed percentage of solids in the underflow was 52.8 per cent. In a similar way the amount of influent, effluent, and underflow of the Dorr tanks can be determined, given any one of the three quantities, from the specific gravities of the solutions and consequently the water consumption of the plant, if no meter provisions are made. As it is comparatively easy to measure the underflow, the calculation is usually made with this known quantity as to the amount in a given length of time.

## Reduction of Ash and Sulfur

The literature of some of the jig and table manufacturers makes most extravagant claims regarding the reduction of sulfur by washing. Among the eastern coals, probably the most susceptible to sulfur reduction is the Freeport, which may show as high as 50 per cent. reduction in sulfur by washing. The Southern Illinois coal is one of the most difficult washing propositions, from a sulfur reduction standpoint, I have seen. We hope to attain a 40 per cent. reduction in sulfur when the washery is "tuned up." In general, we may say that any coal operator contemplating the installation of a washery for the reduction of sulfur may figure on from 25 to 40 per cent. reduction of sulfur under ordinary conditions. He is particularly fortunate if he has a coal that will permit better results. As to ash reduction, there is not much trouble. From 50 to 60 per cent. reduction in ash is not beyond the range of possibilities. However, the percentage of ash and sulfur reduction is not to be confused with ash and sulfur elimination, which is much higher and calculated according to the formulas set forth in the discussion of "mathematical formulas."

## Sink and Float Tests

I am a strong advocate of the use of the sink and float tests for washery control, as I know of no better way of determining the efficiency of the jigs or table. The specific gravity of the solution used naturally depends on the coal to be tested but usually a solution of 1.35 or 1.40 specific gravity will cover the range. Frequent tests should be made on the refuse and the washed coal in a properly designed sink and float machine, using solutions of the same strength for each. As a general rule, the loss of good coal in the refuse should not be much more than 1 per cent. of the input to the washery, which is considered good washing practice. This means that, if there is a washery loss, or refuse, of 15 per cent., the float in the refuse should range from 7 to 10 per cent. The sink in the washed coal should be held to the minimum, also, to obtain the most efficient reduction of impurities, though there is no loss of product as in the case of float in the refuse.

The sink and float tests on the raw coal may not be a correct guide as to the results to be obtained by washing but it gives some indication of the possibilities. I have always maintained that the actual washing operation ought to produce a lower sulfur than is indicated by the determination of the sulfur in the float on the raw coal, due to the fact that water in a jig is much thinner than the dense heavy solutions used for making sink and float tests, and thus permits a separation of the "float sulfur." With this in view it may be permissible to multiply the sulfur in the float by a factor, say 0.9, to give the sulfur in the washed coal by

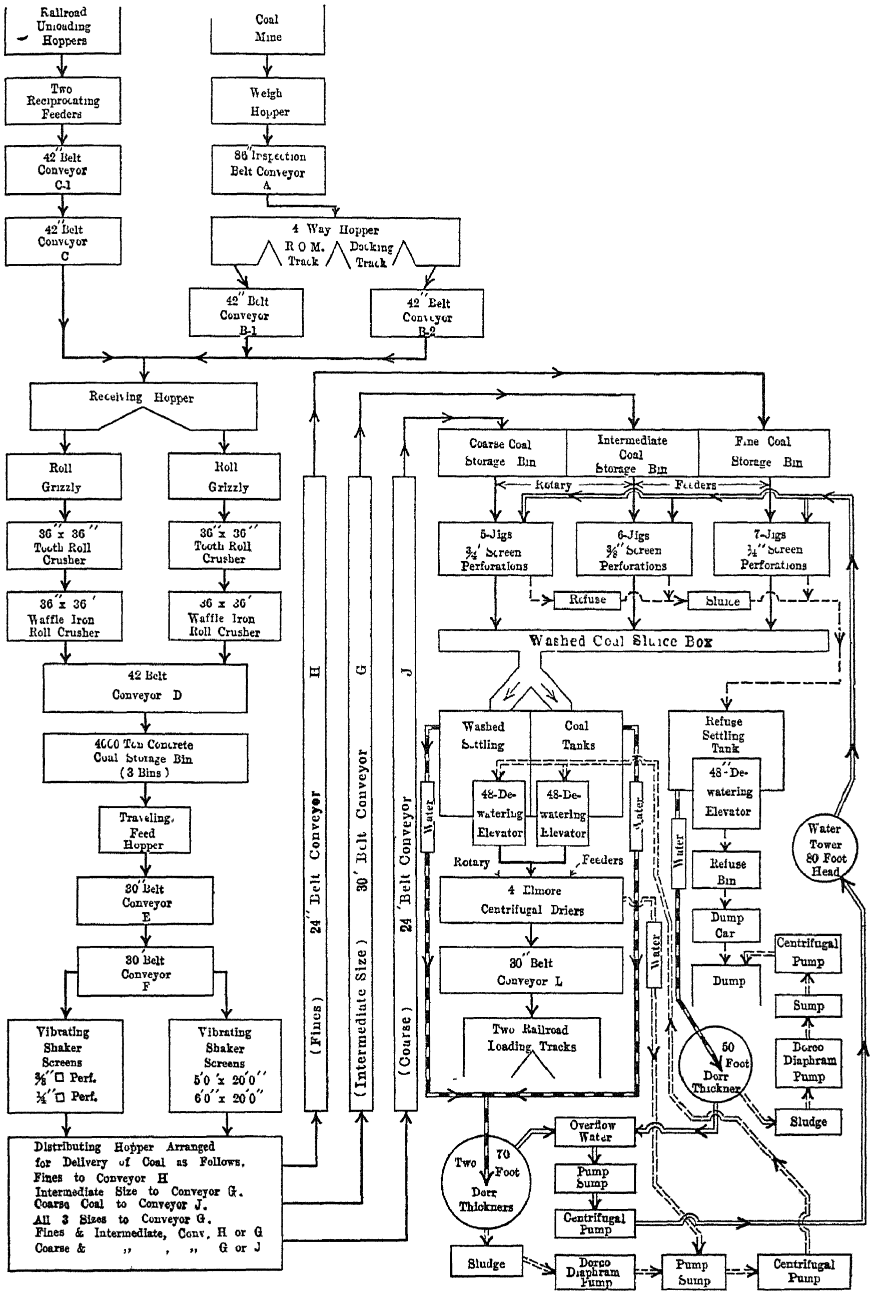


FIG. 2.—FLOW SHEET, MIDDLEFORK WASHERY, U. S. FUEL CO.

jigging or tabling. Thus, if the sulfur in the float from the raw coal is 1.50 per cent. then $1.5 \times 0.9 = 1.35$ per cent., the sulfur in the washed coal.

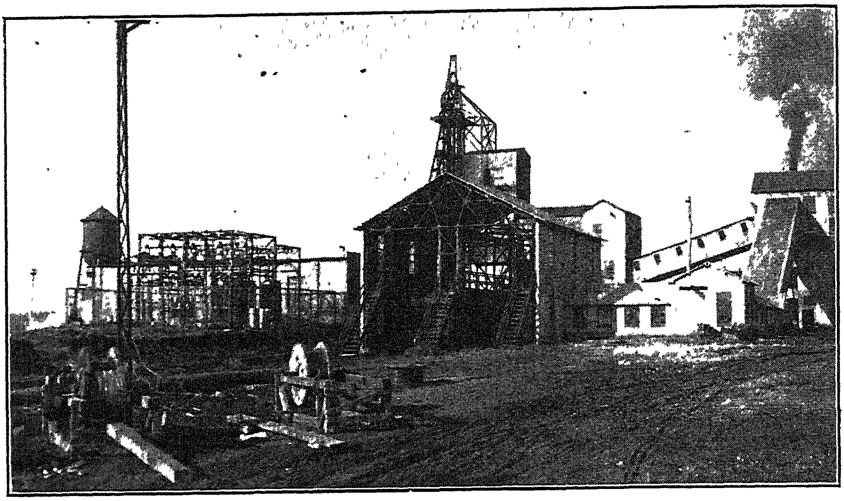

FIG. 3.—UNLOADING STATION FOR FOREIGN COAL.

## CONCLUSION

I am indebted to Mr. G. R. Delamater of the Bethlehem Steel Co. for the flow sheet of the Middlefork Washery, shown in Fig. 2. The flow sheet is not quite correct in some points as some changes have

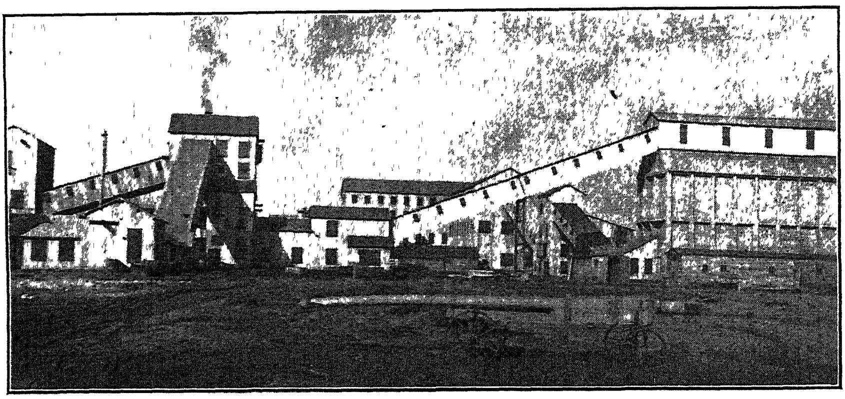

FIG. 4.—CRUSHER HOUSE AND 4000-TON STORAGE BIN.

been made in the crushing plant and the manner of handling the sludge but it shows the general layout. Fig. 3 shows the unloading station for foreign coal, with the tipple and crusher house in the background; and Fig. 4 shows the crusher house with the 4000-ton storage bin for crushed

coal to the right. Finally, as this paper is written several months before its presentation and before all experimentation is completed at the Middlefork washery, some of the statements relating thereto may have to be qualified somewhat after full experience in operation.

## DISCUSSION

G. R. Delamater,* Steelton, Pa. (written discussion†).—Under the heading, "Preparation of Coal," Mr. Campbell discusses the advisability of sizing the coal for the jigs. While I fully realize that conditions alter cases, my method at all times is to do everything possible to avoid sizing. It would be unwise for anyone to say that greater ash and sulfur reduction is impossible by sizing. The thing that can be questioned is, will the greater reduction be sufficient to counterbalance the greater cost? The same thing is true of the proper size of crushing for each individual proposition.

In past years many washeries have been built and equipped for washing two or more sizes, but I know of few instances where these plants were operated more than a year before sizing was abandoned. Perhaps in some of these cases this was due to lack of sufficient development of such scheme of operation; yet I have found that most men engaged in this work, including myself, started into the work with a pretty well-fixed idea that sizing would be beneficial. But trials and tests showed that the mixed size washing is necessary for coals requiring crushing to 1 in. or less.

Coal is of low value, compared with other minerals segregated by washing or concentration, for which reason simplicity must be the watchword: simplicity in the machine used to attain the separations of impurities from the coal, in the accessory plant equipment, and in its arrangement.

Mr. Campbell mentions the quite recent introduction of concentration tables in the washing of coal. It must be admitted that good results in many cases have been obtained, in so far as ash and sulfur reduction are concerned, yet tables also are open to some very serious objections.

The greatest troublemaker to the washery operator is fine coal. It is largely responsible for excessive coal losses. It clogs the water-pipe lines and the jig hutches; and if care is not taken in pump-sump design, the building up of fines and the sudden release to the pumps causes many hours of work and expensive delay. This, therefore, makes important the advisability of careful investigation to determine whether any appreciable ash and sulfur reduction can be attained in the washing of the fines or whether the fines carry any considerable quantity of the impuri-

* Coke Oven Dept., Bethlehem Steel Co. † Received Oct. 28, 1919.

ties. In many cases they do; in many others they do not. Recent developments in screening apparatus have made it possible to screen where screening has been impossible before; this is a feature to which the writer is giving much study at the present time.

Mr. Campbell's reference to single- or multiple-compartment jigs is of much interest as this is another feature in which experience has entirely altered my views. At first, it seemed that the more compartments there were the more rewashing the coal received and the better the results must be. While there are certain conditions under which two or more compartments can, and possibly should, be used, several years of careful experimenting have led me to believe that these conditions seldom, if ever, exist in coal washing. The water flow and pulsation strength are governed by the size and nature of the material treated. Coal containing considerable refuse material of high specific gravity requires greater water flow and pulsation than coal containing only refuse material of a medium specific gravity. In the multiple-compartment jig, the heaviest refuse is admittedly removed in the first compartment, so the second compartment should have a smaller water flow and strength of pulsation. While the outside water feed to the second compartment may be less than to the first, and the pulsation is easily made suitable, it is impossible to avoid the water that flows from the first to the second compartment with the washed coal. This but multiplies with each added compartment until the cross flow seriously affects the proper action on the bed by the pulsation and undesirable disturbances of the bed result. If dewatering were effected between each compartment, this would be avoided. My experience has been that the main claim for multiple-compartment jigs is capacity per jig, but this does not mean per square foot of screen surface, and that three single-cell jigs with a total screen area equal to that of a three-cell jig will treat an equal tonnage at higher efficiency on account of better control of water flow and pulsation strength and because it is necessary to send only one-third the tonnage of coal over any given square foot of the total screen area employed.

The value of a jig over a table is the ability to maintain the mixed area of the bed practically stationary between the two points of removal of washed coal and refuse. Water disturbances that break up this "dead line" ruin the effectiveness of the jig.

Mr. Campbell's statement that any jig that will not produce only two clean products—washed coal and refuse—is inefficient is possibly misleading. I agree that a jig should be capable of such performance but there are good coking coals that are separable into three products: first, the greater percentage suitable for coking; second, a small percentage acceptable only as refuse; and, third, a fair percentage that is neither good enough for coking nor poor enough to throw away as refuse. First washing should segregate only the coking coal and the refuse should be

rewashed to segregate the middlings. While attempts have been made with jigs to produce three products in operation, I have always found them inefficient.

I have experimented for over 5 years with centrifugal driers and must admit that they alone will not successfully dry fine coal when designed for continuous operation. While great claims are made for special systems of draining bins and pits, one has only to visit washeries so equipped, observe the auxiliary equipment added and question the operator of the plant, to learn that these have failed to satisfactorily solve this problem. With this feature still unsolved, the concentration table has serious drawbacks, unless it is necessary to wash the fine coal separately. The fact should be borne in mind, in the study of these questions, that coal washing is each day becoming more closely allied with the byproduct coke plant and high moisture cannot be countenanced in the coal to such ovens. We must forget the loop-holes the beehive oven let us crawl through in the past.

I have never doubted the ability of Dorr thickeners to clarify the water but have seriously doubted the finding of a successful way to handle the underflow or sludge. To send it to the dewatering buckets or to the centrifugal driers was sure to result as the author states; to pump it directly onto the dried coal after the centrifugal driers has the great advantage, as stated by the author, of sure elimination from the washery system, yet the added moisture is a serious drawback, particularly where the washed coal is loaded on cars for shipment. This is another argument in favor of elimination of the fines, unless, as Mr. Campbell says, someone devises a successful dry concentrator.

Mr. Campbell's statement that "mathematical deductions are more accurate than the actual weights," etc. agrees with my experience. It should be added, however, that analyses should be based on large and frequent sampling. The plant of the future should be equipped with automatic samplers, not put in as an after thought but designed in with just as much careful attention as any other part of the equipment. Thousands of dollars are frequently spent on laboratories and chemists to work on samples that are not representative.

G. H. Elmore, Philadelphia, Pa.—Not so many years ago, those of us who were engaged in this line of work had an idea that, if anyone brought to us a coal that contained 2½ per cent. sulfur, we could probably wash it down to about 1¼; that you could take 50 per cent. of the sulfur out of the coal by good, careful washing. That conclusion was absolutely wrong. Furthermore, to simply get an assay of a coal and then try to divine what can be done to that coal by careful washing is utterly impossible. I would rather depend on a physical and visual examination of a coal than a mere assay of it.

Another belief was that the sulfur followed the ash; consequently, if you could get the ash out of the coal, you could get most of the sulfur with it. That was an off-hand rule and was founded on limited experience, but as a rule, it is not true, although there are cases where it would seem to be. Exceptions are so much at hand all the time in studying this problem of the removal of sulfur from coal that we cannot form any rule about them. I have recently been investigating coal from Brazil. It is a very high-ash coal, as are most Brazilian coals. The raw coal runs 30 to 40 per cent. ash and from 4 to 5 per cent. sulfur. The washed coal carries between 16 and 17 per cent. ash, and 0.02 to 0.03 per cent. sulfur. In other words, here is a very high-ash and high-sulfur raw coal, which, when washed, makes a washed coal almost free from sulfur. I have never seen anything like it. As a washed coal, it is one of the lowest in sulfur and one of the highest in ash.

In another case, regarding which several of us are more or less familiar, where the sulfur runs approximately 3 per cent. of the raw coal, the best washed coal that can be obtained has on the average about 2¼ per cent. sulfur. A visual examination of this raw coal shows scarcely any sulfur at all, being a nice, black, clean coal, and, from the standard of either the finished product or the assay, is entirely deceiving as to the possibilities involved for removing the sulfur from it. The only way to determine the results that can be obtained from the washing process is to make a very carefully conducted washing test; you cannot take anything for granted from start to finish. If the investment to be made is a large one, the test should not stop until you have actually built a full-sized machine and other necessary equipment, erected it at the mine, and made a series of test runs under varying conditions, of maximum size, etc. Then put the coal through the coking plant in sufficient tonnage to get regular runs in the blast furnace.

C. A. Meissner, Brooklyn, N. Y.—We have not said anything about the coal washery at Middlefork as yet, because we are not ready. Mr. Campbell correctly states that a good deal of the work is strictly in an experimental stage. We have gone in on a large scale based on a large amount of work that was done in an experimental washery at Joliet, under Mr. Carl Wendell, and I think we are going to obtain all the results we expect. What has interested us very much indeed is this discussion on the question of organic sulfur. We have naturally felt that we ought to be able to reduce our sulfur quite materially. We have seen other mines where table work has reduced sulfur to a very remarkable degree, and we have often wondered why we could not do similar work. If, however, we have from 1 to 1½ per cent. of organic sulfur in our Illinios coals, we must not forget that we cannot wash it out. It is going to stay there.

I want to call attention to another point, that is, the form or the occurrence of sulfur in the coal in the seams. We have a rather curious condition at Middlefork. The center of our seam will be a fairly low sulfur, from 1½ to 2½ per cent. The upper portion of the seam, as I remember about 6 or 8 in., will run up from 2½ to 7 per cent. in sulfur. The lower part, also, say 6 or 8 in., may run from ½ up to 6 per cent. in sulfur. The middle of the seam takes in a large portion of the vein and is fairly averaged in sulfur as against the variations in the upper and lower portions. There is a problem for the geological part of the subject that may be given some study.

# Coals of Ohio and Their Limitations for Byproduct Coke

BY WILBER STOUT,* COLUMBUS, OHIO

(Chicago Meeting, September, 1919)

IN Ohio, the annual output of coke made from native coals has averaged not more than 70,000 tons, or about enough to run a 200-ton blast furnace. Raw coal locally mined from the Sharon, or No. 1, bed was used for iron smelting in Mahoning furnace at Lowellville, Mahoning Co., as early as 1846. This practice was continued in the Youngstown district until the coal became scarce and the Connellsville coke more popular. At present, the Sharon bed contributes a part of the fuel for the production of ferrosilicon iron at Jackson, Jackson Co. About one-third native coal and two-thirds soft coal are used.

In this state, coke, on a commercial basis, was first made in Jefferson County probably about 1861 or 1862. The coals used were the Steubenville Shaft, or Lower Freeport, and the Strip Vein or Middle Kittanning. The practice, however, was discontinued before 1880 because of the competition of superior foreign cokes. Beehive ovens, later built at Leetonia, Columbiana Co., were operated for only a few years; the coal used at this place is the Lower Kittanning or No. 5. In a desultory way, slack from the Pittsburgh coal has been coked at a few places in eastern Ohio, and attempts, either for experimental or for commercial purposes, have been made in other parts of the state. Small quantities of coke were made from the Upper Freeport coal at Salineville, Columbiana Co., and near Nelsonville, Hocking Co.; the results were poor. Tests were made of the Middle Kittanning coal near Zanesville, Muskingum Co., at Moxahala Furnace, Perry Co., and at Washington Furnace, Lawrence Co.; the cokes were all high in sulfur and ash. Attempts were made to coke the Clarion coal at Vinton Furnace, Vinton Co.; a small battery of molding ovens and a pile of high-sulfur coke yet remain to tell the story. The Pittsburgh coal was coked on a commercial scale at Utley and at Lathrop, Athens Co., but the operations were not successful and were soon abandoned.

These failures and indifferent successes are not to be attributed to a lack of demand for a first-class coke, as Ohio has held a commanding place for many years in iron smelting; nor to a deficiency of coals, as the state has a vast reserve of such material. They are to be ascribed to the inferior quality of the native products offered to the trade. The cokes

* Assistant Geologist, Geological Survey of Ohio.

were too high in ash and sulfur or were wanting in structure. Some of the purest coals have but little cementing property whereas others with high sulfur and ash have, at least, fair bonding qualities. Further, Ohio coals are all high in moisture and volatile matter and, therefore, give low yields of coke. Broadly speaking, this state has no coals available

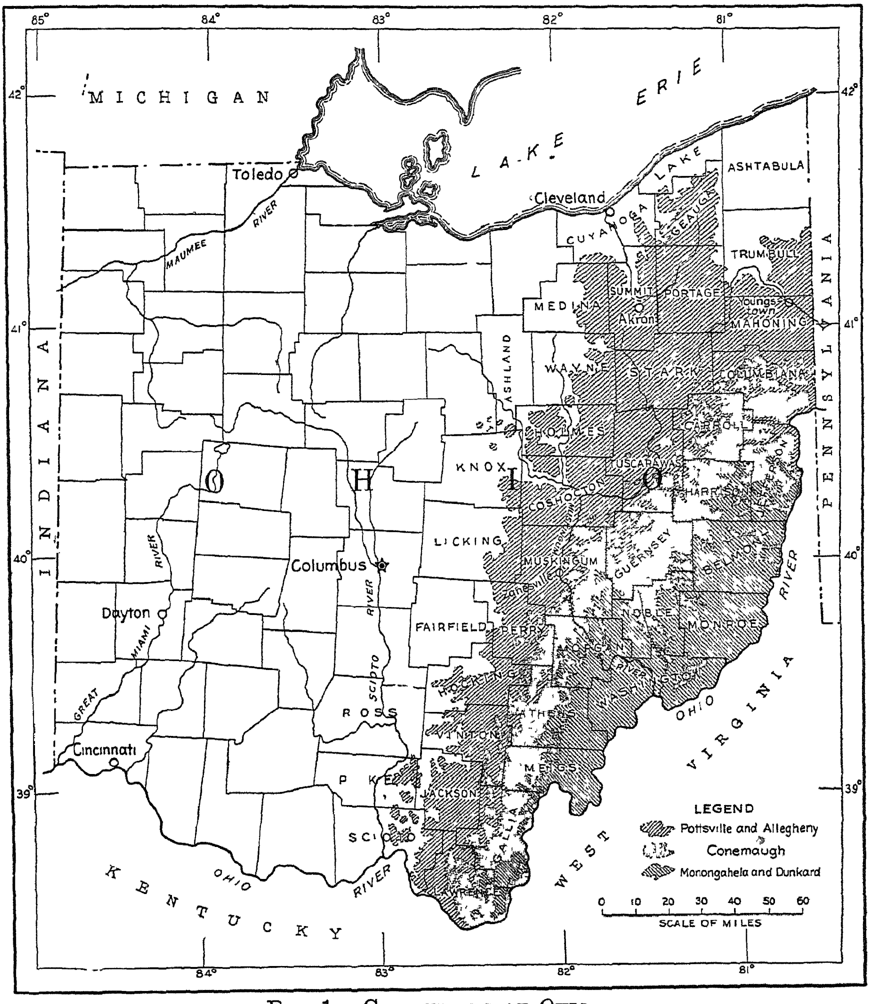


FIG. 1.—COAL FIELDS OF OHIO.

in large quantities and at a low cost that will make a first-class coke in the beehive oven.

The introduction of the retort oven, however, has changed the situation somewhat. Coals not successfully coked in the beehive oven may make a fair product in the byproduct oven. Further, mixtures of coals often give good results. Besides the coke, other products of value are

also obtained and the yield of these is high with the highly volatile coals. In this state, the future prospects are good for the sale of the excess oven gas for domestic and industrial purposes as the supply of natural gas seems near exhaustion. The byproduct oven is a profitable investment in Ohio as attested by the number that have been built and are now under construction. These plants are widely distributed over the state, the kinds and locations are given in Table 1. Besides, there was under construction by the Domestic Coke Corpn , Cleveland, a 60-oven plant, of the Semet-Solvay type, with an annual coke capacity of 311,000 tons.

TABLE 1.—*Byproduct Coke Plants in Ohio, Jan.* 1, 1919*

| Name of Company | Number of Ovens | Kind of Oven | Annual Coke Capacity |
|---|---|---|---|
| Cleveland Furnace Co., Cleveland | 100 | Semet-Solvay | 337,500 |
| Republic Iron & Steel Co., Youngstown | 134 | Koppers | 744,600 |
| Youngstown Sheet & Tube Co., Youngstown | 306 | Koppers | 1,425,000 |
| Toledo Furnace Co., Toledo | 94 | Koppers | 408,600 |
| United Furnace Co., Canton | 47 | Koppers | 204,400 |
| Hamilton Otto Coke Co., Hamilton | 100 | Otto | 168,000 |
| River Furnace Co., Cleveland | 204 | Koppers | 960,000 |
| LaBelle Iron Works, Steubenville | 94 | Koppers | 445,000 |
| Brier Hill Steel Co., Youngstown | 84 | Koppers | 379,000 |
| Portsmouth-Solvay Co., Portsmouth | 108 | Semet-Solvay | 559,000 |
| National Tube Co , Lorain, O | 208 | Koppers | 850,000 |
| Ironton Solvay Coke Co , Ironton | 60 | Semet-Solvay | 311,000 |
| American Steel & Wire Co., Cleveland | 180 | Koppers | 750,000 |
| Dover By-Product Co., Dover, O | 24 | Roberts | 100,000 |

* List furnished by M. J. Tucker, Librarian, *Iron Trade Review*, Cleveland, Ohio.

These operations are all successful, although they are handicapped by having to haul their coals from places in Pennsylvania, West Virginia, and eastern Kentucky. When the additional expense of transporting coke over coal is considered, but little lowering of cost is gained by placing the ovens in the coal fields; moreover, the products are not so readily handled. In none of these plants have Ohio coals been used regularly, even as a part of the charge, except in the Roberts oven at Dover. The only byproduct plant in Ohio using local coal exclusively is at Tunnel Hill, Coshocton Co., which has been in operation about 3 years. In this a cannel coal obtained from the hills nearby is coked chiefly for its coal-tar products, the yield of which is high. The coke produced is not of metallurgical quality.

## STRUCTURE AND DISTRIBUTION OF COAL-BEARING ROCKS

The chief coal-bearing rocks of Ohio are of Pennsylvanian age, although the deposits laid down during Dunkard time contain a few thin

TABLE 2.—*Stratigraphic Relations*

| | Formation | Member | General Description | Thickness Ft | Thickness In |
|---|---|---|---|---|---|
| Dunkard Series | Washington | Washington "A" | Coal, thin, unsteady | 2 | 0 |
| | | | Interval | 60 | 0 |
| | | Washington | Coal, persistent, shaly | 5 | 0 |
| | | | Interval | 65 | 0 |
| | | Waynesburg "A" | Coal, impure, rather steady | 2 | 0 |
| | | | Interval | 38 | 0 |
| Pennsylvania System | Monongahela | Waynesburg, No 11 | Coal, persistent, shaly | 3 | 0 |
| | | | Interval | 60 | 0 |
| | | Uniontown, No 10 | Coal, locally developed | 3 | 0 |
| | | | Interval | 107 | 0 |
| | | Meigs Creek, Sewickley, No 9 | Coal, persistent | 4 | 0 |
| | | | Interval | 34 | 0 |
| | | Pomeroy, Redstone, No 8a | Coal, locally developed | 4 | 0 |
| | | | Interval | 30 | 0 |
| | | Pittsburgh, No 8 | Coal, persistent | 5 | 0 |
| | Conemaugh | | Interval | 188 | 0 |
| | | Harlem. | Coal, persistent, but thin | 1 | 6 |
| | | | Interval | 28 | 0 |
| | | Barton. | Coal, thin, unsteady | 1 | 0 |
| | | | Interval | 29 | 0 |
| | | Anderson | Coal, persistent, thin | 2 | 0 |
| | | | Interval | 23 | 0 |
| | | Wilgus | Coal, locally developed | 3 | 0 |
| | | | Interval | 56 | 0 |
| | | Mason | Coal, unsteady, thin | 1 | 0 |
| | | | Interval | 30 | 0 |
| | | Mahoning | Coal, persistent, usually thin | 1 | 0 |
| | | | Interval | 37 | 0 |
| | Allegheny | Upper Freeport, No 7 | Coal, locally developed | 5 | 0 |
| | | | Interval | 45 | 0 |
| | | Lower Freeport, No 6a | Coal, unsteady, usually thin | 3 | 0 |
| | | | Interval | 41 | 0 |
| | | Middle Kittanning, No 6 | Coal, persistent | 4 | 6 |
| | | | Interval | 31 | 0 |
| | | Lower Kittanning, No 5 | Coal, locally developed | 3 | 0 |
| | | | Interval | 27 | 0 |
| | | Clarion, No 4a | Coal, unsteady | 3 | 6 |
| | | | Interval | 24 | 0 |
| | | Brookville, No 4 | Coal, thin, unsteady | 3 | 0 |
| | Pottsville | | Interval | 22 | 0 |
| | | Tionesta, No 3b | Coal, usually thin | 4 | 0 |
| | | | Interval | 17 | 0 |
| | | Bedford | Coal, unsteady, in places cannel coal | 2 | 0 |
| | | | Interval | 14 | 0 |
| | | Upper Mercer, No. 3a | Coal, unsteady. | 3 | 0 |
| | | | Interval | 36 | 0 |
| | | Lower Mercer, No 3 | Coal, thin, unsteady | 2 | 6 |
| | | | Interval | 74 | 0 |
| | | Quakertown, No 2 | Coal, locally developed | 3 | 0 |
| | | | Interval | 87 | 0 |
| | | Sharon, No 1 | Coal, unsteady | 3 | 0 |

beds of low-grade fuel. The total thickness of such strata is, approximately, 1500 ft. (457 m.) Of this, not to exceed 70 ft. (21 m.) is coal, much of which is found in beds too thin to be of value except for local domestic purposes. The coal fields lie in the eastern and southeastern parts of the state and have a total area of over 10,000 sq. mi. (2,589,900 ha.) The Pennsylvanian system is divided into four formations, the Pottsville, Allegheny, Conemaugh, and Monongahela; and the Dunkard series into two, the Washington and Green. These formations are further subdivided into members. The chief stratigraphic relations are shown in Table 2, in which the average thicknesses of the coals in the main fields and not for the total area are given. The intervals separating the beds are normal for the state.

*Pottsville Formation*

The Pottsville, the oldest formation in the Pennsylvanian system, extends southward from Trumbull and Geauga Counties, on the north, to the Ohio River, on the south, in Scioto and Lawrence Counties. The belt varies from 10 to 40 mi. (16 to 64 km.) in width but averages only about 15 mi. This formation is nearly 250 ft. (76 m.) thick and contains at least ten well-defined coal beds of which, however, only four have been mined for railroad shipment; in fact, only two beds have been extensively mined. In general, the quality of the Pottsville coals is excellent but they have little coking property.

*Sharon or No. 1 Coal.*—The Sharon or No. 1 coal lies on or not far above the Sharon conglomerate, the basal member of the Pottsville formation. Its position is also about 90 ft. (27 m.) below the Quakertown, or No. 2, coal. It is confined to two small areas, one in the northern part of the state, centering around Massillon, Stark Co., and the other in the southern part, lying near Jackson, Jackson Co. In these fields, the bed is broken into somewhat isolated and irregular pockets, which causes trouble and expense in mining. The Massillon is by far the larger and more important and embraces parts of Stark, Wayne, Summit, Portage, Medina, Mahoning, and Tuscarawas Counties; only a small part of the area, however, contains coal. At present the field is only a small producer as much of the coal has been exhausted. This fuel was formerly used quite extensively in the raw state for iron smelting in the Youngstown district. The following section is about typical for the bed:

| | Feet | Inches |
|---|---|---|
| Shale and sandstone | 30 | 0 |
| Coal, Sharon | 4 | 6 |
| Clay, siliceous | 4 | 0 |
| Conglomerate, Sharon | 30 | 0 |

Where present, the thickness of the Sharon coal varies from 1 to 7 ft. (0.3 to 2.1 m.) but the usual measurement is between 3 and 5 ft. (0.9 and 1.5 m.). The bed is not regularly separated into benches by partings but in places it contains thin layers of bony material. Sulfur balls are uncommon. The roof is shale, strong and durable, and the floor is a siliceous clay, well able to support the weight of the overburden. Under such conditions a high yield of fuel is obtained from a given area. The average quality of the coal is shown by the following analysis: Moisture, 5.52 per cent.; volatile matter, 37.60 per cent.; fixed carbon, 52.84 per cent.; ash, 4 04 per cent.; sulfur, 1.17 per cent. In parts of the field, the sulfur is considerably below 1 per cent. The moisture, volatile matter, and fixed carbon are rather constant and the ash seldom exceeds 6 per cent. The coal is open-burning and non-cementing and therefore not fitted for coke making. Moreover, the yield in the byproduct oven would be low.

In the Jackson district, the Sharon coal is confined to parts of Scioto, Liberty, Coal, Washington, and Jackson townships, Jackson Co., and to Marion, Beaver, and Jackson townships, Pike Co. The total area is approximately 28 sq. mi. of which not more than 15 sq. mi. is left for future consumption. This is so broken, however, that but little of it could be mined in a large way. The following section is representative of the coal worked at present:

| | Feet | Inches |
|---|---|---|
| Shale | 20 | 0 |
| Coal, good } Sharon | 2 | 9 |
| Coal, bony } Sharon | | 2 |
| Clay, siliceous | 1 | 0 |
| Conglomerate, Sharon | 40 | 0 |

The Sharon coal in the Jackson district varies from 2 to 5 ft. in thickness but averages close to 3 ft. The conditions for mining are good except that the coal lies on the uneven floor of the Sharon conglomerate and therefore contains rolls and dips. The composition is: Moisture, 8.20 per cent.; volatile matter, 32.88 per cent.; fixed carbon, 52.55 per cent.; ash, 6.37 per cent.; sulfur, 0.49 per cent. The Sharon coal in the Jackson district contains less sulfur than any other coal in the state and on this account it has been regularly used for many years for iron smelting. The ash is claylike in character and in local areas of the field is high. The coal is non-coking and in every respect similar to the Massillon.

The unfortunate thing is that the supply of Sharon coal in either the Massillon or the Jackson district is scarcely sufficient to warrant the establishment of important works. Mixtures of this coal with a high cementing coal, for example standard Connellsville from Pennsylvania, should give a product of metallurgical quality. A half-and-half mixture of these two coals should give a yield of approximately 62.24 per cent.

coke,[1] having an analysis about as follows: volatile matter, 1.61 per cent.; fixed carbon, 89.56 per cent.; ash, 8.83 per cent.; sulfur, 0.90 per cent. No tests, however, have been made with such a mixture of coals to determine the yield and structure of coke. The results may be unsatisfactory.

*Quakertown, Wellston, or No. 2 Coal.*—In ascending order, the next coal of importance in the Pottsville formation is the Quakertown, or No. 2, which is also known as the Wellston and Jackson Hill coal. It lies on the average about 90 ft. (27 m.) above the Sharon coal and nearly 100 ft. below the Lower Mercer limestone, which is a useful bench for reference, and is confined to two small areas in southern Ohio, one around Wellston, Jackson Co., and the other near McArthur, Vinton Co. The Wellston field, which has been worked since 1872, is by far the larger. The total area of the field is nearly 40 sq. mi., but not more than 15 sq. mi. remains for future mining. In the part first worked, but now practically exhausted, the bed is 3 ft. to 4 ft. 6 in. (0.9 to 1.3 m.) thick but in that now mined it will not average over 2 ft. 6 in. For domestic purposes, this fuel has had a wide reputation for many years. The quality of the coal is shown by the following analysis: Moisture, 9.29 per cent.; volatile matter, 32.96 per cent.; fixed carbon, 54.26 per cent.; ash, 3.49 per cent.; sulfur, 1.25 per cent.

The bed is quite free from shale or clay partings and from nodular pyrite. The roof is a tough shale and the floor a siliceous clay, both favorable for high yields in mining. For a thin coal, all the mining conditions are excellent. In the McArthur field, the members average about 3 ft. in thickness but the known area is only a few square miles. The quality of the fuel is excellent, being low in both sulfur and ash.

The Quakertown is one of the purest coals in Ohio and the structure of the bed is such that but little extraneous material is introduced into the fuel in mining. It is a free-burning, non-cementing coal and therefore is not suited for coke making. The cost of mining a thin bed and the small supply now known are also conditions unfavorable even for using it as a part of the burden. The Quakertown coal must be excluded from the list for making byproduct coke.

*Lower Mercer, No. 3 Coal.*—The Lower Mercer, or No. 3, coal lies from 10 to 30 ft. (3 to 9 m.) below the Lower Mercer limestone and is very unsteady in its extension across the state, being wanting in many places. Where present it is usually thin, seldom expanding to as much

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[1] In calculations used in this paper involving Pittsburgh or No. 8 coal from Pennsylvania the composition is assumed to be as follows: Moisture, 2.60 per cent.; volatile matter, 31.80 per cent.; fixed carbon, 58.64 per cent.; ash, 6.96 per cent.; sulfur, 1 08 per cent. The yield in coke in the byproduct oven is considered to be the fixed carbon plus the ash plus 1 per cent. for unexpelled volatile matter and deposited carbon from dissociated paraffins The loss in sulfur is taken to be one-half or one atom of sulfur split off from the pyrite molecule.

as 3 ft. In general the quality of the fuel is poor as it is high in sulfur and ash. In a few places, cannel coal is present on the horizon. The Lower Mercer member has furnished no coal for railroad shipment and only small quantities even for local domestic purposes.

*Upper Mercer, No. 3a, Coal.*—The normal position of the Upper Mercer, or No. 3a, coal is about midway in the interval between the Lower Mercer and the Upper Mercer limestones. In the southern part of the state, the member is present with some regularity but in the central and eastern parts it is more uncertain. The thickness of the bed is seldom as much as 3 ft. (0.9 m.) and usually is below 2 ft. The quality of the fuel is generally very good, as it is low in sulfur and ash. The Upper Mercer coal has been mined only for local domestic purposes.

*Bedford Coal.*—The Bedford member, also belonging to the Mercer group of coals, lies directly, or at most only 1 ft. (0.3 m.) or so, below the Upper Mercer limestone. It is unsteady in extent and thickness and variable in composition and structure. It is found in local areas in Muskingum, Licking, Coshocton, Tuscarawas, Holmes, Stark, and Mahoning Counties. The best known field is in Bedford Township, Coshocton Co., where it has excellent thickness over a few square miles and where it is represented by both cannel and bituminous coal. The thickness varies from 3 to 9 ft. but averages about 5 or 6 ft. The following section taken near the village of Mohawk, Coshocton Co., shows the stratigraphic relations:

| | | Feet | Inches |
|---|---|---|---|
| Flint, gray, calcareous | Upper Mercer | 1 | 3 |
| Shale | Upper Mercer | | 2 |
| Limestone, gray, shaly | Upper Mercer | 2 | 2 |
| Shale, gray | Upper Mercer | 2 | 8 |
| Flint, black | Upper Mercer | 1 | 9 |
| Coal, bituminous, with bony partings | Bedford | 2 | 2 |
| Coal, cannel | Bedford | 5 | 6 |
| Shale and covered | | 14 | 0 |
| Limestone, Lower Mercer | | 3 | 6 |

The bituminous coal occurring with the cannel is usually of poor quality as it is high in ash. The mining conditions are very good. The quality of the cannel coal is shown by the following analysis: moisture, 2.35 per cent.; volatile matter, 47.05 per cent.; fixed carbon, 37.00 per cent.; ash, 13.60 per cent.; sulfur, 2.33 per cent. At present this coal is coked by one plant chiefly for its coal-tar products; the coke is not of metallurgical quality.

*Tionesta, No. 3b, Coal.*—The Tionesta, the highest coal bed in the Pottsville formation, lies about midway in the interval between the Upper Mercer and the Putnam Hill limestones. It is very unsteady, appearing in workable thickness only in small isolated areas across the state, the largest of which are found in Scioto, Jackson, Vinton, Tuscarawas, and

Stark Counties. In the best developed areas, the bed is from 3 to 6 ft. thick and is usually broken by one or more partings. At present, this coal is mined for railroad shipment only near McArthur in Vinton County. The section here follows:

| | | Feet | Inches |
|---|---|---|---|
| Shale, calcareous, fossiliferous | | 10 | 0 |
| Coal, part bony | Tionesta | 1 | 4 |
| Clay | Tionesta | | 7 |
| Coal, part cannel | Tionesta | 3 | 8 |
| Clay, siliceous | | 4 | 0 |

The Tionesta coal is usually high in ash that is clay-like in character. It is a free-burning fuel with little cementing quality and therefore not fitted for coke making. Further, the areas in which the coal has workable thickness are too small to yield the large quantities of fuel demanded for byproduct ovens.

*Allegheny Formation*

The Allegheny is the most important coal-bearing formation. It extends in a belt from Mahoning County, on the north, across Stark, Tuscarawas, Coshocton, Muskingum, Perry, Hocking, Athens, Vinton, and Jackson Counties to Lawrence County on the Ohio River. The thickness of the formation varies from 150 to 250 ft. and averages nearly 200 ft. (60 m.). It contains six well-defined coal beds, five of which have been mined in a large way. It begins with the Brookville, or No. 4, coal and ends with the Upper Freeport, or No. 7, member.

*Brookville, No. 4, Coal.*—The Brookville coal, the basal member of the Allegheny formation, lies directly below the Putnam Hill limestone where this member is present. This coal is locally developed in parts of Vinton, Coshocton, Tuscarawas, Holmes, Wayne, and Stark Counties. The bed has been mined at only a few places for railroad shipment but it has been worked in many localities for small supplies of domestic fuel. The largest field is in Stark and Tuscarawas Counties, where the bed is rather steady and from 3 to 6 ft. thick. The following section is about representative of the Brookville coal in the Canton district of Stark County:[2]

| | | Feet | Inches |
|---|---|---|---|
| Limestone, Putnam Hill | | 5 | 0 |
| Coal | Brookville | 3 | 0 |
| Clay | Brookville | | 2 |
| Coal, bony | Brookville | | 8 |
| Coal | Brookville | 2 | 2 |

In Coshocton, Holmes, Wayne, and Vinton Counties, the best deposits have a thickness of about 4 ft. (1.2 m.) but they are usually confined to a few square miles in extent. The bed is commonly broken by partings of

[2] Ohio Geol. Survey, 5, 232.

clay and bone coal, which cause some trouble in keeping the fuel clean; otherwise the mining conditions are good. The quality of the Brookville coal is shown by the following analysis: moisture, 5.29 per cent.; volatile matter, 39.10 per cent.; fixed carbon, 46.64 per cent.; ash, 8.97 per cent.; sulfur, 3.29 per cent.

The Brookville coal has weak cementing properties but is never low in sulfur and ash. The coal in the lower part of the bed is more impure than that in the upper part but neither is of first-class quality. As the sulfur occurs both in the form of pyrite and in that of organic sulfide, the fuel is but little improved by washing. Large quantities of Brookville coal are yet available but the quality excludes its use for byproduct coke.

*Clarion, No. 4a, Coal.*—In ascending order, the second coal bed in the Allegheny formation is the Clarion, or No. 4a, which lies either directly or only a few feet below the Ferriferous or Vanport limestone and from 20 to 35 ft. (6 to 10 m.) below the Lower Kittanning coal. It is present in the northeastern and southern parts of the state and is thin or wanting in the central part. Small deposits, varying from 3 to 4 ft. in thickness, have been worked in northern Columbiana County for railroad shipment. The quantity of fuel available in this part of the state is small and the quality poor. The important field of Clarion coal in Ohio is located in southern Vinton, eastern Jackson, eastern Scioto, western Gallia, and northern Lawrence Counties. The area is large and the bed persistent. The following section is representative:

| | | Feet | Inches |
|---|---|---|---|
| Limestone, Ferriferous or Vanport | | 6 | 0 |
| Shale | | 1 | 0 |
| Coal | Clarion. | 1 | 3 |
| Clay | Clarion. | | 7 |
| Coal | Clarion. | 1 | 4 |
| Clay with pyrite | Clarion. | | 1 |
| Coal | Clarion. | 1 | 0 |
| Clay, siliceous | | 2 | 0 |

The average thickness of clean coal is about 3 ft. 6 in. (1 m.). The bed is everywhere broken by the two partings and in places by thin layers of bone coal. The fuel produced by the ordinary methods of mining is dirty. An analysis of the coal is:[3] moisture, 5.34 per cent.; volatile matter, 38.94 per cent.; fixed carbon, 44.92 per cent.; ash, 10.80 per cent.; sulfur, 4.33 per cent.

The Clarion coal cokes readily and yields a product of excellent structure, as shown by the results obtained at Vinton Furnace, Vinton Co., in Welsh ovens. The ash and sulfur, however, are far above the limits demanded for coals for byproduct practice. Washing lowers the ash considerably but it is not very effective in reducing the sulfur, for about

[3] Ohio Geol. Survey *Bull.* 9, 291.

half of this is present in organic combination and the remainder in disseminated pyrite. Moreover, the yield of coke from such a coal is low. Although present in large quantities and coking freely, the Clarion coal is thus excluded from the field of byproduct coke on account of its high ash sulfur.

*Lower Kittanning, No. 5, Coal.*—The position of the Lower Kittanning coal is normally 20 to 35 ft. above the Ferriferous limestone and 25 to 45 ft. below the Middle Kittanning coal. Although locally wanting, the member may be traced readily from the Ohio-Pennsylvania line in Columbiana County across Stark, Carroll, Tuscarawas, Coshocton, Muskingum, Perry, Hocking, Vinton, and Jackson Counties to the Ohio River in Lawrence County. This member is an important source of fuel in a number of districts in the state but the horizon is better known on account of the great bed of plastic clay associated with the coal. The Lower Kittanning is one of the few Ohio coals that coke freely but it has limitations due to detrimental impurities. It was coked in a small way for many years in the Leetonia field of Columbiana County. The product was somewhat brittle but otherwise had fair properties. The area is now largely exhausted hence it need not be considered further. In parts of Stark County, the bed is rather steady and from 3 to 4 ft. thick. The following measurement is about representative of the best deposits:

| | | Feet | Inches |
|---|---|---|---|
| Shale | | 10 | 0 |
| Coal | Lower Kittanning | 1 | 10 |
| Shale | Lower Kittanning | | 2 |
| Coal | Lower Kittanning | 2 | 0 |
| Clay | | 6 | 0 |

In this county the analysis of the coal is, approximately, as follows: moisture, 3.83 per cent.; volatile matter, 42.10 per cent.; fixed carbon, 50.22 per cent.; ash, 3.85 per cent.; sulfur, 2.25 per cent. The sulfur occurs largely as pyrite in concretionary form hence the coal would be improved by washing. The quantities yet available, however, are not large.

The Lower Kittanning coal holds its thickness well and extends over a wide area in Tuscarawas County. The thickness varies from 2 ft. 6 in. to 5 ft. but averages more than 3 ft. (0.9 m.). The quantities of fuel yet available are large. In parts of the fields, the bed contains considerable pyrite in large nodules or sheets, which are usually bedded along definite zones; washing would remove most of this. The following measurement is about an average for the field:

| | | Feet | Inches |
|---|---|---|---|
| Shale | | 10 | 0 |
| Coal | Lower Kittanning | 2 | 6 |
| Shale with pyrite | Lower Kittanning | | 2 |
| Coal | Lower Kittanning | | 6 |
| Clay | | 9 | 0 |

The analysis follows: moisture, 4.30 per cent.; volatile matter, 40.83 per cent.; fixed carbon, 46.67 per cent ; ash, 8.20 per cent ; sulfur, 3 43 per cent.

In a small field north of Newcomerstown, Tuscarawas Co., the Lower Kittanning coal is very pure, being low in both sulfur and ash. By choosing the best deposits of coal in the county, a fair grade of coke could be made but the yield would be low, approximately 56 per cent. The by-products, however, would be high and find a ready sale.

In Guernsey, Coshocton, and Muskingum Counties, the Lower Kittanning coal is very unsteady and generally of poor quality. The continuity of the bed is much better in Perry County, where large acreages are present in the northeastern and southern parts. The thickness of the bed varies from 3 to 5 ft. but averages nearly 4 ft. The section given below was obtained in Pike Township.[4]

| | | Feet | Inches |
|---|---|---|---|
| Shale | | 0 | 7 |
| Coal | Lower Kittanning | 0 | 6 |
| Horn coal | Lower Kittanning | | 1 |
| Coal | Lower Kittanning | 1 | 0 |
| Parting | Lower Kittanning | | ½ |
| Coal | Lower Kittanning | 1 | 9 |
| Clay | | 1 | 0 |

The approximate analysis of the coal in Perry County is: moisture, 6.90 per cent.; volatile matter, 38.53 per cent.; fixed carbon, 47.32 per cent.; ash, 7.25 per cent.; sulfur, 2.90 per cent.

Everywhere in the field the sulfur is high, seldom being below 2 per cent.; the improvement by washing would not be sufficient to give a first-class product. The ash in the coke, however, could be kept below 10 per cent.

In its extension southward, the Lower Kittanning coal is not important until Jackson and Lawrence Counties are reached; here the bed holds fair thickness over rather large areas, expanding from 2 ft. 6 in. to 5 ft. The average measurement and structure are shown in the following section:

| | | Feet | Inches |
|---|---|---|---|
| Shale | | 20 | 0 |
| Coal | Lower Kittanning | 2 | 4 |
| Clay | Lower Kittanning | | 2 |
| Coal | Lower Kittanning | | 6 |
| Clay | | 6 | 0 |

The mining conditions are fair and with a little care the coal may be kept free from extraneous materials. The approximate composition is: moisture, 6.43 per cent.; volatile matter, 38.34 per cent.; fixed carbon, 47.22 per cent.; ash, 8.01 per cent.; sulfur, 2.29 per cent. Carload sam-

[4] Ohio Geol. Survey *Bull.* 9, 187.

ples of this coal washed and then coked in beehive ovens gave products with the following analysis:[5]

| | Per Cent | Per Cent |
|---|---|---|
| Volatile matter | 2 95 | 2 60 |
| Fixed carbon | 86 60 | 88 85 |
| Ash | 10 45 | 8 55 |
| Total | 100 00 | 100 00 |
| Sulfur | 1 004 | 1 35 |

The Lower Kittanning coal was coked for years by the Ashland Coal & Iron Co. at Ashland, Ky. The coke produced was of fair quality but the yield was low. By mixing this coal with Pocahontas or other good grades of West Virginia coal, both the yield and the quality of the product could be considerably increased. The Lower Kittanning member offers the best possibilities for coke making in the byproduct oven of any coal in southern Ohio.

*Middle Kittanning, No.* 6, *Coal.*—Bownocker says,[6] "The Middle Kittanning coal, on account of its quantity and quality, is the most valuable in Ohio. It is found along the state line in Columbiana County and can not only be followed with ease across the state to Lawrence County, on the Ohio River, but it is worked in every county where it should appear above drainage and in most of them on a large scale." It has its maximum development in the Hocking Valley field in Perry, Hocking, and Athens Counties. The bed lies from 25 to 45 ft. (7.6 to 13.7 m.) above the Lower Kittanning coal and from 80 to 100 ft. below the Upper Freeport member.

The Middle Kittanning coal is regularly present in the eastern and and southern parts of Columbiana County, but it has an average thickness of less than 3 ft. The quality of the fuel, however, is very good. It was coked for a number of years near Hammondsville but as the cost of mining was high the operations were finally abandoned. The thickness and quality of this coal are also much the same in Stark County, where the best deposits are found in the eastern and southern parts. The Middle Kittanning coal is present above drainage over much of Tuscarawas County and is largely mined both for local purposes and for railroad shipment. The bed varies from 3 to 5 ft. in thickness. The average measurement is about as follows:

| | | Feet | Inches |
|---|---|---|---|
| Shale | | 10 | 0 |
| Coal | Middle Kittanning | 2 | 4 |
| Shale | | | 2 |
| Coal | | 1 | 3 |
| Clay, siliceous | | 3 | 0 |

[5] Ohio Geol. Survey *Bull.* 20, 386.
[6] U. S. Geol. Survey *Prof. Paper* 100-B, 49.

In places, a few inches of bony coal appears above the main bed. Thin partings are also present in both the upper and lower benches of coal in some mines. The average composition of the fuel is: moisture, 3.76 per cent.; volatile matter, 41.31 per cent.; fixed carbon, 48.16 per cent.; ash, 6.77 per cent.; sulfur, 3.85 per cent. This analysis shows that the sulfur is far too high for byproduct coke. Washing improves the coal considerably but not to the extent necessary for good results. The thickness and composition of the Middle Kittanning coal in Coshocton and Muskingum Counties are much the same as in Tuscarawas. The quantities of fuel available in both counties are large and the mining conditions excellent. In the Hocking Valley district of Perry, Hocking, and Athens Counties, the bed has excellent continuity and swells to 5 ft. or more in thickness. It is regularly present in three benches, which in places are further separated by irregular bony layers. On the average, the Middle Kittanning coal in this district is lower in sulfur than it is elsewhere in the state. The structure of the coal in the main part of the field follows:

| | | Feet | Inches |
|---|---|---|---|
| Shale | | 10 | 0 |
| Coal | Middle Kittanning | 1 | 8 |
| Shale and bone coal | Middle Kittanning | | 6 |
| Coal | Middle Kittanning | 2 | 0 |
| Shale | Middle Kittanning | | 4 |
| Coal | Middle Kittanning | 1 | 6 |
| Shale | Middle Kittanning | | 1 |
| Coal | Middle Kittanning | 1 | 6 |
| Clay, siliceous | | 3 | 0 |

The average composition is: moisture, 7.00 per cent.; volatile matter, 34.91 per cent.; fixed carbon, 51.38 per cent.; ash, 6.71 per cent.; sulfur, 1.32 per cent. In part of this field, the sulfur in the coal is less than 1 per cent. and the ash less than 6 per cent.; these could be considerably lowered also by washing. From clean coal, the yield of coke would be nearly 60 per cent. and the sulfur not above the standard of 1 per cent. The limitation, however, in the main Hocking field is that, as a whole, the Middle Kittanning coal has little coking power. The lower bench cements fairly well but the two above are lacking in this property. Satisfactory results could be obtained only by mixing it with some standard coking coal such as Pocahontas. Using a mixture of one-half Hocking Valley and one-half Pocahontas No. 3, the yield in coke would be approximately 71.28 per cent.,[7] the composition of the coke is: volatile matter, 1.40 per cent.; fixed carbon, 90.59 per cent.; ash, 8.01 per cent.; sulfur, 0.67.

[7] The average composition of 38 analyses of Pocahontas No. 3 coal given in West Virginia Geological Survey (1903), **2**, 695–6 is as follows: moisture, 0.26 per cent.; volatile matter, 17.27 per cent.; fixed carbon, 77.77 per cent; ash, 4.70 per cent.; sulfur, 0.62 per cent. These figures are used in the foregoing calculation.

By washing the Middle Kittanning coal, the ash could be lowered somewhat and the sulfur kept safely within the limits required. The byproducts, especially the excess gas, would have a ready market. Under such conditions, the Middle Kittanning coal in the Hocking Valley field has more favorable prospects for coke making than it has elsewhere in Ohio. The quantities of coal yet available are large and a constant supply assured. South of this field, in Vinton, Jackson, Gallia, and Lawrence Counties, the Middle Kittanning coal is thin and patchy, the best areas being only a few square miles in extent. Further, the quality of the fuel is generally poor.

*Lower Freeport, No. 6a, Coal.*—The Lower Freeport coal, lying about midway in the interval between the Middle Kittanning and Upper Freeport members, is very unsteady in Ohio. The only two areas worthy of consideration are near Steubenville and Amsterdam, Jefferson Co., where it is mined in a large way. The average thickness of the bed in the Steubenville region is nearly 4 ft. (1.2 m.). The quality of the coal is shown in the following analysis: moisture, 2.06 per cent.; volatile matter, 39.06 per cent.; fixed carbon, 53.96 per cent.; ash, 4.92 per cent.; sulfur, 1.79 per cent.

In this district, the Lower Freeport coal was formerly coked quite extensively for the local furnaces but at present is not so used. The product was not up to the present standard as the coke was rather brittle and dense and high in sulfur and ash. The composition is approximately as follows: volatile matter, 1.34 per cent.; fixed carbon, 91.28 per cent.; ash, 7.38 per cent.; sulfur, 1.97 per cent. The area of coal in the Amsterdam field is large and the thickness of the bed from 3 to 6 ft. (0.9 to 1.8 m.). The sulfur content, however, is more than 2.50 per cent. or it is too high for coke making.

*Upper Freeport, No. 7 Coal.*—The Upper Freeport coal, the youngest member in the Allegheny formation, extends from the Ohio-Pennsylvania line, in Columbiana County, southwesterly across the state to the Ohio River, in Lawrence County. The bed lacks persistency but at that it ranks third in importance. In general the fields are not large and are more or less disconnected. In Columbiana County, the Upper Freeport coal has been mined for many years near East Palestine and Salineville and the fields are yet far from exhaustion. The normal thickness of the bed is between 5 and 6 ft. (1.5 and 1.8 m.). The structure is shown by the following section:

| | | Feet | Inches |
|---|---|---|---|
| Sandstone, Mahoning | | | |
| Shale | | 3 | 0 |
| Shale, black | | 1 | 0 |
| Coal | Upper Freeport | 4 | 4 |
| Shale | Upper Freeport | | 2 |
| Coal | Upper Freeport | 1 | 0 |
| Clay | | 4 | 0 |
| Limestone | | 2 | 0 |

The coal in the upper bench contains less sulfur and ash than that in the lower bench, which in places is not removed from the mine. It cokes freely but the product is of poor quality owing largely to the percentage of sulfur. The composition of the fuel is. moisture, 2.32 per cent.; volatile matter, 39.08 per cent.; fixed carbon, 52.78 per cent.; ash, 5.82 per cent.; sulfur, 2.88 per cent.

The Upper Freeport coal has excellent thickness in the southwestern part of Carroll County but the best of the field is now exhausted. But little coal of value is found on this horizon in Tuscarawas County. The most important field, in Ohio, is the Cambridge, Guernsey Co., where the bed has been extensively mined for many years. Large quantities of coal, however, are yet available. The structure follows:

| | | Feet | Inches |
|---|---|---|---|
| Shale | | 3 | 0 |
| Coal | Upper Freeport | 4 | 1 |
| Shale | Upper Freeport | | 2 |
| Coal | Upper Freeport | 1 | 7 |
| Clay | | 3 | 0 |

The quality of the coal is shown by the analysis: moisture, 4.11 per cent.; volatile matter, 36.50 per cent.; fixed carbon, 54.39 per cent.; ash, 5.00 per cent.; sulfur, 1.30 per cent.

This coal has been successfully used in the byproduct oven but the structure of the coke is not up to the standard. By mixing it with other coals, more satisfactory results should be obtained. When coked alone, the yield is about 60 per cent. and the sulfur not above 1.25, which may be considerably lowered by careful selection and preparation. This field has possibilities for byproduct coke plants that are worthy of investigation.

The Upper Freeport coal is also present in Muskingum, Perry, Hocking, Athens, Vinton, Jackson, Gallia, and Lawrence Counties but the areas are usually small and the best of the fuel largely exhausted. The most productive regions are in central Muskingum and northern Athens Counties. The thickness of the bed is from 2 to 6 feet.

### *Conemaugh Formation*

The Conemaugh formation in Ohio contains at least six well-defined coal beds, some of which are remarkably persistent. They are everywhere thin, seldom expanding to as much as 3 ft. and usually measuring about 1 ft. (0.3 m.). With few exceptions, these coals are dense and highly volatile and contain a large percentage of sulfur, which is combined largely with the organic matter. On the whole, they offer no possibilities for coke making.

*Monongahela Formation*

The Monongahela formation outcrops in a broad belt extending along the Ohio River from Jefferson County to Lawrence County. The total area both above and below drainage is approximately 3000 sq. mi. and the average thickness more than 200 ft. The formation is of interest chiefly for its coal beds, which in ascending order are Pittsburgh, Pomeroy, Meigs Creek, Uniontown, and Waynesburg.

*Pittsburgh, No. 8, Coal.*—The Pittsburgh coal, lying at the base of the Monongahela formation, outcrops over a wide area in Ohio but the member has workable thickness only in three fields, which are widely separated. In its extension from the great districts in Pennsylvania into eastern Ohio, this coal increases in sulfur and ash, loses in coking and heating qualities, and shrinks in thickness. In importance, however, the Pittsburgh coal ranks second in this state, being surpassed only by the Middle Kittanning member.

The most important field is the Belmont, where the bed has excellent thickness and continuity over nearly the whole area. Although coal has been mined here since 1825, the quantities yet available are sufficient to last for many years. At present this county is the largest producer in the state. The thickness of the bed is from 2 to 6 ft., the typical structure as given below:

| | | Feet | Inches |
|---|---|---|---|
| Shale | | 8 | 0 |
| Coal | Pittsburgh | 1 | 0 |
| Shale | Pittsburgh | | 10 |
| Coal | Pittsburgh | 2 | 3 |
| Shale | Pittsburgh | | ¼ |
| Coal | Pittsburgh | | 2 |
| Shale | Pittsburgh | | 1 |
| Coal | Pittsburgh | 1 | 5 |
| Shale with pyrite | Pittsburgh | | 1½ |
| Coal | Pittsburgh | 1 | 3 |
| Clay | | 3 | 0 |

The structure varies somewhat from place to place but the bed always has a few thin partings, which are not especially troublesome in mining. The coal is bright and solid. The general character is shown by the following analysis: moisture, 3.74 per cent.; volatile matter, 36.94 per cent.; fixed carbon, 50.08 per cent.; ash, 9.24 per cent.; sulfur, 4.29 per cent. Taking the bed as a whole, this analysis shows that both the sulfur and the ash are entirely too high for byproduct coke. Some parts of the deposit are purer than others but even by careful selection and working this coal will make only a low-grade coke. The structure of the product, however, is very good.

In the Federal Creek field, which lies in the southwest corner of

Morgan County, and in the northeast part of Athens County, the Pittsburgh coal is mined for railroad shipment. The bed varies from 3 to 6 ft. in thickness and, besides thin partings, is regularly divided by a structure of clay about 1 ft. (0.3 m ) thick. Attempts were made to coke this coal for the general market but owing to mining troubles and to the poor quality of the product the result was a failure. In this field, the sulfur is seldom below 2.50 per cent. or the ash below 6 per cent.

The third field of Pittsburgh coal lies in southern Gallia County, but the area is small and the coal of poor quality. It is not mined except for local use.

In summarizing the points in reference to the Pittsburgh coal, the main facts are: The quantities of coal available are large; the mining conditions are good; and the coal cokes freely; but the sulfur and ash are too high to yield a suitable coke for metallurgical use.

*Pomeroy, No. 8a, Coal.*—The second coal bed in the Monongahela formation is the Pomeroy, which lies from 25 to 50 ft. (7.6 to 15 m.) above the Pittsburgh coal and about the same distance below the Meigs Creek member. The only field of importance is in Meigs County, where this coal has been mined for shipment by rail and water for many years. The area yet available is not large. The bed contains some shaly coal in the upper part but otherwise is free from partings. The general thickness is between 4 and 5 ft. and the mining conditions are very good. The composition is: moisture, 6.79 per cent.; volatile matter, 34.91 per cent.; fixed carbon, 47.72 per cent.; ash, 10.58 per cent.; sulfur, 2.42 per cent. The minimum in sulfur is about 1.50 per cent. and in ash about 7 per cent. On account of the high ash and sulfur, the poor structure of the coke, and the low yield attained, the Pomeroy coal is outside the field for byproduct work. Moreover, careful selection and thorough washing will scarcely improve the quality sufficiently for satisfactory results.

*Meigs Creek, Sewickley, No. 9, Coal.*—The Meigs Creek coal, which is correlative with the Sewickley of Pennsylvania, lies from 60 to 100 ft. (18 to 30 m.) above the Pittsburgh bed and is of workable thickness in parts of Belmont, Harrison, Monroe, Washington, Noble, Morgan, Muskingum, and Guernsey Counties. This coal is variable in thickness, structure, and composition. It swells from 2 to as much as 6 ft. in height but it averages about 4 ft. The bed is usually divided by one or more partings of bone coal, shale, or clay, and also contains thin, closely spaced, papery layers of shaly material. The average composition, shown by nineteen analyses, is as follows:[8] moisture, 4.11 per cent.; volatile matter, 36.03 per cent.; fixed carbon, 48.26 per cent.; ash, 11.60 per cent.; sulfur, 4.28 per cent. As about half the sulfur is in organic combination and as much of the ash is so uniformly distributed through the coal, the improve-

[8] Ohio Geol. Survey [4] *Bull.* 9, 301.

ment by washing is not large. Although the field is large, the Meigs Creek coal gives no promises for coke production.

*Uniontown, No.* 10, *Coal.*—The Uniontown coal is present over a wide area in southeastern Ohio but is best represented in Belmont and Monroe Counties. The best deposits are from 3 to 5 ft. thick but they are confined to rather small areas. The coal is generally high in ash and sulfur, and therefore not fitted for coke making.

*Waynesburg, No.* 11, *Coal.*—The Waynesburg coal, although found in several counties in southeastern Ohio, is of little importance as the bed is thin and uncertain and the fuel poor in quality. It is mined chiefly by the farmers for domestic purposes

### *Dunkard Series*

Rocks laid down during Dunkard time are found in a narrow belt extending along the Ohio River from Belmont County to Meigs County. The maximum thickness of the series is more than 600 ft. Several coal beds are present but these have little or no value. The Waynesburg "A" and Washington members are best developed and have been mined in places for local supplies of fuel.

## Summary

Owing to the importance of the iron and steel industry in this state and to the number of byproduct ovens well distributed over the area, the needs for local coking coals are very great but are not satisfactorily supplied. The quantity of coal available is sufficient to last for many years, but the general quality is below the standard demanded for making metallurgical coke. The best possibilities lie with the Lower Kittanning, Middle Kittanning, and Upper Freeport members in certain fields in which the impurities are below the average. Refined methods of picking and working will also add to the supply. If the local coals are used alone, the yield of coke is rather low and the structure at best only fair; but if mixed with high-grade coals from Pennsylvania or from West Virginia, the resulting product will be safely within the limits of usefulness. The limitations are such that the many failures to make coke in beehive ovens were not without cause.

# Sulfur in Producer Gas*

BY FREDERICK CRABTREE† AND A. R. POWELL,‡ PITTSBURGH, PA.

(Chicago Meeting, September, 1919)

WHEN Professor Stoek asked for a paper on the above subject, it was too late to prepare by June 1, or near that time, one that would involve any appreciable amount of experimental work or original research; but the suggestion was made that a review of the literature bearing on the subject might be submitted in good time, and that later this could be supplemented by any experimental data obtained during the summer, if experiments were carried out. In accordance with this suggestion, there are submitted the following notes on references found in the literature of recent years bearing on sulfur in gas. It is to be understood that, in general, these notes express the opinions of the authors whose papers are abstracted, not necessarily the opinions of the abstractors.

Very little published information was found bearing directly on producer-gas sulfur, so this has been supplemented, in certain phases of the subject, by references to illuminating gas. Methods of analysis and processes for the elimination of sulfur from the gas are, without much doubt, very closely related in these two products. As in illuminating gas, sulfur is found in producer gas mainly as hydrogen sulfide, mixed with a much smaller and variable quantity of carbon bisulfide.[1] The presence of minute quantities of other organic-sulfur compounds has been intimated, but no definite statement is made concerning these.

The amount of sulfur present in producer gas is variable and depends on several conditions. The primary factor is the quantity of sulfur in the fuel, but the form in which the sulfur exists, the temperature of the producer, the air-steam ratio, and the efficiency of the scrubbing process are all determining factors. When the coal used contains less than 0.5 per cent. of sulfur, the sulfur in the gas will not exceed the quantity found in the average illuminating gas.[2] Sulfur in producer gas may safely run higher than that in coke-oven gas, due to the lower flame temperature

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* Published by permission of Director of U. S. Bureau of Mines.

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[1] C. Tupper: *Iron Age* (1915) **95**, 455.

[2] Dowson and Larter: "Producer Gas," 95. London, 1907. Longmans, Green & Co.

of the former.[3] Tupper[4] claims that about 40 per cent. of the sulfur in coal passes into the gas produced from it. From this he figures that 1 ton of Westmoreland coal analyzing 1 5 per cent. sulfur will produce a gas containing 720 gr. per cu. ft., after passing through the scrubbers. This last figure is evidently an error and 720 gr. per 100 cu. ft. is intended; this agrees fairly closely with the sulfur content of many raw illuminating gases.

Methods of analysis for the producer-gas sulfur would not differ materially from the methods used for illuminating gas, since the same type of compounds are present in both. Analysis for total sulfur is based on the general principle of burning the gas in measured quantity and absorbing and oxidizing the products of combustion to sulfuric acid. This may be accomplished by absorbing the products of combustion in a solution of bromine and potassium carbonate.[5] Another method would be to burn the gas in an atmosphere containing ammonia. The condensed moisture would then be oxidized by bromine water and the sulfate content determined by any one of the several methods possible.[6] This latter method has been in actual use for producer-gas investigation and an apparatus has been devised utilizing the principle.[7]

The sampling of producer gas has been thoroughly discussed in a gas-power session.[8] The usual qualitative test used to detect small quantities of sulfur in gas is by the use of lead acetate paper.[9] The analysis for the hydrogen sulfide content, irrespective of the other sulfur constituents, is most conveniently performed by an iodometric method.[10] Another method depends on passing the gas through standard potassium-iodate solution and titration with thiosulfate.[11] Cadmium acetate sometimes serves as a means for the determination of $H_2S$.[12]

A method has also been devised for the estimation of the carbon bisulfide constituent of commercial gas.[13] The gas is passed successively through KOH and $H_2SO_4$ solutions and the $CS_2$ constituent is dissolved out by means of alcoholic KOH. This is acidified and titrated for the xanthate formed by the interaction between the $CS_2$ and the alcoholic

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[3] Latta: "North American Producer Gas Practice," 395. New York, 1910, D. Van Nostrand Co.

[4] *Loc. cit.*

[5] A. Longi: *Gazetta* (1898) **28,** 1.

[6] C. Jenkins: *Jnl.* Am. Chem. Soc. (1906) **28,** 542.

[7] R. Threlfall: *Jnl.* Soc. Chem. Ind. (1907) **26,** 355.

[8] *Jnl* Am. Soc. Mech. Engrs. (1917) **39,** 40.

[9] W. Dibdin and R. Grimwood: *Analyst* (1902) **27,** 219.

[10] C. Tutwiler: *Jnl.* Am. Chem. Soc. (1901) **23,** 173.

[11] Th. Schumacher and E. Feder: *Zeit. Nahr. Genussm.* (1905) **10,** 649.

[12] A. Muller: *Jahrb. Gasbeleucht.,* **43,** 792; *Jnl.* Soc. Chem. Ind. (1901) **20,** 73.

[13] E. Harding and J. Doran: *Jnl.* Am. Chem. Soc. (1907) **29,** 1480.

KOH by a standard copper acetate solution. Résumés of various commercial methods of sulfur-gas analysis may be found in any standard gas-engineers' handbook.[14]

The source of sulfur in producer gas is, of course, the fuel. Not all of the sulfur in the fuel passes off in the gas but, in general, a high-sulfur coal will give a high-sulfur gas. Tupper[15] claims that about 40 per cent. of the coal sulfur passes into the gas. Coal possessing more than 0.5 per cent. of sulfur content is not advocated for use in producers.[16] A summary of reports from producer gas plants gives the following data[17] as to the maximum allowable amount of sulfur in the fuel: anthracite 1.5 to 1.75 per cent.; bituminous coal, 1.0 to 3.0 per cent.; bituminous coal (double-zone plant), 1.0 to 1.5 per cent. The effects of other factors than the fuel have never been determined.

When producer gas is burned for various industrial purposes, the sulfur burns to $SO_2$ mixed with a small quantity of $SO_3$.[18] With an excess of air, more $SO_3$ is generated.[19] This sulfuric acid is corrosive in its action on metal parts such as boilers, metal linings, and valves. Furthermore, the sulfur absorbed by the overflow water of the producer and the scrubber water makes it difficult to dispose of this water under certain sanitary conditions.[20]

Fernald[21] summarizes some of the troubles reported by plants as due to the sulfur in the producer gas. An anthracite plant claimed clogging of the passages where the velocity of flow changed, also the eating out of sheet-steel parts. A manufacturing plant using bituminous coal complained of the cutting of water-cooled valves, and three other bituminous plants were troubled with unpleasant fumes in the factory. A lignite gas-producer plant claimed that the valve seats became pitted, while another lignite plant was troubled with the corrosion of roof-metal work, guy ropes, etc.

A great deal of controversy has developed over the effect of sulfur in producer gas on gas-engine cylinders. Latta[22] says that the sulfur dioxide and sulfur trioxide both attack the packing of the pistons and the stuffing-boxes of engines. The effect of the sulfur on the cylinders themselves has been the subject of many important lawsuits. The consensus of opinion

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14 See also E. Russell: *Jnl.* Chem. Soc. (1900) **77**, 352

15 *Loc. cit.*

16 Latta: *Loc. cit.*, 169.
Dowson and Larter: *Loc. cit.*, 95.

17 R. Fernald: Operating Details of Gas Producers. U. S Bureau of Mines *Bull.* 109 (1916).

18 U. Collan: *Zeit. Anal. Chem.* (1895) **34**, 148

19 M. Dennstedt and C. Ahrens: *Zeit. Anal. Chem* (1896) **35**, 1.

20 Dowson and Larter: *Loc. cit.*, 260.

21 *Loc. cit.*, 47.

22 *Loc. cit.*, 169.

seems to be that the results have been exaggerated.[23] The engine at the testing plant of the United States Geological Survey has been running on producer gas since the plant was established and shows no injurious effects, although coals running as high as 8.1 per cent. sulfur have been used.[24]

The sulfur in producer gas is a big disadvantage to the steel manufacturer since the open-hearth steel is contaminated by sulfur.[25] For this reason, Grammer advocates the use of blast-furnace gas enriched with water gas or byproduct coke gas in place of the producer gas. The washing of the producer-gas coal is usually not sufficient to overcome this objection. Efforts to remove the sulfur from the steel often result in injury to the steel, so the better plan is to remove the sulfur from the producer gas.[26] It has been said that steel in soaking pits or other heating furnaces absorbs sulfur from the gas used in heating and that possibly much trouble has been encountered in the way of surface sulfur absorption owing to the use of producer gas containing excessive amounts of sulfur.

There are several methods for decreasing the amount of sulfur in producer gas. The proper selection of fuel is one of the most important. The allowable sulfur percentages in coals have already been mentioned, and somewhat the same limits apply to other fuels also. In case the sulfur of a coal runs rather high, it is possible to decrease this by washing processes before feeding the coal into the producer. One thereby decreases the sulfur in the producer gas.[27] A process of crushing and washing the coal is particularly effective where iron pyrite occurs in comparatively large masses in the seams.[28]

The removal of sulfur from producer gas as such is not much more difficult than its removal from illuminating gas. Sulfur is usually removed from illuminating gas by the iron-oxide absorption process. Another method is the so-called Feld process, in which sulfur dioxide is mixed with the gas, thereby forming elementary sulfur by the interaction of the $SO_2$ and the $H_2S$. Another method, especially adapted to the removal of the $CS_2$ constituent, is the Pippig and Trachmann process. Aniline is used to remove the $CS_2$. About six-sevenths of the sulfur of illuminating gas is removed by this process according to the author.[29]

In plants where the iron-oxide method is in use, the carbon bisulfide is not removed by this treatment. It has been found that reheatment of the gas to 1200° F. removes 70 per cent. of this organic sulfur. Nickel

[23] J. Miller: Power Gas and the Gas Producer *Popular Mechanics Mag.*, Chicago, (1910) 95.

[24] Latta, *loc cit.*, 82.

[25] F. Grammer: *Trans.* (1908) **39**, 545.

[26] Tupper, *loc cit.*

[27] Grammer, *loc. cit.*

[28] Latta, *loc. cit.*, 169.

[29] F. Frank: *Jahrb Gasbeleucht.* (1903) **46**, 488; abst., *Jnl.* Soc. Chem. Ind (1903) **22**, 859.

and other catalysts assist this action.[30] Hydrogen sulfide alone may be eliminated from gas by treatment with potassium carbonate. This substance is claimed to be more efficient than iron oxide and may be regenerated by passing air through it.[31] Treatment of the gas-making coal with lime in the retort will decrease the sulfur content of the gas; 2 per cent. of lime has been recommended for this purpose.[32]

The reaction for the absorption of hydrogen sulfide by iron oxide has been given as follows: $Fe_2(OH)_6 + 3H_2S = Fe_2S_3 + 6H_2O$. The formation of any $FeS_2$ is prevented by the presence of traces of ammonia.[33] For best results it has been found that the iron oxide should be well hydrated. The purifying material used may be regenerated by exposure to air[34] or the sulfur may be removed from the iron oxide by extraction with coal tar distillates.[35] Iron oxides from various sources have given good results in the absorption of hydrogen sulfide. Some of those tried out have been bog ores, the residue left after the extraction of alumina from bauxite and the sedimentation from waste water from coal mines.[36]

Of late years the Feld process for recovery of sulfur from coal gas has reached a commercial scale in Germany.[37] In this process ammonium tetrathionate is utilized.

The application of these sulfur removal processes to producer gas has received a certain amount of attention, but not on any very large scale in this country. Bog-iron ore and wood chips used in the same manner as in illuminating gas purification have been suggested.[38] Of course, any such process of purification involves the cooling of the producer gas, which might be a big disadvantage for certain industries. Purifiers were installed in the St. Louis producer-gas testing station of the Bureau of Mines, when it was found that an iron-oxide purifier became saturated with sulfur after a run of 6 or 8 hr. with coals containing not over 2 per cent. sulfur. Since some of the coals used contained as high as 8 per cent., this particular device was considered a failure in this case.[39]

Tupper[40] gives a process now in use in Germany for the purification of producer gas. It consists essentially in filtering the gas through filter boxes built up of successive layers, on racks, of iron oxides. The iron oxides used are various commercial varieties, which have recently dis-

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[30] H. Papst· *Am Gas Light. Jnl.* (1911) **94**, 407.

[31] Petit: *Gas World* (1915) **63**, 374.

[32] J. Patterson: *Gas Lighting* (1912) **116**, 449.

[33] L. Gedel: *Jahr. Gasbeleucht* (1905) **48**, 400.

[34] E. Jones: *Am. Gas Light Jnl.* (1916) **105**, 245

[35] E. Murphy: *Jnl. Gas Lighting* (1916) **136**, 396.

[36] A. Scott: *Gas Jnl.* (1917) **138**, 101.

[37] *Gas Age* (1918) **42**, 198; *Chem. Abs.* (1918) **12**, 2428.

[38] Reinhardt: *Trans.* (1907) **37**, 685.

[39] R. Fernald and P. Smith: Résumé of Producer-gas Investigations. U. S Bureau of Mines *Bull.* 13 (1911).

[40] *Loc. cit*

placed the bog iron formerly used abroad. Regeneration of the oxide consists in exposure to air. The temperature of the gas during the filtering process should be 90° to 100° F. (32° to 38° C.) particularly when the process is started. This process is not materially different from the ordinary illuminating gas purification. Wallace[41] also gives many data on gas purification. The use of well-hydrated iron oxide is much superior to the dehydrated. It is also a big factor in hydrogen-sulfide elimination that the iron oxide be well aerated during recuperation.

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[41] *Loc cit*

# Low-sulfur Coals of Kentucky

BY WILLARD R. JILLSON,* FRANKFORT, KY.

(Chicago Meeting, September, 1919)

WITHIN the last ten years Kentucky has become celebrated for its low-sulfur bituminous coals. Prior to this time, many investigators had discovered the abundance of this coal but the fact was unknown to the general public until the last extension of Kentucky's mountain railroads was completed. The coals of Kentucky are, broadly, separated into two geographic units—the eastern and the western coal fields. On a basis of their sulfur content these coal fields fall into three units, which are indicated on the accompanying map, by numbers, 1, 2, and 3. The area marked 1, the very southeastern part of the state, shows the lowest sulfur, all coals being averaged. In this field the maximum is 1.04 per cent. and the minimum as low as 0.68 per cent.—both averages and not individual analyses, which would of course show much greater variation. Included in this group are the following counties: Lawrence, 0.87 per cent.; Martin, 0.75 per cent.; Johnson, 0.73 per cent.; Magoffin, 0.87 per cent.; Floyd, 0.88 per cent.; Pike, 0.68 per cent.; Knott, 1.04 per cent.; Perry, 0.77 per cent.; Letcher, 0.80 per cent.; Leslie, 0.70 per cent.; Harlan, 0.79 per cent.; Knox, 0.86 per cent.; and Bell, 0.92 per cent. The total average sulfur content for these thirteen counties is 0.82 per cent.

Bordering the restricted low-sulfur field of eastern Kentucky is the No. 2 belt (slightly darker), which is commonly known as the western border of the eastern coal field. Here the maximum sulfur is 2.91 per cent. and the minimum 1.05 per cent. Again these are averages and not individual analyses. The counties in this group are: Greenup, 2.60 per cent.; Boyd, 1.67 per cent.; Carter, 1.07 per cent.; Morgan, 1.40 per cent.; Wolfe, 1.83 per cent.; Lee, 2.93 per cent.; Breathitt, 1.85 per cent.; Owsley, 1.09 per cent.; Jackson, 1.06 per cent.; Rockcastle, 2.30 per cent.; Clay, 1.09 per cent.; Laurel, 1.08 per cent.; Pulaski, 2.24 per cent.; Whitley, 1.05 per cent.; McCreary, 1.50 per cent.; and Wayne. The sulfur percentage average for this area including fifteen important counties is 1.65 per cent.

In the western Kentucky field, the great increase in the percentage of sulfur is at once apparent. Ohio County with 3.78 per cent. is the highest and Hancock with 2.87 per cent. is the lowest. The percentages by counties is as follows: Hancock, 2.87 per cent.; Henderson, 3.22 per cent.; McLean, 3.27 per cent.; Muhlenberg, 3.31 per cent.; Hopkins,

* State Geologist of Kentucky.

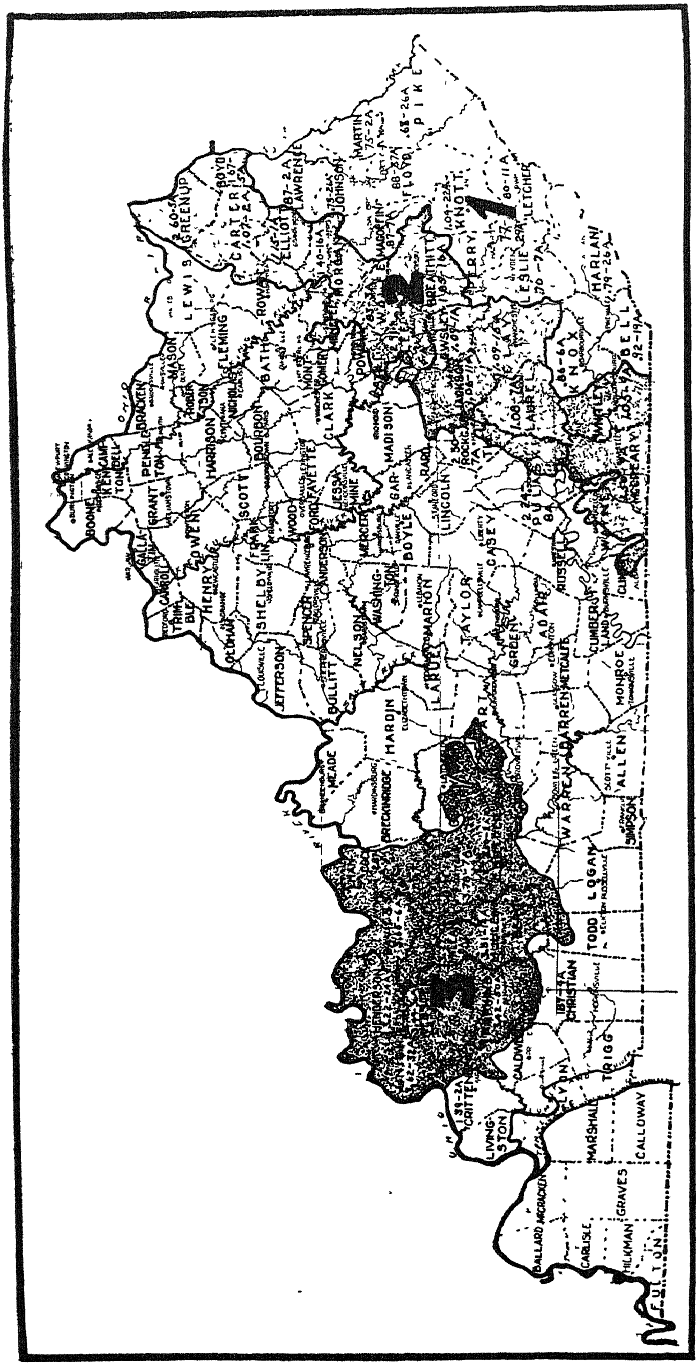

FIG. 1.—SKETCH MAP OF KENTUCKY, SHOWING COAL FIELDS WITH SULFURS.

3.42 per cent.; Webster, 3.49 per cent.; Butler, 3.50 per cent.; Union, 3.62 per cent.; Daviess, 3.65 per cent.; Ohio, 3.78 per cent. The sulfur percentage average for the area, including ten counties and all of the western Kentucky coal field is 3.41 per cent.

In arriving at the above averages upon which the boundaries of the three districts were determined, 514 selected air-dried analyses were used out of a total of about 750, which were reviewed. These analyses were made by two agencies, the U. S. Bureau of Mines and the Kentucky State Chemist. The number of analyses used per county may be found by referring to the map. For example, *Floyd*, 0.88—37*A*, means that the Floyd County percentage of sulfur of 0.88 was secured as an average from 37 analyses.

In the following table of the important low-sulfur coals of eastern Kentucky access was had to over 250 additional analyses of the same character out of which 160 were selected and used in compilation of the data presented below. The following table is not intended to be considered in the light of a definite correlation. Work toward such a correlation is now under way, but final results have not as yet been obtained. The arrangement given, however, is correct within broad lines and will serve the purposes of this paper. Many small coals have been omitted and a number herein given as separate coals will eventually be coördinated. The coals here listed are found only in sulfur districts 1 and 2, as indicated on the map.

Although considerable stratigraphic work has been done on the western Kentucky coals, no attempt is made here to portray the sulfur characteristic of these coals, since the district is without what may be called a commercial low-sulfur coal. It is interesting, from a stratigraphical standpoint, to note the result of the plotting of the sulfurs of the coals of Kentucky, especially since, to date, the Pennyslvanian system in this state has never been mapped, except in outline. Within broad limits, it may be said that sulfur district No. 1 corresponds with the Wise-Kanawha or Upper Pottsville, which forms the main body of the Kentucky commercial coals. Sulfur district No. 2 corresponds roughly with the Norton-New River or Middle Pottsville, and the Lee-Pocahontas or Lower Pottsville. It is relatively of much less importance. Sulfur district No. 3, the western coal field, is probably Middle and Lower Pottsville.

The summation of averages of these sulfur percentages shows quite as much of a difference between districts 2 and 3 as between 1 and 2. It must be concluded, then, that great and striking differences of sedimentation attended not only each division of the Pottsville, but that within any one unit of Pottsville time due to geographic location and proximity of continental or insular masses the same variations in point of size probably existed. It may be significant that lowest sulfured coals of Kentucky are found farthest removed from the principal Potts-

*Tentative Sequence of Important Commercial Coals of Eastern Kentucky*

*Alleghany*

COALS

Stamper, very high, small area
Hilton, very high, small area

*Pottsville*

*Wise—Kanawha—Upper Pottsville*

COALS

Hindman, High Splint, 1.45 (2A—Perry Co.), 0 675 (2A—Letcher Co )
Cornett (Cannel), 0 59 (1A—Letcher Co ).
Francis.
*Flag*, 0.906 (4A—Leslie Co.), 0.80 (5A—Perry Co ), 0.55 (1A—Letcher Co.), 0 75 (1A—Letcher Co ), 0.67 (11A—Letcher Co.).
*Hazard*, 0 80 (5A—Perry Co ), 1 28 (4A—Leslie-Knott Co.), 0.70 (1A—Letcher Co.).
Haddix, 0.816 (10A—Breathitt and Perry Co ), 1.06 (1A—Letcher Co.).
Hamlin, 0.87 (1A—Letcher Co.).
Pardee, 1 14 (1A—Letcher Co.)
Fireclay Rider (McGuire Cannel), 0.69 (1A—Leslie Co.).
*Fireclay—Dean*; *Hyden (No. 4). Lower Lignite*, 0.87 (6A—Perry Co.), 0.72 (10A—Perry Co ), 0.765 (4A—Letcher Co.), 0.413 (3A—Bell Co ).
Whitesburg—Mills (?), 1.318 (1A—Letcher Co.), 0 79 (1A—Letcher Co ).
*Amburgy*, 0 613 (4A—Leslie Co.), 0.98 (1A—Letcher Co ).
*Elkhorn, Taggart*—Moss (?)—(No. 3), 0 58 (6A—Letcher and Knott Cos ), 0.654 (15A—Letcher Co.), 0.971 (10A—Floyd Co.).
Howard—Ivel—(No. 2).
*Van Lear—Prestonsburg*—(No 1)—*Millers Creek*, 0.593 (10A—Johnson Co.), 0.698 (17A—Floyd Co.), av. at Van Lear, 0.50.
*Straight Creek*, 0.902 (8A—Bell Co.).
Collier.
*Blue Gem*, 0.8766 (1A—Knox Co.).
Bacon Creek (Lower Blue Gem).
Harlan—Shelby Gap—Lily (?), 0 805 (2A—Letcher Co.), 0.776, (12A—Harlan Co.).

*Norton—New River—Middle Pottsville*

*Imboden*, 1.33 (1A—Letcher Co.).
Many unimportant coals.
Dorchester.
Manchester, wide variation between, 0 47 (low) and 2 65 (high) (Clay Co.).

*Lee—Pocahontas—Lower Pottsville*

*Beattyville*, 0 233 (7A—Lee Co.), wide variation between, 1.041 (low) and 3.991 (high) (Lee Co.).
Many unimportant coals.

ville marine sea, the lower Mississippi embayment. Progressing toward this typical marine sea from east to west, the Pottsville sulfurs rise rapidly; and after one crosses the shallow channeled area on and just west of the Cincinnati anticlinal barrier in Central Kentucky, the maximum is reached. Here, then, in the certain westward movement of these Pottsville waters out of the long arm extending northeast into Pennsylvania, may lie the logical solution to the problem of the distribution of the low-sulfur coals of this period.

## Effect of Sulfur in Coal Used in Ceramic Industries

BY C. W. PARMELEE,* B. SC., URBANA, ILL.

(Chicago Meeting, September, 1919)

THE ideal fuel for burning ceramic wares is the one that, among other characteristics, has little or no sulfur. For that reason wood was long considered the most desirable fuel but its high cost has practically eliminated its use except in regions where it is very abundant or other circumstances warrant its choice. The fuels commonly employed are natural and producer gas, oil, and coal. The last named is undoubtedly the most important, not only in this country but abroad. Occasionally anthracite is used but the larger part is bituminous.

The quality of the coal used depends on the kind and the value of the products being manufactured. For some materials, such as common brick, hollow block, etc., the quality of the coal is of less importance than in the manufacture of polychrome terra cotta, pottery, porcelain, etc. It is obvious that in burning clays that contain notable amounts of pyrite, marcasite or gypsum, as is the case with many common clays, the sulfur content of the fuel is of much less importance.

The permissible amount of sulfur in the coal used will depend on the kind of product being manufactured.

"A good sample[1] of coal commonly used in pottery ovens has something like this quantity of sulfur: total sulfur, 1.20 per cent.; sulfur in ash, 0.11 per cent.; volatile sulfur, 1.09 per cent."

This is based on English pottery practice. Correspondence with a number of leading manufacturers in this country has produced the following figures:

*Sanitary Ware.*—(*a*) 1.0 per cent. maximum, (*b*) 0.5 per cent.

*Sewer Pipe.*—(*a*) 1.2 per cent., (*b*) 1.1 per cent. present run, 3.0 per cent. has been used.

*Terra Cotta.*—(*a*) 0.5 per cent. basis of contract, 1.0 per cent. approximate.

*Pottery.*—1.0 per cent. contract, 1.5 per cent. probable content.

*Enameled Brick.*—1.3 per cent. maximum.

(*a*) and (*b*) designate different firms giving figures.

Since a pottery kiln for sanitary ware will require about 14 tons of coal for a burn, the quantity of $SO_2$ that will be generated from a 1-per

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* Professor of Ceramics, University of Illinois.

[1] Mellor· *Trans.* Eng. Cer. Soc. (1907) **6,** 71.

cent. content of sulfur will be considerable. Other kinds of wares will require much greater fuel consumption, depending on the size and the type of the kiln and the weight of the clay products being burned.

The objections to the presence of sulfur in the fuel for clay products may be grouped as follows: The clinkering of fuel in the firebox. The action of the oxides of sulfur in waste-heat driers. The effect of the oxides of sulfur on the clays during burning. The effect of the oxides of sulfur on glazes and colors during burning. The effect of the oxides of sulfur on burned clay products.

### Clinkering of the Ash

Clinkering is so commonly associated with a high sulfur content that it is included here. Owing to the high temperatures developed in the fireboxes of kilns, a badly clinkering coal greatly increases the labor of firing and seriously damages the walls of the firebox owing to the slagging action. A mass of liquid slag settling down on the grates chokes off the air supply, preventing the maintenance of a proper excess of air in the hot gases, thus seriously interfering with the development of acceptably colored ware.

### Action of Sulfur in Waste-heat Driers

The waste heat of burning kilns may be used for drying the cruder products. The chief objection to the application of this process to a larger range of wares is the accumulation of the oxides of sulfur in the moisture on the partly dried wares and the attack of the acid water upon compounds of lime, magnesia, and alkalies present in the clay forming sulfates, which subsequently appear as a white-wash. Moreover, the gases taken up by the moisture condensing in the cooler parts of the drier upon the steel drier cars result in a serious corrosion of the framework.

### Effect of Oxides of Sulfur on Clays During Burning

A difficulty commonly experienced with high-sulfur coal is the development of "kiln white" or a coating of sulfates upon the surface of the ware. These sulfates are formed by the condensation of moisture containing sulfuric acid upon the cold ware during the early stages of the firing. This kiln white is very disfiguring. Its formation may be prevented by a proper method of firing; namely, the passage of a large volume of air through the kiln during the period of the burn when this trouble may be expected to develop.

The color of burned-clay products is undoubtedly affected by the character of the gases with which they come in contact during the burning. The mode of attack at very low temperatures has been mentioned. Investigations carried on under higher temperature conditions show that

clays ignited at red heat in a stream of sulfur trioxide are attacked and that iron present will, in some degree, form a sulfate which, at a higher temperature, decomposes leaving red ferric oxide behind.[2] Hopwood and Jackson[3] state that "experience of firebrick makers has shown that if one side of their kilns be exposed to a fresh breeze the combustion of coal on that side being thereby rendered more complete, the bricks obtained from this part of the kiln are always of a deeper color than the remainder, because in this part of the kiln, the sulfur is not completely oxidized and not available as a decomposing agent for the clay substance."

The same authors[4] explain the red specking of white earthenware as due to the contact of the ware with ferruginous sand that, during the early stages of the firing, is attacked by the acid vapors giving rise to the formation of ferric sulfate. This ferric sulfate is subsequently decomposed with formation of a speck of red ferric oxide on the surface of the ware. Other phenomena of a similar sort are discussed by the authors, all of which are attributable to the action of sulfur trioxide formed from the sulfur of the fuel.

Seger[5] calls attention to the action of sulfur trioxide in the development of a red color in the burning of limey clays which normally give a cream or dirty white color. He showed, by analysis, that the yellow centers of the brick contained 0.61 per cent. $SO_3$, while the red outside coat contained 8 49 per cent. $SO_3$.

## Effect of Oxides of Sulfur upon Glazes and Colors During Burning

*Blistering.*—This is the most common manifestation of the action of sulfur gases. The trouble probably has its origin in the absorption of acid fumes by moisture and the condensation upon the glaze during the early stage of the burn. The sulfuric acid thus deposited attacks the glaze constituents, forming sulfates that later, at a higher temperature, are decomposed and cause the development of the blisters and pimples. The trouble will manifest itself in the parts of the kiln where the circulation of the gases is somewhat retarded. The method of prevention is to maintain a proper circulation of air through the kiln during this period of the burn.

*Spitting Out.*—A piece of ware may have a perfect surface when it is removed from the glaze kiln, but subsequent heating to a much lower temperature causes gas to escape from the glaze, leaving small blebs. The kind of escaping gases and the causes operating to produce this

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[2] Hopwood and Jackson: *Trans.* Eng. Cer. Soc. (1903) **2**, 95.

[3] *Loc. cit.* [4] *Loc. cit.*, 98.

See also Kerl: Abriz der Thonwaarenindustrie, 497. 1871.

[5] "Collected Writings," **1**, 364. Easton, Pa., 1902. Chemical Pub. Co.

phenomenon are not well understood but it is quite probable that sulfur dioxide and sulfur trioxide are the offenders.

These gases may have been absorbed and held in solution or they may be liberated by the dissociation of sulfates in solution. Pelouse[6] has shown that mirror glass may dissolve up to 3 per cent. sodium sulfate and hold it in solution without losing any of its transparency, although there will be a frost-like opalescence upon its surface. A similar phenomenon may be observed with glazes of certain types. If glasses or glazes so saturated are supplied with more silica, the formation of blisters will follow. Glazes, glasses, and slags can absorb considerable quantities of gases which may be liberated again. Moore and Mellor[7] found that a frit (a glass) absorbed from 4.7 to 5.7 c.c. coal gas, which could again be set free, collected, and burned.

Slags may contain a large quantity of gas. Johnson[8] states that it may be seen passing off in large volumes when a ladle of slag is quickly dumped and that it ceases as soon as the slag ceases to flow.

*Darkening of Colors.*—The influence of the oxides of sulfur upon the development of colors is undoubtedly very great. Because of the small sulfur content of wood, it has long been considered a most desirable fuel for porcelains.

Taxile Doat,[9] one of the most accomplished of modern ceramic artists, has said "until the coming of a chemist who will obtain with coal the fresh and brilliant palette which wood affords, I will confine myself to the fuel of which I am about to speak"—namely, wood.

Edwards[10] has shown, experimentally, that sulfur gases may cause the darkening of green colors obtained with chromium. A correspondent writes that trouble is experienced particularly in the production of green and brown shades on enameled brick if sulfur gases are present in abnormal quantities.

*Sulfuring and Feathering of Glazes.*—All burners of clay products using the Seger cones as temperature indicators are familiar with the fact that when these cones are subjected to exposure to sulfur gases under favorable conditions they show either a dulling of the surface or blistering. The blistering is due to the causes already described. The dulling of the surface with the appearance of a feathery crystalline coating is caused, according to Mellor,[11] by the formation of a layer of sulfates. The conditions most favorable for this are an oxidizing atmosphere at a temperature near the congealing point of the glaze and, of course, a sufficient concentration of the oxides of sulfur in the kiln atmosphere. He recom-

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[6] Seger's "Collected Writings," 646 [7] *Trans.* Eng. Cer. Soc. (1909) **7**, 7.

[8] Johnson: "Principles, Operation and Products of the Blast Furnace," 217. New York, 1918. McGraw-Hill.

[9] "Grand Feu Ceramics," 132. [10] *Trans.* Eng. Cer. Soc. (1912) **11**, 175

[11] Mellor: *Trans.* Eng. Cer. Soc. (1907) **6**, 71.

mends, therefore, that coal as free as possible from iron pyrites be used at this stage of the burn.

*Staining of Body.*—The accumulation of a very dilute solution of sulfuric acid upon the ware during the early stage of the firing may lead to the attack and the solution of color oxides used in the decoration of the ware. In that case it sometimes happens that the soluble sulfates thus formed will penetrate the body, reaching the surface of the other side of the piece where the evaporation of the water will cause a deposition of the color and its subsequent fixation by firing. This sometimes results in a complete reproduction of the pattern on the surface opposite to its original location. Sometimes the color solution merely drains to a lower portion of the same surface where it lodges, forming an unsightly blemish.[12]

*Salt Glazed Ware.*—This glaze is developed by the action of the vapors of common salt volatilized in the kiln during the firing of the ware attacking and combining with constituents of the clay to form an insoluble glassy coating. It is developed on such wares as face brick, sewer pipe, conduits, chemical stoneware, and drain tile. These wares are made from such suitable clays as are available, which frequently contain notable quantities of pyrite or marcasite. The fuel used for such a purpose generally cannot be selected with the same care as is necessary in the production of the finer grades of wares. However, some care is used to obtain a coal as low in sulfur as circumstances will warrant.

The trouble most frequently experienced in the development of the salt glaze as influenced by the sulfur is similar to that already mentioned, a dulling of the glaze. Another trouble due to the sulfur gases is the formation of a surface coating on the ware of calcium sulfate, which is generally credited with preventing a proper development of the glaze.

*Effect of Oxides of Sulfur upon Burned-clay Products.*—Information regarding this is very meager. Seger[13] has pointed out the possibility of the destruction of the surfaces of glazed brick by sulfates that had their origin in the action of sulfur gases of the coal used in burning the ware. This destructive action may be deferred for a long period after the manufacture of the brick. However, this may be possible only under rather exceptional conditions of manufacture as described by him. A very important phase of this problem that, as yet, has been inadequately studied, is the action of the sulfur dioxide and trioxide upon refractories such as firebrick and blocks, retorts, etc.

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[12] Idem.

[13] "Collected Writings," 376.

# Sulfur in Coal, Geological Aspects*

BY GEO. H ASHLEY,† WASHINGTON, D. C.

(Chicago Meeting, September, 1919)

THE following paper is intended to be suggestive only, and to open the way for discussion and further observation. Its preparation was requested only two days before the time limit set for the submission of the papers comprising this symposium. Thus no time existed for detailed studies, so it is largely a memorandum of such facts as have come incidentally to the notice of the writer in the study of coal in the field.

The field study of sulfur in coal has received very little special attention or description from coal students. The presence of sulfur concretions is noted in hundreds of descriptions, and certain benches of a coal bed may be described as "sulfury" or the presence and position of sulfur bands may be noted in coal sections in connection with sampling.

It may be accepted that some sulfur is present in all coal. The amount varies greatly in different regions, in different beds, and in the same bed in different areas. The anthracite of Rhode Island comes the nearest of any coal known to the writer of being free from sulfur, several analyses of that coal showing less than 0.1 per cent. Subbituminous coals in Colorado in a few places have close to 0.2 per cent. of sulfur on the dry-coal basis and in other states subbituminous coals approach this lower limit. Bituminous coals seldom run under 0.4 per cent. of sulfur on the dry basis, though in a few places dry bituminous coals in Colorado and Washington have between 0.3 and 0.4 per cent. The better known bituminous coals have almost everywhere over 0.4 per cent. The best of the Pocahontas and New River coals run close to 0.4 per cent., but there is very little commercial coal in Pennsylvania, Ohio, Indiana, Illinois, Kentucky, or the other states, that runs under 0.5 per cent. on the dry basis. From these minimum figures, the amount will range up to 5 or 6 per cent. or higher if visible sulfur is not excluded. It must be remembered that the usual analysis of a coal bed is not intended to give the total proportion of sulfur in the bed but the amount in the coal after the removal of the more obvious and larger sulfur masses.

Professor Parr has already pointed out that, chemically, sulfur occurs in several forms: as pyrite, organic sulfur, sulfates, and possibly as uncombined sulfur. Sulfur in the form of sulfates is commonly the obvious result of the weathering of pyrite and may be found in the coal within

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the zone of weathering and be absent in the same mine where the coal is entirely unweathered. Little is known of the occurrence and nature of the organic compounds of sulfur. For our purpose, then, attention may be confined to the occurrence and origin of the pyrite in the coal.

Pyrite occurs in the coal in many forms. First, it may be noted as pyrite bands parallel to the bedding and extending over large areas with more or less uniform thickness and in approximately the same position in the coal bed. Such bands, serving as binders or partings according as the coal sticks to or separates from them, may be observed running all through mines, and in places appear to have an extent of many miles; so that apparently the same bands may be met with in mines many miles apart. This is particularly true where the band, instead of having a uniform thickness, consists of a succession of lenses lying more or less nearly edge to edge or widely spaced but in the same position in the coal bed. These bands may be from ¼ in. to 1 in. (6 to 25 mm.) thick, if regular, or if in the form of contiguous lenses they may have a thickness of several inches. Such a band of lenses often accompanies a band of bone coal or shale, as in the blue band of the blue-band coal of Illinois. The No. 3 coal of Parke County, Ind., may be cited as an example of a coal carrying thin regular bands of pyrite. A study of a large number of coal sections seems to indicate that such bands are most common near the middle of a coal bed or well to the interior of the coal, and that there is no marked tendency for them to be more abundant in the upper part of the bed than in the lower. As a rule, these thin regular bands, as do all other forms of pyrite, stick to the coal, requiring the use of a rattler to free the pyrite for the recovery of the pyrite or for cleaning the coal.

The second mode of occurrence is in the form of flat lenses occurring irregularly both horizontally and vertically. These lenses may be a fraction of an inch thick and 1 or 2 in. across or may have a thickness of several inches and a breadth of a foot or more. From these sizes they may grade up into sulfur bands, in which the thickness is a very small part of the horizontal extent. These lenses may appear bright and brassy or more commonly are black on the outside and bright or dull gray on the inside. Experience has shown that the dull gray lenses may carry as large a percentage of sulfur as the brassy lenses. Where these lenses are short and thick, the surface may be coated with a shiny blanket of coal, possibly anthracite, as a result of a differential movement of the shrinking coal against the non-shrinking lenses.

In their occurrence in the bed such lenses differ greatly, being abundant in some parts of a mine and rare in other parts, or being abundant in certain benches of the coal and rare or lacking in others. Nor are they constant in the latter respect, being abundant in the lower bench in one area and in some other bench of the same bed in other areas.

In the third type of occurrence, which grades out of the second, the pyrite is found in more or less rounded masses "balls," "nigger-heads," etc. These, like the last, may be irregularly developed through the coal horizontally and vertically but are especially abundant in certain beds at the top or bottom or in certain definite horizons accompanying bone coal or black shale. Such concretions are well known in connection with the No. 5 bed of Illinois and Indiana or the No. 9 bed of western Kentucky, but a similar occurrence is characteristic of many other beds in the central interior coal field. They are particularly characteristic of coals having a black "sheety" shale roof, in which are found pyritized marine fossils. They occur in the black shale, locally not protruding down into the coal, but commonly protruding from the roof downward into the top of the coal. They may be widely scattered in the roof, which in general is smooth, or they may be so close together as to coalesce and give the roof a continuous botryoidal appearance. In some districts these concretions are very large; I have measured them up to 7 ft. (2 m.) in maximum extent and locally have seen them extending down into the coal nearly 4 ft. When broken open, they may prove to be all pyrite (I include marcasite here under the term pyrite), either brassy or dull gray, the latter predominating, or they may prove to be only a crust of pyrite changing to calcite on the inside. When of calcite they may contain brachiopods and other shells. In one district in Pike County, Ind., the concretions consist of a hard, thin shell filled with a soft black mush, or mud, which runs out when the shell is broken with a pick. The larger concretions appear to be mainly lime, but the smaller may be largely pyrite. The faces of the concretions are usually slickened, probably due to their being forced down into the vegetal mass forming the coal by the gradual subsidence of the roof as the vegetal mass is condensed under pressure and wastage. In some areas these concretions loosen readily from the roof, but more commonly they are held tight to the roof or within it and the coal breaks away from them.

The fourth mode of occurrence of pyrite in coal is in the form of fine disseminating particles through the coal. As this form is the subject of a paper by Mr. Thiessen[1] it will not be discussed here.

The fifth mode of occurrence is in the form of joint fillings or as irregular plates filling or lining fractures in the coal. Sulfur in this form is obviously secondary and has been deposited since the coal bed has taken its present form or during the folding of the bed, at which time it was broken and creviced. The sulfur, in this form, may occur as thick plates in widely spaced joint planes so as to be separated by hand without great difficulty; more commonly, it is scattered irregularly all through the coal and, if in large amount, requires that the coal be

[1] This Volume.

crushed and washed in order that it be separated. The thin plates may lie vertically in the joint faces or horizontally between the bedding laminæ, or as is frequently found, in the form of nests of connected plates spreading out in various planes and with the utmost irregularity. Pyrite in this form may be distinguished as "secondary platy."

Time has not sufficed for the writer to make a comprehensive review of the general conditions under which sulfur occurs in greater or less amount but a few relations seem worth noting in passing for their possible value in guiding future studies or in eliciting data already known.

A comparison of the sulfur in the Rhode Island anthracite with that in other coals of the United States suggests that the extreme metamorphism which that anthracite has undergone may have, in a measure, desulfurized it. How this could be brought about without driving off the carbon is not clear. The anthracitization of the white-ash bed of bituminous coal at Madrid, N. Mex., does not appear to have perceptibly changed the percentage of sulfur in the coal. The anthracite of Pennsylvania is commonly low in sulfur but not as low as some of the bituminous coals of the same state.

Second, a comparison of the sulfur in bituminous coals with that in lignites (moisture and ash free) indicates little difference For example, of 64 lignite analyses[2] from North Dakota, Texas, and Montana only 2 showed less than 0 5 per cent. sulfur; 21 showed between 0.5 and 1 per cent. sulfur; 31 between 1 and 2 per cent. sulfur; 8 between 2 and 3 per cent.; while 2 had over 4 per cent. of sulfur. This would indicate that most of the sulfur in bituminous coal was there when the coal was in the lignite stage. Carrying this comparison further, it may be noted, in about 250 analyses of peat in Davis's peat paper[3] (using dry peat and allowing a loss of one-third of the weight of the dry peat in the change to bituminous coal), that bituminous coal derived from such peats without gain of sulfur would contain as follows: 14 per cent. would have less than 0.3 per cent. sulfur, 35 per cent. would have between 0.3 and 0.8 per cent. sulfur, 28 per cent. would have between 0.8 and 1.35 per cent. sulfur, 9 per cent. would have between 1.35 and 2.20 per cent. sulfur, and 15 per cent. would have above 2.20 per cent. of sulfur. Comparing these figures with the average of the bituminous coals of the country, especially if the marine coals of the central fields are excluded, would seem to show that most of the sulfur in the coal was acquired at the time of or during the deposition of the coal. That what has been called "secondary platy pyrite" has come into the coal since its deposition seems highly probable from its manner of occurrence, though such an origin may remain to be proved. Further, it is a fact of observation in many mines where the roof is in part shale and in part sandstone, that the highest

[2] U. S. Bureau of Mines *Bull.* 22. [3] U. S. Bureau of Mines *Bull.* 16.

percentage of sulfur is found in those parts of the mine where the roof is sandstone. In view of the permeability of sandstones, as compared with shales, to moving waters, it would appear probable that at least the excess of sulfur under the sandstone might be due to secondary enrichment, though without more detailed study than the writer has yet made such a relationship could not be asserted. It is probable that a detailed study of the coal in such mines might reveal a difference in character and occurrence of the sulfur that would at once indicate whether such a supposition were correct or not.

Third, David White[4] suggests a relationship between the presence of high sulfur and marine conditions surrounding the laying down of the beds. As bearing on that theory it may be noted: First, that in the analyses of peat quoted above most, though not all, of the high-sulfur peats are salt marsh peats; second, if the analyses of Indiana coals in *Bull* 22 of the Bureau of Mines are separated into groups, one of which consists of the analyses of coals having a gray-shale roof with plant remains and another of the analyses of coals having marine fossils in the roof shales, which are commonly black and sheety and overlain with a marine limestone, it is found that none of the first group of coals has as high as 3 per cent. of sulfur while none of the second group of coals has less than 3 per cent. of sulfur. Again reviewing the analyses of Illinois coals, it may be noted that the analyses of the No. 5 coal are all high in sulfur except those near Harrisburg. The normal roof of the No. 5 coal is a black shale containing marine shells, but near Harrisburg the roof is a sandstone. Likewise, the analyses of the No. 6 bed commonly show high sulfur except in the very limited area of thick coal in Franklin and Williamson Counties. A study of the roof bed of No. 6 shows that, in general, the coal is overlain by a limestone in places lying directly on the bed, in others separated from the coal by a small thickness of black shale, like that over the No. 5 bed. In an area of thick coal in Franklin and Williamson Counties, however, there appears to have been a basin in which the limestone rises 15 ft. or more above the coal and the immediate roof is a plant-bearing gray shale. The coincidence is at least worthy of note and offers a definite line for future study. A similar comparison of the sulfur in the Klondyke and Mingo beds of Mingo Mountain, Tenn., shows that while both beds are low in sulfur, the sulfur is highest in the Klondyke bed, which is immediately overlain by a shale carrying marine fossils, while the Mingo bed is directly overlain by a shale carrying plant remains.

A careful review was made of the roof of the Pittsburgh coal to see if such an explanation would account for the change from the low-sulfur conditions found in the coal in the Georges Creek and Connellsville

[4] U. S. Bureau of Mines *Bull.* 38.

areas at the east, as compared with the high-sulfur conditions found in the same bed in Ohio to the west. Definite evidence was not found. Mr. Campbell calls attention to the fact that, in part of Washington County, the Pittsburgh bed is overlain by an abundance of plants. In general, however, plant remains are very rare over the Pittsburgh bed. Lingula are found in the shale over the Pittsburgh bed in places and might be interpreted as indicating marine conditions, though that is not yet definitely proved. It may be noted, however, that the roof in the eastern part of the field usually consists of shales or sandstone overlain in some districts by a limestone, while farther west, in Ohio, this same limestone is much closer to the coal bed and in places lies immediately on top of the coal bed. This limestone is believed to be a fresh-water limestone, but it may be that changed conditions indicated by the difference in the section of rocks just noted has something to do with the difference in the sulfur contents of the beds. Again, comparing the Pittsburgh bed in the Georges Creek region, which carries a very low percentage of sulfur, with the coals of Allegheny age underlying it, which were at least laid down when there were incursions of the sea, this latter fact may have something to do with the higher percentage of sulfur found in the lower beds at most points.

A comparison of the sulfur in different beds of the same general region does not indicate any definite relationship between age and sulfur content. Lack of time has not allowed the writer to make an extensive study of this phase of the subject but so far as he has gone he has not found that the older coals of a series in a county or state, for example, uniformly carry less sulfur than the younger coals. Thus, in the coal beds of Ohio, taken in descending order, there occurs, according to Mr. Bownocker,[5] the following percentages of sulfur: Meigs Creek coal, 4.28; Pomeroy coal, 2.42; Pittsburgh coal, 3.81; Upper Freeport coal, 2.54; Upper Kittanning, 3.30; Lower Kittanning, 3.26; Clarion, 4.33. Nor does there appear to be any definite relationship between the percentage of sulfur and age of coal beds when comparison is made in the analyses of coals directly overlying one another, as where they are mined from the same shaft. It thus appears that the percentage of sulfur is controlled by the conditions existing during the laying down of each bed.

It is not intended to discuss the origin of pyrite in coal in this paper, as that is mainly a chemical problem, but to call attention to certain factors involved. It has been the general assumption that the sulfur of the pyrite in coal has been derived from sulfates, the presence of which is recognized in both ground waters and sea waters. Bischof estimates that the ash of beechwood has enough sulfuric acid and iron oxide to form about one-fifty-thousandth of the weight of the wood in pyrite.

[5] U. S. Geol. Survey *Prof. Paper* 100-B.

If, however, outside sulfates are brought in there is enough iron in the wood to yield 23 times as much pyrite. Firwood could yield 10 times as much as beechwood, or, roughly, with the outside sulfates, 0.5 per cent. of the weight of the wood. The fuci are estimated to contain nearly 4 per cent. of the dry plants in sulfuric acid. Where pyrite occurs as a regular band in the coal, it would appear to have been deposited simultaneously with the coal, possibly as iron carbonates or bog-iron ore such as are forming in many swamps today, which afterward, by decomposition of calcium or magnesium sulfate, was converted into pyrite with liberation or removal of calcium carbonate. The same would be true, possibly, of definite bands or lenses. The pyrite nodules with calcite cores suggest that, in those places, the nodules were originally of calcite and had been partly pyritized afterward. The abundance of nodules with calcite cores suggests that most or all of the nodules were originally calcareous and deposited by concretionary action. The fact that most of the nodular pyrite in coal is today in the form of flat lenses suggests that the formation of the lens left it at first in a plastic condition, so that in the motion of the peat bed under pressure the nodules lost their normal globular or oval shape and were compressed into more or less very flat lenses. In either of these examples it is obvious that the pyrite is the result of secondary processes, which may, however, have taken place before the bed of peat was buried. In this connection it must be remembered that most coal beds form ready channels for the movement of ground waters so that the possibility of the change taking place at a much later date through the transformation of original calcareous or other deposits must be considered. The type of pyrite deposits described as "secondary platy pyrite" would seem to be obviously the result of infiltration at a date possibly long subsequent to the laying down of the coal bed.

# Distribution of Anthracite*

BY A. S. LEAROYD,† WASHINGTON, D C

(Chicago Meeting, September, 1919)

THE Anthracite Division, Bureau of Distribution, of the United States Fuel Administration, came into existence about Oct. 20, 1917. There had been no definite policy determined upon and the distribution had been carried out by the producers and distributors of this fuel as under normal conditions. The domestic sizes had been shipped from the beginning of the coal year on Apr. 1 in about their usual proportions to the various sections of the anthracite consuming territory of this country, which, in a general way, extended as far west as Nebraska and Montana, and in the south along the Atlantic seaboard to Florida. With the advent of cold weather, about the middle of November, acute shortages commenced to develop all over the northeastern section of the country, due to largely increased demand by the public in general, to enormously increased population in centers where industrial war activity was concentrated, and by the needs of the war industries themselves.

Commencing with the early part of December, this demand was aggravated by extreme low temperatures and heavy snow, and the production decreased for the same reasons. From that date until the change in the weather, about the latter part of February, it was simply a question of getting coal into communities as emergencies occurred, to take care of individual cases. No definite plan of distribution could be followed. The coal mined between Apr. 1 and Nov. 1 had been distributed as mined and therefore only current production was available for distribution.

The Anthracite Division, during this period, consisted of a Director, two assistants, together with an office force of seven or eight clerks and stenographers.

The trying experiences of the winter of 1917–18 had, however, shown a ready willingness on the part of the anthracite operators and distributors to coöperate in every way with the United States Fuel Administration. It was realized that if a definite and comprehensive plan of distribution could be formulated to become effective Apr. 1, 1918, determined on the actual needs of each state and community, the producers

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* Read before Pennsylvania Anthracite Local Section, June, 1919.

† Chief of Anthracite Division, U. S. Fuel Administration.

and distributors would follow this plan and that with the advent of the winter of 1918–19 each community would have its coal and, with proper local distribution, the bitter experience of the former winter could be avoided. The anthracite operators and distributors were called together in a conference at which a brief outline of the general plan was laid before them. They were asked to recommend three persons as members of the Anthracite Committee, which would have charge of the detailed distribution; they recommended Mr. J. B. Dickson, of the firm of Dickson & Eddy, Mr. W. J. Richards, President of the Philadelphia & Reading Coal & Iron Co., and Mr. S. D. Warriner, President of the Lehigh Coal & Navigation Co. These three gentlemen, therefore, were appointed by the United States Fuel Administrator as the Anthracite Committee, with headquarters in Philadelphia.

The Committee gathered from each operator and producer his complete distribution by communities and states for the coal year commencing with Apr. 1, 1916, to serve as a basis for the allotments by states and communities for 1918–19. The 1916 coal year was taken because it was a much more normal year than the year commencing with Apr. 1, 1917. The figures for some 16,000 communities were gathered for 100 per cent. of the production. The tonnages were then entered on cards, one showing the total to the town and backed by one for each producer shipping to that community showing his tonnage, the total of the latter equaling the community tonnage. These were arranged alphabetically, by states, and showed the detailed distribution of practically every ton of domestic anthracite for the basic year. The community totals were drawn off on to typewritten sheets by states, and each state set then showed the amount of domestic anthracite that each community consumed in a normal year when, practically speaking, everybody could get all the coal they needed.

Based on the experience of the winter of 1917–1918, it was realized that, from a transportation standpoint, with the great increase in war industrial activity, it was hardly possible to do more than move sufficient bituminous coal into that section of this country comprising the New England States, eastern New York, eastern Pennsylvania, New Jersey, Maryland, and Delaware to keep the essential war industries in operation, and that, therefore, if the domestic consumer was to have his requirements taken care of, so far as the section mentioned was concerned, it must be done with anthracite. On the other hand, states like those south of the Potomac and Ohio Rivers, and west of the Mississippi, as well as Ohio, Illinois, Indiana, and western Pennsylvania, could, in whole or in part, substitute bituminous coal or other fuels. The largely increased needs of the Government, for cantonments and camps, of the war industries that must use domestic anthracite in their manufacturing processes, and the public utility gas plants, had also to be provided for.

The Government needs were determined, a careful check of the war industries made, and the requirements of the water-gas plants carefully catalogued. These consumers required 1,450,000 gross tons of domestic sizes more than in 1916–17. The production used was the same as in 1917–18, which with the shortage of labor was considered the maximum possible. This was a total of 79,000,000 tons, 68 per cent. domestic sizes, or 53,720,000 gross tons. Deducting 1,450,000 tons, the excess over 1916 for the Government, war industries and gas plants, there remained 52,270,000 for distribution as against shipment for all purposes of domestic sizes during the 1916 coal year of 51,677,460. Included in the latter figure was the consumption of domestic sizes by industries, hotels, apartment houses, schools, municipal buildings, public utilities, and about 2,500,000 tons used by the steam railroads for locomotive and other purposes.

It was realized that a large part of the railroad tonnage and a part, at least, of that used by the other consumers mentioned could, by the substitution of bituminous coal or the steam sizes of anthracite, be conserved for domestic consumption. It was from this conserved tonnage that the liquid tonnage to fill emergencies must come, and which, inasmuch as the total expected production was to be allotted, would provide the factor of safety.

After the most careful study by the Anthracite Committee, the allotments to the states to which anthracite was to be furnished were determined. The factors used in the conclusions reached were those already given. The sheet then stood as follows:

| | Gross Tons |
|---|---|
| Expected production domestic sizes, 1918–19 | 53,720,000 |
| Requirements, U. S. Government, War industries, Gas plants } in excess over 1916 | 1,450,000 |

Allotments, 1918–19:

| | Gross Tons | Per Cent. of 1916–17 | |
|---|---|---|---|
| N. E. States | 10,172,000 | 115 | |
| New York, Pennsylvania, New Jersey, Delaware, Maryland, District of Columbia | 30,857,500 | 111 | |
| Virginia | 102,000 | 40 | |
| Ohio and Indiana | 518,400 | 40 | |
| Illinois | 1,550,600 | 70 | |
| Michigan | 1,200,000 | 75 | |
| Iowa | 145,000 | 40 | |
| Minnesota, Wisconsin, North and South Dakota | 2,380,000 | 88 | |
| Canada | 3,600,000 | 92 | |
| Export, Cuba, Bermuda, Newfoundland | 51,900 | 90 | |
| Steam railroads | 2,500,000 | 100 | |
| Totals | 53,077,400 | | 52,270,000 |

Nebraska, Montana, Missouri, and all states south of the Ohio and Potomac, except Virginia, were cut off entirely. The difference in the allotments to the western states was determined by the availability of other fuels for domestic purposes and by the presence of large cities, where anthracite had been the principal domestic fuel and where a complete turn-over to bituminous coal would have been impossible as against the smaller towns where in the past anthracite had constituted but a small part of the total domestic consumption and where the turn-over could be accomplished without extreme hardship. It will be noted that in order to take care of the east, where anthracite was not only the natural domestic fuel but because of transportation conditions must fill the domestic needs, a reduction to the west of 40 per cent. and a complete elimination of anthracite in a number of western and southern states were necessary.

After the state allotments had been determined, the Federal Fuel Administrator for each state was furnished with a statement showing the shipments of the 1916 coal year by communities and was advised as to the total allotment covering the 1918 coal year. It was for him to determine what changes should be made in the distribution by communities as of 1916; in other words, what communities could get along with less coal than they had in 1916—those whose requirements would not exceed 1916 and those where increased amounts of domestic anthracite were needed, due to largely increased population or other reasons. The various state administrators, after conference with their county and local committees, finally determined with the anthracite committee on the exact tonnage of domestic sizes of anthracite to move into each community in their states to which anthracite was to be furnished as domestic fuel. These figures were furnished the producing or distributing companies upon whom the community ordinarily depended for its anthracite, and these sources of supply forwarded the coal in proportionately increased or decreased quantities as against their 1916 tonnages.

This plan, except in rare cases, meant no change in the normal source of supply for a community. In one, where there was an increase of 25 per cent. over 1916, the shippers who furnished the coal in 1916 simply increased their tonnage to their regular dealers 25 per cent. It did mean, however, a considerable hardship commercially on many producers who were obliged to give up entirely states like Nebraska, Missouri, and the South, where they had taken years to build up a trade, and to ship increased quantities to the east, where, as soon as the war was over, the tonnage would go back to normal. It is notable, however, as showing the coöperation of the industry as a whole with the Fuel Administration, that the plan was adopted and literally followed without a complaint. The local fuel administrators took charge of local distribution in communities. Each consumer signed a card covering his requirements and care-

ful check was kept of amount required, sizes used, amount on hand, and so on.

At this time all producers, distributors, and dealers were ordered not to deliver domestic sizes to any consumer, except practically only for domestic use, without a permit from the United States Fuel Administration. This forced apartment houses, office buildings, stores, and industries, to apply to Washington for a permit. Each one was investigated, and whenever by change of grate, or otherwise, either steam sizes or bituminous coal could be used, the permit to use domestic sizes was denied. Many small industries, apartment houses, office buildings, schools, and hotels had used pea, chestnut, or egg coal for steam purposes. They were obliged to turn to other fuels. Domestic sizes were only given to such industries as brass and cupro-nickel plants, where the industry was essential and where only domestic sizes could be used. About one thousand permits were issued, about nine thousand denied. At the same time negotiations with the steam railroads resulted in a reduction in their requirements from 2,500,000 to 1,000,000 tons.

As showing the increased amount of work that this plan of distribution entailed on the Anthracite Division in Washington, the following figures of total communications received and forwarded by the Anthracite Division, are interesting:

| | |
|---|---|
| January, 1918 | 1,702 |
| April | 4,184 |
| May | 6,228 |
| June | 8,614 |
| July | 11,449 |

The Division increased to a Director, two assistants, and a clerical and stenographic force of eighteen.

The community totals for 1918–19 were entered on the cards in the records of the Anthracite Committee and each producer's card to each community had entered upon it the new tonnage, either increased or decreased, as compared with 1916, which that producer contributed to the community total.

As each month passed, the producing companies reported to the Anthracite Committee the shipments made by them to each community where they had an obligation. The total to each community entered in the records of the Anthracite Committee showed the monthly progress of shipments into each and every community. Where for some reason or other a community ran behind or ahead, the monthly records at once showed it and an order was issued by the Anthracite Committee to the producers shipping to the town, to speed up or retard shipments accordingly.

The general plan involved the movement of coal in approximately twelve monthly installments, modified, of course, by the necessity of

completing the total allotment for movement via the Great Lakes to the Lake Superior Docks, Lake Michigan Docks, and Canadian water ports before the close of November. The plan also contemplated somewhat more than the monthly proportion to New England, particularly for water deliveries. To offset this increased monthly proportion for Lake and New England destinations, somewhat less during the spring and summer than the one-twelfth was shipped into the states nearest the mines. This plan was successfully carried through, and at the close of the lake season, on Dec. 1, the entire northwestern tonnage had been forwarded from the Lake Erie ports. All of the Canadian water destinations had received their allotments, and New England was substantially ahead of the eight-twelfths due on an equal division on that day. With the close of the lake season, the lake tonnage was available for distribution into the territory nearer to the mines.

In order to provide a quick remedy for any emergencies where a small amount of coal was required at once, the state administrators were given the right to call on the Anthracite Committee for shipment to them of emergency coal, which the state administrator could put into any community where the need of a few cars was immediate. To equalize distribution within the community, the state administrator could divert from one dealer, who had perhaps more than his share, to another one who had less; he could also call upon the Anthracite Committee to speed up the movement into communities that, for some reason or other, were not receiving what they should.

The production during October and November was seriously interfered with by the influenza epidemic, which prevailed to an alarming extent in the mining region. On the other hand, the mild weather of November and December cut down what would have been the consumption with normal fall and winter weather, and it was estimated that one about offset the other, so that on Jan. 1. the situation was comparable with that which would have prevailed had production during October and November been normal, and the weather of November and December customary for those two months. The emergency cases before the Anthracite Committee on Jan. 1 were limited to a very small number; the orders issued for the immediate shipment of coal being extremely few and the whole situation in most excellent shape, proving that the plan was basically sound, that the allotments were sufficient, and that the execution had been carried out successfully.

The plan had been carried through as originally outlined, with no change excepting in a few instances where state administrators had increased or decreased the allotments to certain communities. The success in its execution was due to the careful attention given to its details by the Anthracite Committee.

The following figures will show the detail of the distribution into some

few eastern cities where the allotments for 1918–19 were in excess of the actual 1916–17 consumption, including principally manufacturing cities where war activities were centered:

| | Shipments Coal Year 1916–1917 | Allotments Coal Year 1918–1919 | Allotments Apr 1, 1918 Jan 31, 1919 | Shipments Apr 1, 1918 Jan 31, 1919 |
|---|---|---|---|---|
| Boston, Mass. | 1,428,368 | 1,571,608 | 1,309,673 | 1,329,225 |
| Worcester, Mass... | 263,142 | 299,107 | 249,256 | 239,575 |
| Fitchburg, Mass | 63,634 | 62,500 | 52,083 | 50,724 |
| Lawrence, Mass | 84,941 | 102,700 | 85,564 | 84,950 |
| New Bedford, Mass. | 125,290 | 126,800 | 105,638 | 105,544 |
| Providence, R. I.. | 330,501 | 409,029–B | 340,857 | 291,368 |
| New Haven, Conn. | 309,027 | 341,902 | 284,918 | 279,652 |
| Bridgeport, Conn. | 223,636 | 300,000 | 250,000 | 258,817 |
| Hartford, Conn. | 276,202 | 285,000 | 237,642 | 228,320 |
| Greater New York | 7,143,844 | 7,998,385 | 6,665,321 | 5,989,477 |
| Albany, N. Y. | 278,847 | 295,977 | 246,648 | 241,881 |
| Binghamton, N. Y | 121,499 | 139,100 | 115,946 | 114,436 |
| Schenectady, N. Y. | 165,636 | 196,300 | 163,580 | 160,983 |
| Syracuse, N. Y | 322,631 | 380,212 | 316,843 | 310,403 |
| Rochester, N Y. | 514,829 | 586,178 | 488,482 | 422,156 |
| Newark, N. J. | 581,221 | 631,935 | 526,612 | 451,071 |
| Paterson, N. J | 211,044 | 222,000 | 185,030 | 186,045 |
| Trenton, N. J. | 194,739 | 224,000 | 186,663 | 192,454 |
| Philadelphia, Pa | 2,353,649 | 2,800,000 | 2,333,333 | 2,282,586 |
| Chester, Pa. | 72,031 | 98,500 | 82,083 | 78,580 |
| Allentown, Pa | 146,312 | 160,000 | 133,333 | 130,690 |
| Bristol, Pa. | 12,089 | 25,000 | 20,833 | 22,085 |
| Harrisburg, Pa | 122,919 | 163,214 | 136,012 | 116,758 |
| Wilmington, Del | 140,831 | 159,200 | 132,667 | 128,977 |
| Baltimore, Md. | 635,800 | 682,173 | 568,477 | 602,274 |
| Washington, D C . | 470,128 | 604,137 | 503,447 | 458,934 |

The armistice, early in November, cut off almost at once a large portion of the industrial consumption and the mild winter resulted in largely decreased consumption by householders, so that by Feb. 1 the tonnage moved was sufficient to provide for the needs of the entire coal year. During January, the whole situation had so eased up that any community could get what coal it wanted, and where certain cities like Providence, Rochester, and Newark show for ten months less than ten-twelfths of the allotment, it was due to inability to take the coal rather than inability to secure it. The allotment for Providence was undoubtedly excessive, and this, with strict conservation of the domestic sizes, resulted in the situation as shown. At Rochester, after the allotment was made, a considerable tonnage of coke was made available to displace anthracite, which accounts for a large part of the discrepancy in this city. In Newark, by the strictest sort of control, many former

consumers of pea, chestnut, and egg coal were only given bituminous, or the steam sizes of anthracite, making available here for domestic consumers a considerable tonnage that formerly had gone for other purposes.

On Feb. 15, all control over anthracite by the Fuel Administration was withdrawn. As a result of the warm winter and the armistice, the mines went on short time in February and March.

The records of shipment for the complete coal year will provide the the most valuable basis for any future plan of distribution which might become necessary. The figures of 1916–17 are complete, and the figures of the 1918–19 coal year, under the war conditions that prevailed, will show the difference due to such war conditions and will, further, show the accuracy of the community allotments as a whole. If, in the future, necessity should arise for control of the distribution of domestic anthracite, it would be difficult to devise a more complete system than the one described. The experience gained and the records kept will enable the control of distribution in the future to be taken up and successfully carried out on the shortest possible notice.

No control was exercised by the Anthracite Committee over buckwheat, rice, and barley, the three steam sizes of anthracite that are only used to a slight extent for domestic purposes. They had their regular consumers and the tonnage was sufficient, so that the normal movement from the producer to the consumer successfully and satisfactorily took care of the situation.

# Research in the Coal-mining Industry

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(Chicago Meeting, September, 1919)

RESEARCH, primarily, is finding out the truth. Research applied to engineering opens the door to new principles and processes, the application of which benefits mankind in a material way. The engineer believes that investigation instituted with the primary view of making the results directly applicable to the industries is the highest type of research. Coal is essentially the basis of modern industrial civilization; therefore, any considerable research pertaining to coal must link itself to the industry by practical application of the results of the research. In other words, all investigational work connected in any way with coal leads to conclusions that permit a better understanding of the origin, nature of occurrence, composition, better and safer mining, more careful preparation, or greater efficiency in utilization of coal.

Modern engineering research has been fostered essentially by universities and in the laboratories of institutions of learning. Coal problems that lend themselves to solution in the laboratories have been first attacked and are perhaps the farthest toward solution. For example, a great amount of research work has been carried out with the various phases of chemistry connected with coal. On the other hand, the equally important problems connected with methods of mining have received but little attention, primarily because their solution involves the continued use of a great and expensive laboratory, which is none other than the mine itself. As many of the problems already solved have had chemistry as their background, so many of the problems unsolved have physics as their background; there are also unsolved problems that deal in coal economics. Even psychology must be called upon in future coal-mining research as a basis for solving those great problems that deal with human relationships. In general, considerable advance has been made in coal research dealing with chemistry, not so much progress has been made with problems having physics and engineering as a background, while there is a tremendous range of untouched problems that deal essentially with business and psychology.

Research in coal mining is not a new development. In the chronicles of the northern coal trade as published in the *Transactions* of the North

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of England Institute of Mining Engineering,[1] it is recorded that about the year 1740 the mischievous practice of screening coal was introduced at one of the collieries. Even at that time some investigator had found that certain advantages were to be secured in combustion by sizing the particular coal, to the evident discomfort of the chronicler. The historic researches of Clanny, Davy, and Stephenson, leading to the discovery and application of the safety lamp in coal mining early in the 19th century are well known. It is noteworthy that Davy and Stephenson succeeded by the two opposite methods of research. Davy, the educated chemist, by scientific research leading to a definite conclusion, and Stephenson, the mechanic, by sheer native ability as an experimenter and inventor.

After the experiences of the world during the recent war in regard to fuel supply, the importance of coal cannot be overemphasized. It needs no argument therefore that any investigation or research designed to increase knowledge of supply, of more efficient production, of better preparation, or more efficient use of coal is not only legitimate but absolutely necessary if the country is to maintain its lead as a commercial nation. Of all the essentials to modern civilization, coal is the one factor whose use involves absolute destruction without hope of replacement.[2] Ores when mined are in use for many years and generally may be reclaimed partly and reused. The exhaustion of known high-grade bodies only leads to the development of lower grade bodies as necessity arises Nitrate beds, the exhaustion of which was once feared, can now be replaced by the recovery of nitrates from the air. Timber will grow again within a comparatively few years. Even the farm lands of the world may probably be made inexhaustible by better methods of cultivation. While the future will see many beds of coal worked, the exploitation of which is not now commercially profitable, yet a too low-grade coal deposit becomes simply a carbonaceous shale that will not burn and from which it does not seem feasible ever to extract the carbonaceous matter in such state that it may be used as a fuel. The definite supplies of coal have, in a general way, been measured and their intelligent use becomes a necessity.

## Present Status of Coal-Mining Research

Everybody recognizes that the war has brought permanent changes to the industries of every country. Abroad, these changes are leading to a larger recognition of the importance of engineering in world affairs and of the need for engineering advice, also the need of more accurately obtained facts, such as those gained in a well-conducted engineering

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[1] (1865–6) **15**, 205.

[2] Petroleum and natural gas also come in this class of destructible essentials.

research. This country cannot be dependent on foreign nations for processes that can be developed at home.

The most notable example of sytematic study of after-the-war problems is the British Ministry of Reconstruction, which has been in operation over a year and has been organized into several scores of subcommittees, many including engineering research. One of their fifteen main commissions is on coal and power, and is subdivided into two main committees and four subcommittees. The important committee is called the Coal Conservation Committee; its duties are to consider and advise on: (1) What improvements can be effected in the present methods of mining coal with a view to preventing loss of coal in working, and to minimize cost of production. (2) What improvements can be effected in the present methods of using coal for the production of power, light, and heat, and for recovering byproducts, with a view to insuring greatest possible economy in production and the most advantageous use of the coal substance. (3) Whether, with a view to maintaining their industrial and commercial position, it is desirable that any steps be taken in the near future, and if so, what steps, to secure the development of new coal fields or the extension of coal fields already being worked. There are subcommittees on mining, power generation and transmission, carbonization, and geology, that will investigate questions of mining and preparation of the fuel for industrial and commercial purposes and the application of carbonization.

In the Department of Scientific and Industrial Research, out of twenty-one subcommittees, at least four are concerned directly with research in the coal industry as follows: (1) A Fuel Research Board to investigate the nature, preparation, and utilization of fuel of all kinds, both in the laboratory and, when necessary, on an industrial scale. (2) A standing committee on engineering, which specifically includes mining. (3) A Mine-rescue-apparatus Research Committee to inquire into the types of breathing apparatus used in coal mines and, by experiment, to determine the advantages, limitations, and defects of the several types of apparatus, what improvements in them are best, whether it is advisable that the types be standardized, and to collect evidence on these points. (4) An Irish Peat Inquiry Committee to investigate the digging, preparation, and use of peat.

The British Government has placed a fund of a million pounds at the disposal of this research committee to enable it to encourage the industries mentioned to undertake research. It is the intention to combine the powers and resources of the industries with Government aid, and funds allotted to each industry will be expended by a committee appointed by the contributing firms in that industry, the results to be available for the benefit of all the contributing firms. Each firm joining such an organization will receive the following benefits:

1. It will have the right to put technical questions and to have them answered as fully as possible within the scope of the research organization and its allied associations.

2. It will have the right to recommend specific subjects for research and, if the Committee or Board of the research organizations of that industry consider the recommendation of sufficient general interest, the research will be carried out without further cost to the firm making the recommendation, and the results will be available to all firms in the organization.

3. It will have the right to the use of any patents or secret processes resulting from all researches undertaken either without payment for licenses or on only nominal payment, as compared with firms outside the organization.

4. It will have the right to ask for a specific piece of research to be undertaken for its sole benefit at cost price, and, if the governing Committee or Board approve, the research will be undertaken.

Furthermore, a Bureau of Information with a regular service of technical information will be available. It is anticipated that the association will make every possible use of existing facilities and scientific research, and that special research institutes may be established in the future.

In this country, the National Research Council, in its Committee on reconstruction, recommended three fields of research: Materials, men or man power, and ideals or principles. These can all be applied to the problems in the mining industry. Up to the present time, most of the coal researches in this country have had to do with materials, including technology, with the exception of the studies having safety of human life as a basis.

Through Congress and with the coöperation of local agencies, the United States Bureau of Mines during the past 10 years has been able to undertake coal investigations relating to mine accidents, testing fuels and mining appliances, and to educational work for the safety of miners. Considerable progress has been made on certain problems, but new and greater problems are awaiting solution. As in other lines of research, many of the easy problems have been solved, and the future must see this Government Bureau devoting itself to more difficult problems of slower solution and of such magnitude or extent that more restricted and smaller research agencies, as mining colleges and private companies, cannot afford to carry on.

Governmental or private research in coal problems, to be of any real value, must be of direct benefit to operators, miners, and engineers actually engaged in producing and utilizing coal. These men are so absorbed with the tasks of production that they do not have time to make their individual researches and knowledge available for the industry and, frequently, they do not come into contact with research results that might benefit them. In any thought of a research program

in coal for the future, special plans must be made for getting the results to every one in the industry who might be benefited.

## Fields of Research in Coal Mining

Research in coal and coal mining naturally divides itself into several branches as follows.

1. Research as to resources, origin, occurrence, and constitution, the province of the geologist and the chemist.

2. Research in development and exploitation, designed primarily to increase safety and efficiency and the application of more scientific knowledge and greater mechanical skill in getting the coal out of the ground. This is the province of the mining engineer, with the coöperation of the chemist, the physicist, and other engineers attacking the special problems of coal mining.

3. Research in preparation for the market, in the past, has been the field of the mining engineer or, at times, of the mechanical engineer as it involves the use of complex machinery. Probably this field belongs to the metallurgist and mining engineer in the same way that the field of ore dressing has developed into a specialty the needs of which are met by the hydrometallurgist called an "ore-dressing engineer."

4. Research in utilization, a field belonging to the fuel or combustion engineer and generally considered a branch of mechanical engineering. Recent developments in the coking and byproduct industries have brought a number of other technical professions into contact with coal utilization, notably the coke specialist or metallurgist, together with the gas engineer and the industrial chemist.

5. Research in coal economics. Coal mining and the coal trade, through the ever increasing complexity of the industry, are becoming matters of a business that links technology with economics. As a rule, the most successful coal-mining engineers have been those who have combined their technical knowledge with the business of the industry. How wide the field and how vital the need for more knowledge are best illustrated by the relative importance of the U. S. Fuel Administration during the war and the vast number of problems that confronted it.

6. Research in human relations. This field has been designated variously as human relations, industrial relations, social betterment, and welfare work. It should not be confused with any attempts to be paternalistic to the worker. How to put the results of research where they will do the most good is just as truly a problem in this field as is any consideration of labor and capital.

## Research as to the Resources, Origin, Occurrence, and Constitution

Through study of the earth strata of the United States, the geologist has been able to approximate the possible areas underlain with coal, to know in a general way the beds of present and future commercial thickness that occupy these areas, and thus to arrive at a general idea of our total coal resources. However, brought down to any given local area, there is not enough geological knowledge to estimate with any degree of accuracy the coal tonnage to be expected from this area. There are exceptions to this rule, however. In some states the coal beds are of such regularity over large known areas that it is safe to forecast resources by our present knowledge. In many districts, however, detailed private exploration has revealed that the coal formerly thought to extend over a given area in commercial thickness and with regularity has thinned to a width where it is not of commercial importance or else it is contaminated with shale and dirt bands or other impurities, thus rendering it of less value than was formerly thought; or that the coal, although existent over the area really lies in a series of lenses or pockets, thus decreasing greatly the reserves of coal thought to exist there. In some cases the reverse has been true and the opening of mines in a new region has proved the coal to be of greater thickness and value than had been previously thought. The time has come when the local agencies, notably the State Geological Surveys, should be put in a position to make more detailed studies as to the local occurrence, distribution, and character of the coal beds within various states. This should be advocated even to the extent of appropriating money to drill certain doubtful areas. There is no other expenditure of a like amount of money that might add so much to the wealth of a state and do away with the doubt and uncertainty that often forbids the investment of private capital in a coal enterprise.

It is, further, the province of the geologist, in coöperation with the chemist, physicist, microscopist, and the biologist to determine with greater exactness the nature and origin of the coals and the laws and facts governing the occurrence of impurities in the seams. For example, the most deleterious impurity in coal today for many purposes is sulfur. Yet of the origin, distribution, and method of occurrence of sulfur in coal but little is known. These basic facts are essential if progress is to be made in the use of high-sulfur coals in operations now requiring low-sulfur coals, and particularly if the sulfur is to be removed from these coals, either by some process before the coal is utilized or in the coking process or even in the gas itself. Research on the constituents of coal must also be the basis of any new developments in methods of coking,

new uses of byproducts, or in fact any radical improvements in the combustion processes that use coal as a fuel.

Scarcely anything is known about the changes that have led to the present state of the solid hydrocarbons known as coal, or the conditions by which nature has recemented and fixed these hydrocarbons in the hardened mass. These may be questions of temperature, pressure, and solution; physical chemistry as related to the nature of coal. Besides these there is a host of more nearly purely geological problems in connection with the strata and flora associated with the coal beds but which have a more indirect connection with the economic working of the coal.

## Research in Development and Exploitation

Research in the mining of coal is properly a part of the greater general subjects of coal conservation and of safety to life. By coal conservation is meant the intelligent development and extraction of coal to supply the fullest tonnage demanded, and not that kind of conservation which has the idea of retarding development in order to save coal for the future.

### *Development*

There are a number of problems connected with coal-mine development on which the accessible data are inadequate. Some of these refer to the best methods of sinking a shaft under different conditions to the bed, or indeed as to under what particular conditions a shaft rather than a slope should be used to open up the deposit. Questions like the use of proper shaft linings also give an idea of the sort of problems on which more data are needed. There also is the question of the shaft bottom layout, which is often made according to the experience of the engineer in charge and without a full study of local conditions or a knowledge of experience elsewhere, which should influence his decision. Published data at present are inadequate.

### *Exploitation*

Gradually, as a mine is developed, more and more problems are encountered that can be considered more properly under exploitation. In general, the investigation problems in coal mining proper involve three subclasses: those connected with the winning of the coal itself, or efficiency in mining; those referring to safety underground; and those relating to the use of mechanical appliances.

*Efficiency in Mining.*—Is it not a sad commentary on our modern economic business and engineering knowledge that little or no advance has been made in the methods of working coal in the past 50 years?

It is true that districts have often adopted methods new to them, but those methods have had their counterparts in some of the older mines of this country or in Europe. In a few of our coal mines, we extract 90 per cent. of the coal; in others, less than 40 per cent In some, the claimed extraction of 60 or 70 per cent. shrinks under actual measurement of the total coal in a given area to perhaps less than 50 per cent. It is a question for plain discussion whether or not the country as a whole can afford to allow systems of mining to continue that lose beyond recovery this large percentage of the coal left in the seam after mining. There is scarcely a coal seam mined in the United States from which it would not be possible, by a change in systems of mining, to extract at least 90 per cent. of the coal. The operators are not to blame for not adopting new systems that would, in most cases, increase the cost per ton of coal, neither are the miners to blame for preferring to work under systems to which they are accustomed. It is primarily a question of research and experiment on a large scale. The problems, in many cases, are so serious and involve so much experimental work and expense that any coal-mining company might not be justified in doing this with its own funds. There should be started at public expense a demonstration mine in every important district to try out better methods and systems of mining and to solve local difficulties. This would be research of the highest order. It will be many years before all coal is mined by the ultimate method of total extraction. As a preliminary, there should be a logical development into greater use of true panel systems, which would permit of a considerably larger extraction of the coal than at present, together with greater safety. There are also many problems in the present systems. For example, what should be the width, extent, and number of the entries; what is the proper relation of room to pillar width; how can the most coal be won from a given area with a minimum amount of dead and narrow work; how is roof pressure best controlled? Many of these problems go back to the fundamental problem of the depth, pressure, and action of the strata over the coal, or as it is generally called, the question of subsidence. How much and in what way do the various overlying strata subside; what is the influence on the subsidence of different methods and rates of mining; what is the effect on the surface; how can subsidence be controlled; what influence does the angle of the dip of the seam and strata have on subsidence; what is the action when mining coal from more than one seam at a place; and how should mining proceed in relation to pillars in order to preserve the greatest amount of coal?

In mining, how is roof slacking best controlled? Experiments should be made in the use of asphalt, gunite, and other air-resisting coatings. The question of the size of pillars that should be left around shafts and between panels and entries involves not only the weight and pressure

of the strata, but also the crushing strength of materials, tests on which should be made, not in machines but under conditions actually found in practice.

There is room for tremendous advance in the present methods of ventilation. Data are needed showing the advantage of tight stoppings, smooth walls, and passages of large cross-sectional area. A corollary to this is the need of cheap fireproof curtains and brattice cloths. Furthermore, no satisfactory system for humidification of the mine air has been worked out and yet this would be of tremendous value in keeping down the evils of dry coal dust and in reducing the cost of sprinkling and other dust-abatement devices.

One of the great costs in mining is timbering. It is certain that in many cases today it is an actual dollars and cents saving to use some form of timber preservative now on the market, yet little general advance has been made in this work. Detailed research will also show that less timber can be used in mining without increased danger to life.

*Safety in Mining.*—Another great problem in coal mining is safety of human life. Along certain lines of safety in coal mining, much advance has been made. The present statistics show that nearly one-half of the accidents in coal mines are due to fall of roof and coal; this means there is a great and universal problem to be solved. It involves a difficult field for study: first of all statistical investigations, then experimental, and finally working out and putting into effect practical methods and plans of eliminating this hazard.

Through research into the nature and cause of gas and dust explosion, considerable progress has been made in the past few years in decreasing accidents from this cause. Much remains to be done in perfecting apparatus for the better prevention and control of mine explosions and in gaining more knowledge of the actual conditions that control the diffusion of gases in a mine. There should be special studies made of the effect of good and bad ventilation on the health and efficiency of the worker; there is uncertainty as to what is good or bad air in mines. It has been reported that coal mining is a healthy occupation as regards death from tuberculosis and kindred diseases which are prone to affect some metal-mine workers, but that the coal miner is prone to other diseases. Reliable data on these points will be necessary not only to inform the public as to the actual conditions that surround the coal-mine worker and to prepare a way for possible betterment of health and sanitation but they will be of tremendous value if the problems in the coal-mining industry are ever considered from a national viewpoint.

A frequent source of life and property loss in coal mining is the shotfirer's explosion. Although these explosions result in great damage to property the loss of life is generally small, the shot-firers being the only men in the mine. There is thus need for a system of shot firing from

without the mine that will be so simple and effective as to make certain its universal adoption. Even though there are today a number of such systems in use, local conditions may render these systems ineffective.

Much progress has been made in the development of permissible explosives. It is still desired that their cost should be cheapened and that they should operate with the production of a minimum of fines While the development of such explosives may be a chemical problem, its application and practice must be brought out by the mining industry.

The time has come when the many safety methods, conditions, and regulations underground should be standardized so that a man going from one district to another might not be under the necessity of learning anew and because of the increasing importance of compensation laws and insurance.

Coal mining, in spite of tremendous advances in safety and decrease in accident rate must still be classed as a hazardous occupation, and research in safety will always hold a direct reward to the man who may discover and apply his discovery. Every problem in a mine, whether of working or mechanical, has a safety point of view worth detailed investigation.

*Mechanical Appliances.*—In England, probably 90 per cent. of the coal is mined by hand, while in the United States probably less than 10 per cent. is so mined. The advance in America in mechanical and electrical haulage and hoisting facilities has been equally marked. There is need of further knowledge as to the relative safety and efficiency of alternating- and direct-current electricity for underground conditions. While it is possible that individual manufacturing companies have much data on these points, this information has not been easily accessible to the general mining public. Coal-cutting and drilling machines have reached a high stage of development. This being the case, there is need for research in the standardization of the operations connected with their use and the publication of the data. There is still to be developed a coal-cutting machine that will successfully operate in steeply pitching seams. The electric trolley locomotive has largely replaced the mule for main haulage and much progress has been made in installing the storage-battery locomotive in district and room haulage. With underground electric machinery, improvements are demanded that will give greater freedom from the dangers of causing explosions from sparking and from other causes. Some progress has been made in the use of gasoline motors underground. The dangers from exhaust gases and from possible ignition of gas from the exhaust are problems to be worked out entirely.

It seems that the electric safety light is the last word in safe efficient mine lights, yet the end of the development along this line should not be considered as reached. A mine light to be perfect should possess three main features: it should give a maximum amount of light, be without

danger of causing an explosion, and have little weight. A study of present-day lights will show that there is still room for improvement in these directions.

Without giving the subject of breathing apparatus serious study, one is apt to think that it is a specialized apparatus applicable and used only in extreme places. What its possible value is to the mining industry has been emphasized by the British Commission on Scientific and Industrial Research, who, out of only 21 committees to consider research in all commercial industries, have one devoted solely to the development and standardization of mine rescue breathing apparatus.

In knowledge and control of mine fires, there has been progress. In a general way the causes of mine fires, the composition of the gases resulting, and the general dangers that attend their fighting and control are known. Much data are yet to be recorded as to the best and most practical method of their control and extinguishment under different conditions that may arise. Such research must be made both at the mine and in the laboratory.

The most deadly gas in a coal mine is carbon monoxide, the product of incomplete combustion. In spite of many individual opinions to the contrary, this gas gives absolutely no warning of its presence by smell, sight, feel, or taste. It would be of great benefit could research disclose a simple and efficient mechanical means of testing for this gas similar to the method of testing for methane in the miner's safety lamp. A simple gas mask that will absorb and take carbon monoxide out of the air as breathed is also yet to be perfected.

Further mechanical problems awaiting solution underground are those connected with pumping, such as resistance of parts against acid water and non-sparking pump motors, and those having to do with caging or dumping systems at the foot of the shaft by which the coal can be more quickly handled without breakage. The time has come when a representative commission should standardize safety gates in and around mines, safety catches on skips and cages, and in fact every kind of mechanical appliance in which there are elements of danger. Some day research should discover an easy method of testing a hoisting rope for weakened and hidden broken strands.

Some of the deep mines today are designed to hoist up to 8000 tons per 8-hr. shift from a single two-compartment shaft. This must be at the rate of more than 16 tons per minute. In this general problem of large output from a single opening, the mining engineer has progressed farther than the mechanical appliances. In other words, he is able to develop his mine and to mine and transport to the bottom of the shaft this large amount of coal. The neck of the bottle that limits the output is the shaft and the hoisting equipment. This field of research is primarily the province of the mechanical or electrical engineer.

A few years ago it was customary to think of head frames and tipples as being well standardized and yet recent investigation has shown it possible to design more efficient head frames at a less cost. A comparison of photographs of head frames made 20 years ago with ones erected in the past few years will show what progress has been made.

## Preparation of Coal

Coal is black and burns! Up to a few years ago this seemed to be the extent of the information about coal in possession of the general public. Coal was sold as it was gotten out of the mine without regard to preparation, except for a simple bar screening, which took out the fines because they could not be used on the type of grate then in common use in boiler plants. Furthermore, the householder objected to fines because of the extra dust and dirt they made.

In all countries it has been the rule that the first mines worked were those containing the thick coal of highest purity and it has only been after a certain stage in the development that there must be mined thinner seams or those seams containing impurities. Naturally these impurities, especially if visible to the eye, must be removed from the coal before it is placed on the market. The public, within the last few years, has been educated to the fact that different coals are of different nature, that they have different heating values, and that ash and dirt in coals are not only a dead loss in burning, but that they greatly increase the freight without producing any benefit. The recent increase in the number of letters the writer has received on the subject of coal preparation and washing is proof that, in the future, more attention must be paid to this subject. Due to our cleaner seams the coal operators have not had to pay as much attention to the removal of impurities as have the coal producers in the European countries. However, the combustion engineer is finding out the sizes and what form of preparation and ash content best meets his needs in a particular coal and the producer has got to meet these needs. Already, at many coal mines, there have been installed extensive mechanical appliances to economically handle and size the coal. A problem always before the designing engineer is how to do this work with a minimum production of fine sizes or degradation of the more valuable sizes into the smaller sizes.

Coal washing has not received in this country the attention it deserves. Many seams of coal are of a nature to permit their use for byproduct coking and other purposes that bring better prices than the average, were it not for the impurities in the coal, particularly sulfur, but often dirt and shale and sometimes phosphorus. Many coal-washing plants in this country have not been successful. There has been a tendency, first, to adapt European plants to American coals and conditions; second, to attempt to install standardized plants without reference to

the particular nature of each different coal; and, third, to go ahead without competent engineering advice. In the state of aggregation of impurities and their removal, each coal presents a different problem, which should be studied as an individual research before an attempt is made to adapt a washing process to the coal. Moreover, in this country there has been the necessity of making the washing process simple and cheap due to the small difference in value between the coal as mined and the clean coal and to the necessity of close competition with the large supplies of clean coal that have been available. These conditions have changed permanently and in many cases careful study of the washing process would reveal it worth while to install processes even of some detail in flow sheets and designed to rid the coal of impurities. This is particularly true of the removal of sulfur from coking coals and in the removal of impurities from small sizes of coal that must stand a long railroad haul to market.

As a washing and hydrometallurgical problem, the preparation of ores, due to their greater value, has received much more scientific attention than the preparation of coal. With certain limitations as to complication of processes, some of the ore-dressing methods and appliances that have proved successful in their field may be adapted to the preparation of cleaner coal for the market with entire commercial success. This entire question of coal washing opens up a field for research in hydrometallurgy to the mining and mechanical engineer and to the metallurgist, whose possibilities can be measured only by considering that our output of coal has doubled each 10 years, that its value today is greater than any other mineral product, and that the public and special users have begun to appreciate the extra value which is in clean coal.

## Utilization of Coal

The details of investigation of coal as a fuel properly concern the mechanical and combustion engineer and the mining engineer is interested only so much as it affects the character and amount of his output. It is sufficient to call attention to the fact that today there is being utilized from 5 to 15 per cent. of the possible energy in coal; the rest is lost. A few years ago up to 40 per cent. of the coal mined was wasted by rejection in the mine or in the tipple of the smaller sizes. To the men who have discovered grates and accessories and other methods of burning this small coal and turning it from a waste into a valuable product the country owes a great debt. One of the most important discoveries made as a result of our war activities has been the possibility of combining liquid oil fuel with pulverized coal to the more economic utilization of each. These are only suggestions which indicate that the field of usefulness for the investigator in coal utilization is without limit.

A different phase in research on coal utilization has been the investiga-

tions on coke and coal-gas manufacture and the utilization of byproducts. The present interest and advances already made in low-temperature distillation, coking of coals formerly thought non-coking, discovery and use of an almost innumerable variety of byproducts from the coking industry are worthy of mention. The writer is assured, by the industrial chemists and others interested, that the future in the coke byproduct industry is even more promising than was the past.

## Research in Coal Economics

It has been suggested that the coal-mining engineer meets less original and less difficult technical problems as compared with the metal-mining engineer. If this is true, it is also true that the coal-mining engineer in developing into his proper sphere as a coal-mining operator runs into more acute problems in business and economics. Therefore, the coal-mining engineer, working as he does under extremely keen competition, must have grounding and knowledge in the laws and practices that help him save and expend wisely fractions of a cent a ton on certain important operations, and to follow and take advantage of proper railroad rates, the trend of the markets, and the factors that enter into the selling of and the market for his product. Research and greater knowledge in these lines is desirable. It is a well-known fact that until the time of the war many coal operations in this country were being run without knowledge of the actual cost of producing the coal or as to whether or not a real profit was being realized on the operation. The reports now required from these operators by the Federal Trade Commission have been of immense value in standardizing mine accounting and promoting better business practices. This is as truly a piece of research leading to important results as though it were a problem whose principles had been discovered in the chemical laboratory. Again, every problem connected with coal mining has its economic aspect and the research investigator in coal-mining work should bear in mind that any answer to his research must come out in dollars and cents.

Many other mining industries in this country have taken up the practice of publication of costs and through this have been able to effect standardization of work with accompanying lowering of costs. There seem to be various reasons why this has not been done in the coal industry, preferably because of close competition existing. Attention is called to the possibilities of investigation into the various conditions and costs of the items of expense in mining and selling coal, and profiting thereby from each other's successes or failures.

More and more the subject of coal-mine taxation is bringing home to the operator, as well as to the taxing body, how little is known or can be agreed upon as to what constitutes the proper figures for the taxing value of the property, what is operating and what is capital expenditure, how

increase of and depletion of reserves should be handled, under what conditions a mine is developing or under actual commercial production, and what part of the work is dead work necessary to the extraction of the coal, and what part might be capitalized. It was only the other day that the writer heard of a case where an assessor, who was a better accountant than an engineer, wished to capitalize a mine fire and consider it as an asset. Again, the field for research of an engineer with business experience and accounting knowledge is great. A corollary of this is that there is in this country today a great mass of mining laws and regulations whose greater unification is to take place. The study of each law in its application under different conditions and in different places offers the largest field for investigation.

## Human Relations

The study of human relations deals with men and primarily with the miners and employees of any organization. The writer recently saw mining costs which showed that of all of the operating expenses of a certain mine, over 80 per cent. was chargeable directly for wages and salaries. Any study of efficiency or technology designed to decrease cost or increase production must, therefore, recognize and learn about the human factor.

Recently the Coal Industries Commission of Great Britain made a report to Parliament on the industry. The members favored an increased wage, a shorter working day, and the miners' claim for a more efficient organization of their industry. It is significant that in arriving at their conclusions, they made note of an absence of data on which to base adequate recommendations regarding just hours of labor, decrease of output to be expected under changed conditions, and changes in output through the expected use of more mechanical appliances. These questions are susceptible of analysis through data gathered and compiled from the industry as a whole.

Labor turnover, housing conditions, recreation facilities, standardization of work, the employee's interest in the business, rights in arbitration measures, and discipline in the mine are some of the questions that are, or will be, of national importance. The truth about them will be known only through much research.

Acknowledgment is made to Dr. David White, United States Geological Survey, and to Dr. J. J. Rutledge and Mr. J. W. Paul, United States Bureau of Mines, for data used in this paper.

## DISCUSSION

J. J. Rutledge,* McAlester, Okla. (written discussion†).—Research work has often a more immediate and practical application to the in-

* Geologist, U. S. Bureau of Mines. † Received Oct. 6, 1919.

dustries than even the investigators themselves realize, but coal mining has not attracted the investigator, except in so far as the mine-accident prevention work and fuel investigations are concerned. In future, the coal geologist will study the subsurface geology of the various coal seams as the petroleum geologist now studies the oil sands, and from the results of these studies the coal operator will be able to locate his shafts to the best possible advantage and to avoid faults that would interfere with underground work. There would have been many thousands of dollars saved, and perhaps some reputations, if a good stratigrapher had been called into consultation in the southern part of Somerset County, Pa., where a local seam of coal, quite thick, was mistaken for the Pittsburgh seam, which, if ever present at that point, had long since been eroded.

Why will the Quemanhoming coal carry more sulfur without injuring the grate bars than George's Creek Big Vein? Only the investigator can answer and yet the fact that the above is true means a greater price for the thinner coal.

The owner of the land in Oklahoma receives one-fourth of the cotton crop produced by the tenant who farms his land and one-third of the corn crop. In the spring, when cotton is just beginning to grow, frequently the owner will see that the land is kept free from grass if the tenant does not in order that he may not lose his rent; yet owners of coal lands in the same state permit the lessees of their land to wastefully mine only from 40 to 50 per cent. of the coal in the seam and to pay a royalty only on the amount mined while the remainder of the coal is left unmined in the ground and is irretrievably lost. Even the farmer is more business-like than the coal-land owner and mine operator. Moreover, by the adoption of an improved system of mining the coal recovery can be increased from 40 to 50 per cent. to 70 and 80 per cent. and the cost of production reduced 15 to 50 c. per ton.

The coal-mining industry is as much entitled to have demonstration mines, supported at Government expense, as the farming industry is to have the demonstration farms and plats. In this work the Government should be the pioneer, as private capital cannot lead the way on account of financial and labor difficulties. At least two machine longwall mines in the Southwest are recovering all the coal with a production of 90 per cent. lump. If roof-slaking could be prevented during the hot summer months, the saving would be from 15 c. to $1 per ton of coal produced. Will it not pay to investigate and learn what will stop the slaking?

Even those who were, and are still to some extent, prejudiced against the use of permissible explosives and machine mining testify to the reduction in accidents and destruction of property since black powder has been abolished. Research gave the permissible explosive and it was

a profitable investment. The closed light, had it been in use, would have saved many lives and much loss of output and wages and destruction of property. The oil and gas operators gladly use the closed lights developed by research work.

The economics of coal mining should be investigated· Why was the zone system the best regulation that the coal-mining industry ever had? Why should a portion of the best longwall coal field in America, in northern Illinois, almost within sight of the great Chicago market, be practically abandoned and the southern Illinois coal mines, 200 mi. farther south, ship their coal directly past the old longwall district into the Chicago market? How is it that southern Illinois operators can ship coal to Texas and drive therefrom the Oklahoma operators whose mines are only a few hundred miles distant? Research will tell.

Psychology will tell why some operators have no trouble with their miners and others are always bothered by strikes.

But the greatest field for research is in methods of working coal. Thousands of dollars are spent in sinking shafts and equipping mines with the latest and most efficient machinery, but when the coal seam is reached, some foreman or subordinate employee decrees what methods of working shall be employed, prejudice and inexperience holding full sway. When demonstration mines are provided and research work is carried on, the "angle of break" will be as important to the mining engineer as the "angle of repose" is to the civil engineer and the laws governing subsidence will be known to some extent, and coal mining will take rank as a profession.

J. W. Paul, Pittsburgh, Pa.—The feature of this problem that appears to me to command the greatest attention at the present time is the need of greater efficiency in getting coal out of the ground, that is, in getting a larger percentage of recovery. That problem is now being presented to the country very prominently by the miners' organization, which claims that the private operator is wasting the fuel of the country. It is due largely to the neglect of the operator and the employee to coöperate in efforts of greater efficiency in the extraction of the larger percentage of coal.

In many sections there is a great waste of fuel, because the methods used are not applicable to the conditions and only 40, 50, and 60 per cent. of the available coal is being brought to the surface; the remainder is left in such a condition that it will never be recovered. In the more recently developed fields, where the operating concern is properly financed and trained engineers design the layout of the mine, the percentage of recovery is high, in some cases running to 95 per cent. and in a few instances to 98 per cent. It is possible to obtain a greater recovery of coal in other parts of the country if the operator and the miner will coöperate and use

the information available through the technically and practically educated mining engineer. The conditions are largely due to the fact that the coal operators have not fully appreciated the value of the trained mining engineer and the time is now at hand when I believe they are awakening to the fact that they have long neglected this important service.

E. N. ZERN,* Pittsburgh, Pa.—Historical references, in a paper of this kind, are always interesting. Just about 100 years ago, in England, a controversy was raging largely between the followers of three men, each of whom had made a valuable contribution to the method of mine lighting. Without going into details, I believe we can safely conclude that Davy should be given credit for applying the principle of the wire gauze to safety lamps, Clanny credited with the introduction of the globe, and Stephenson credited for the addition of the hood or bonnet.

The author refers to the "mischievous practice of screening coal," introduced in Europe about the year 1740. In this country, cleaning and sizing was first done in the anthracite regions of Pennsylvania, owing to the large amount of rock and dirt mined with the coal. The practice was next adopted in Illinois, in order to compete with coals shipped into the central states from West Virginia.

The matter of standardization is of exceeding importance to the industry. The lack of it has been costly to the operator everywhere. There are at least 24 different track gages in American mines; in many cases several different gages are found in mines operated by one company. This, in turn, requires different gages for locomotives, coal-cutting machines, and mine wagons, and necessitates the tying up of capital in a multiplicity of spare parts. There seems to be no valid reason why more than four to six track gages are needed to take care of all mining conditions encountered. The same lack of standardization is found in the various types of mine wagons in use. The head of one important manufacturing concern states that its shop files contain 2000 drawings of mine cars of different patterns varying in size and dimensions. He also stated that the company has never built identically the same car for any two companies. There appears to be no good reason why in a region such as the Fairmont of West Virginia there should not be from three to six standard designs of mine cars instead of fifty or sixty as now found.

F. W. SPERR, Houghton, Mich.—It has been stated that much higher extractions are attained now than formerly—even as high as 95 per cent. It is also said that no progress has been made in methods of working coal in the last 50 years, but some of us can recall that, in the decade from 1883 to 1893, enormous progress was made in the methods of extracting coal. The extraction was raised from about 30 per cent. to 35 and 40 per cent.,

* Editor, Keystone Cons. Pub. Co.

to 85, 90, 95 or 96 per cent. by 1893, when the big panic came on and coal proprietors went begging for operators to come and mine in any way they pleased if they would only take a lease. I am glad to know that something of the old-time efficiency is being attained again and, as was well stated, it is all a matter of laying out the proposition in the right way before operations begin. It should be merely an engineering problem to determine the physical conditions, and then lay out the field with the premeditated design of getting all the coal out of the ground.

In the first place, let me say by way of caution, be sure when buying the coal rights to buy the surface. For extensive underground mining operations, no matter what the mineral is, you should have the surface rights so that you can do what you please underground regardless of what happens to surface. The only way to do clean mining on an extensive scale is to let the surface fall in. In coal mining, of course, you may refill, but that is an expensive proposition, and even then there will be some subsidence. But I doubt whether it is possible to go into any of the old mines where from 40 to 50 or 60 per cent. of the mineral is left standing in pillars and get out very much of this coal without too much danger to life, without first refilling. But a high percentage of extraction can be accomplished if the proposition is laid out in the right way in the beginning.

I am met with the argument that the miners make it impossible. That I do not understand. It is beyond me to fathom why they should want more money for mining pillar coal than for mining face coal when they can mine it just as easily and just as safely.

H. I. Smith, Sullivan, Ind.—One of the main troubles at the present time is the class of men that we have working in our mines. Formerly, the miners were from Wales, England, or Scotland, where they had been used to regular mining—pick mining. Nowadays, we have men from Southeastern Europe, Italy, and Austria, who never saw a coal mine before they came to this country. They are accustomed to shoveling the coal from the floor and putting it into the cars behind their machines. Also, the unions have put various regulations into effect. Those with regard to room lights, track location in the room, and the number of working places to a man, are of importance in the matter of pulling pillars. They specify that the rooms must be, in some cases, 30 ft. wide and in other cases wider. They specify that the track must be laid in the center of the room, which makes pulling pillars a very expensive proposition as it is necessary to relay the track and the gob piles must be moved if a lot of refuse is thrown back.

The Chairman (Carl Scholz, Chicago, Ill.)—I would like to say, for the benefit of the engineers present, that Illinois University has made a most exhaustive study of coal extraction and those members who

have not received a copy of that bulletin can obtain it by writing to the Illinois University at Urbana. I would suggest that you add it to your library as one of the most convenient reference books you can find.

F. W. Sperr.—When those miners to which I referred obtained an extraction of 89 to 95 or 96 per cent., all that the miners were required to do was to shoot down the coal and shovel it into the cars. It was all machine mining. The miners were from Southern Europe and were called "hunks," as at the present time. Do I understand that the specification as to how the mine should be worked is made by the miners?

H. I. Smith.—Yes, to a certain extent.

F. W. Sperr.—There is a chance for a coming together. I would say that the engineer should make the specifications, which should be the safest in the world, safer than any miners could possibly specify. Is the matter as to whether the tracks are in the center of the room or along the ribbed side, a question of safety or of the distance the men must shovel? If laying the track up on one side is not sufficient, lay it on both sides. Pull out one track when the pillar is being pulled back. It is not the large pillar you want, but a pillar just large enough for you to drive your room up quickly and pull it back quickly. We used to drive 30-ft. rooms and 15-ft. pillars. That was the standard.

J. M. Clark.—In West Virginia we find that the best extraction is obtained by driving comparatively narrow rooms and leaving heavy pillars. Very often when the pillars are too narrow they will crush and the screening will come out; coal is lost by not having large pillars.

F. W. Sperr.—That all depends on the nature of the coal. The width of the rooms and the width of the pillars were always gaged by the nature of the ground.

Chairman Scholz.—When the rooms are driven up, we have pillars 60 ft. square to support the panel. We hope by that large pillar to be able to obtain a larger extraction than we could get on 20-ft. pillars.

T. T. Read, Washington, D. C.—Coal mining is often done under very poor living conditions, not only for the men but for the managers as well. Competitive conditions have served to prevent sufficient money being spent to make coal-mining towns agreeable places in which to live. But it is becoming increasingly necessary to improve these towns because the type of labor that is content to live and work under poor conditions is not likely to be available in large supply in the future. There is really no reason why the coal industry should not be able to provide such things. The consumer should pay enough for the coal so that the man who produces it can have good conditions under which to live and work.

Another part of this personnel problem that is interesting and important at the present day is its mental aspect. Companies have been providing better homes, facilities for recreation, good milk for children, good schools and things of that sort, but have not given sufficient study to the mental attitude of the worker; making the doing of a good job a matter in which he finds pleasure and happiness. That is going to be one research field in coal mining. It is the one on which we know the least and on which a great deal of work must be done in order to bring about the proper understanding and control of the problem.

# Some Factors that Affect the Washability of a Coal*

BY THOMAS FRASER,† AND H. F. YANCEY,‡ URBANA, ILL.

(Chicago Meeting September, 1919)

Because of the present interest in the subject of sulfur in coal and its removal, such information as is available in the coal-washing literature on the various factors that determine the adaptability of a coal for wash-

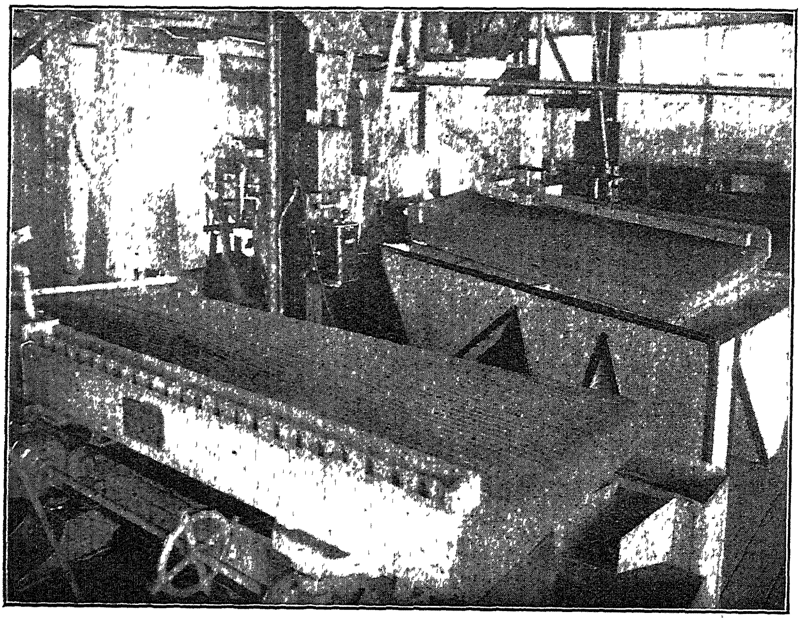

Fig. 1.—Coal-washing tables of one-fourth commercial size.

ing have been collected and are presented here together with some observations based on results of experimental work carried out in the coal-washing laboratory of the mining department of the University of Illinois, and investigative work at Illinois washeries in connection with the work on reduction of the sulfur content of coal, being carried on at the Urbana Station of the U. S. Bureau of Mines. This investigative

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* Published with the permission of the Director, U. S. Bureau of Mines.
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work consisted of the examination and sampling of commercial washeries, the chemical and physical examination of raw and washed coals, and coal-washing tests with jigs and washing tables of one-fourth commercial size. The tables used are shown in Fig. 1. The writers have not had a wide experience in commercial coal washing.

The process of washing a coal consists in the mechanical removal of impure pieces heavier than the pieces of clean coal, and higher in ash and sulfur content. The characteristics of a coal, therefore, that determine the possibility of improving it by washing, are the characteristics of these impurities, that is, their physical and chemical form and their specific gravity and behavior in water.

## Physical Form of Impurities

The two removable impurities commonly present in coal which are of chief interest in connection with coal washing are shale or slate, and pyrite. Calcite and gypsum are ordinarily present only in insignificant amounts, although in some cases their removal may result in an appreciable improvement.

The reduction in ash content of a coal by the removal of shale or slate is commonly less difficult than the removal of sulfur. In order of their removability, the various forms in which the ash in a coal occurs may be classed as follows:

1. Clean solid pieces of shale or slate ¼ in or more thick with a natural plane of breakage between shale and coal. This class of impurity includes the definite shale or slate bands occurring in the coal bed, and extraneous matter from the roof and floor introduced during mining.

2. Bands of clean shale "frozen" to the adjacent coal, and thin bands of shale and coal interbedded.

3. Clay, shale dust, and friable shales that disintegrate in water.

4. Very fine ash particles distributed through the coal, forming carbonaceous shale, bone, and high ash coal. If ash occurs in this form to an excessive degree, it makes impossible the production of a low ash coal by washing. Two coals of this nature have been examined. In one case, a washed coal of good appearance free from visible extraneous impurities analyzed 21 per cent. ash, and in another case coal lighter than 1.4 specific gravity analyzed 13 per cent. ash.

Although it is possible to remove some of the ash occurring in any of these forms; a coal to be classed as easily and economically washable should contain most of its excess undesirable ash in the form of separable clean particles of shale or slate of high specific gravity. A very painstaking visual examination will reveal a great deal as to the removability of the ash impurities in a coal. This is true in a lesser degree of the sulfur impurities.

The easily removable form of sulfur is the lens, ball, or band of clean pyrite of ⅛ in. (3 mm.) or more in thickness. The solid fine-grained band breaks with little sliming and, due to its high specific gravity, it is almost completely removable. A somewhat different form, not uncommon in Illinois, is of a porous cellular structure and usually of a bright yellow color. It is much lighter than the solid pyrite and breaks down easily into thin plates and slime, difficult to remove. Specific-gravity determinations on pieces of these two kinds of pyrite from a Franklin County, Ill., coal showed 4.0 for the solid pieces and 2.9 for the porous pieces.

The most troublesome visible forms of sulfur for the washery man are the thin bands, plates, and veinlets of pyrite distributed through some coals; this form is very common in the southern Illinois field. Knife-edge plates sometimes fill all the joint fissures in large lumps of coal, the individual plates being in many cases so thin as to resemble spots of gold paint. When the coal is crushed for washing, a large part of this sulfur adheres to the coal and a considerable part of that which breaks free is slimed or broken into very thin plates, which have a tendency to float when the coal is washed. The thin bands of pyrite sometimes occur interbedded with thin bands of coal, forming pieces part coal and part pyrite that may be removed in their entirety by jigging at a comparatively large size, whereas if the coal were crushed fine in an effort to free the pyrite washing would not be successful. A small percentage of this kind of pyrite gives the appearance of a much larger quantity.

In all the coals used in the investigations, tests have indicated, also, the presence of pyritic sulfur in a very finely divided state. Samples of coal, finer than ⅛ in., carefully hand picked to reject all pieces showing a trace of impurity, contained pyritic sulfur. Analyses of three such samples of a Williamson County, Ill., coal are given in Table 1. It is believed that these samples were crushed to such size as to expose all the natural cleavage faces on which the visible forms of pyrite are commonly found.

TABLE 1.—*Hand-picked Samples of a Williamson County, Ill., Coal*

| Sample No. | Ash, Per Cent. | Pyritic Sulfur, Per Cent | Organic Sulfur, Per Cent. |
|---|---|---|---|
| 1 | 3.58 | 1.27 | 0.79 |
| 2 | 4.82 | 1.06 | 0 79 |
| 3[a] | 4.27 | 0.84 | 0.79 |

[a] Float on solution of 1.3 sp. gr.

The physical form of impurities present not only affects the amenability of a coal to improvement by washing, but determines also the

size to which a coal must be crushed. Now that tables are being commonly used for washing coal, it might be expected that since the coal can be crushed finer for washing, a cleaner coal might be produced by that method. Actual tests with tables have indicated, however, that this is problematical. It is not safe to assume that any coal which can be successfully washed on coarse coal jigs can be cleaned more completely by washing at a finer size on tables. The effect of finer crushing upon the completeness with which impurities may be separated from coal depends upon two opposing tendencies. First, the more finely a raw coal is crushed, the more completely will the particles of impurities be detached from the particles of clean coal. Second, the finer a coal is crushed the more difficult it becomes to separate the pieces of clean refuse from the pieces of clean coal. The proper size for washing the raw coal, therefore, depends on the proper balance of these two tendencies. Thomas J. Drakely,[1] of Wigan, England, in a paper on coal washing expresses the opinion that in British practice, as a general rule, the size at which the most complete separation is secured is 1¼ in. (3.1 cm.). The common American practice would indicate that here the size is considerably smaller. It is difficult to say what is the minimum size of coal that can be improved by washing. The finest coal that has been successfully treated in washing tests at Urbana was an overflow sludge from a washery settling tank. The size is indicated in Table 2 showing the results of the screening tests. By treating this coal on a washing table, the ash content was reduced from 12.8 per cent. to 8.1 per cent. and the sulfur from 1 7 to 1.5 per cent. with a recovery of 93 5 per cent. of the feed as clean coal.

TABLE 2

| Size | Cumulative Percentage, |
|---|---|
| On ⅛ inch | 3.7 |
| On 20 mesh | 17.8 |
| On 80 mesh | 65.6 |
| Through 80 mesh | 34 6 |

## Chemical Forms of Sulfur in Coal

It is well known that sulfur occurs in coal in three forms; namely, as iron pyrite, as organic sulfur in combination with the coal substance, and as calcium sulfate. Until quite recently very little attention has been given to the chemical forms of sulfur in coal. At present the technical men of the iron and steel, coke, and gas-making industries are giving more attention to this subject. In the past it has been customary to consider "combined sulfur" as well as total sulfur, in determining the

[1] Coal Washing: A Scientific Study. *Trans.* Inst. Min. Engrs (1917–18) **54**, 419.

improvement in a coal to be expected due to washing, but no very accurate methods have been available for the determination of "combined sulfur." The amount of combined, or organic, sulfur present in coal is frequently underestimated; it has often been considered as constituting a negligible percentage of the total amount of sulfur present. Powell and Parr[2] have recently published improved methods for determining the chemical forms of sulfur in coal. Their methods have been used in the present work. Table 3 shows the relative amounts of the different forms of sulfur present in five samples of coal from four states.

TABLE 3.—*Forms of Sulfur in Raw Coal. Values Given in Percentage-moisture-free Basis*

| Coal From | Total | Pyritic | Sulfate | Organic |
|---|---|---|---|---|
| Tennessee, White Co... .... | 4 87 | 3 59 | 0 11 | 1 17 |
| Per cent. of total sulfur . . | . | 74 0 | 2 0 | 24 0 |
| Kentucky, Pike Co. .. ..... | 0 46 | 0 13 | . . | 0 33 |
| Per cent. of total sulfur | ... | 28 0 | . | 72 0 |
| Illinois, Williamson Co . . | 1 83 | 1 04 | trace | 0 79 |
| Per cent. of total sulfur . | .. | 57 0 | . . | 43 0 |
| Illinois, Franklin Co. .. ... | 3 51 | 1 84 | . . | 1 67 |
| Per cent. of total sulfur. . . | . .. | 52 0 | .. . | 48 0 |
| Indiana, Green Co... . . . | 1 66 | 0 89 | .. . | 0 77 |
| Per cent. of total sulfur .. | . | 54 0 | . | 46 0 |

The samples given are not necessarily representative of the county or of the local field from which they were obtained. The results are tabulated for the purpose of calling attention to the amounts of the different forms of sulfur present, and especially to point out the high values for organic sulfur. The coals so far examined were all very low in sulfate sulfur. It appears that this is true for most of the coals in the United States, with the exception of some occurring in the extreme West. Coal that has been in storage and has weathered increases in sulfate sulfur, due to the oxidation of pyrite to iron sulfate.

In connection with the work on the forms of sulfur in coal, determinations were made on the original raw coal samples and on some of the washed products. The reason for determining the different forms of sulfur was to note the effect of washing on the forms and the distribution of sulfur in the products. Analyses are given of two lots of coal. Table 4 shows the results for a White County, Tenn., coal and for samples repre-

[2] Forms in Which Sulfur Occurs in Coal. Univ. of Illinois, Eng. Ex. Sta. *Bull.* 111.

senting a day's run at a 1200-ton commercial washery in Williamson County, Ill. The five washed coals given are the various sizes produced at this washery. Sample *A* represents a small sample of the cleanest washed coal taken near the head end of the table during a test run in the laboratory.

TABLE 4.—*Forms of Sulfur in Raw and Washed Coal Products. Values in Percentage-moisture-free Basis*

| Product | Total Sulfur | Pyritic Sulfur | Sulfate Sulfur | Organic Sulfur |
|---|---|---|---|---|
| White County, Tenn.: | | | | |
| Raw coal.... . .. . . | 4 87 | 3 59 | 0.11 | 1 17 |
| Table washed coal . . | 3 02 | 1 84 | Less than 0.01 | 1 18 |
| Jig washed coal ... .. | 3 80 | 2 61 | .. | 1 19 |
| Sample A ... ... . | 2 67 | 1 51 | .. | 1 16 |
| Williamson County, Ill.: | | | | |
| Raw coal .... ... .. | 1 83 | 1.04 | Trace | 0 79 |
| No. 1 washed coal .. . | 1 81 | 1 05 | .. | 0.76 |
| No. 2 washed coal. .. | 1 56 | 0 78 | . . | 0 78 |
| No. 3 washed coal . .. | 1 57 | 0 82 | . | 0 75 |
| No. 4 washed coal... . | 1 57 | 0 81 | . | 0 76 |
| No. 5 washed coal . . | 2 33 | 1 57 | . | 0 76 |

These results indicate that only pyritic and sulfate sulfur are removed by washing. Iron sulfate resulting from the oxidation of pyrite is soluble in the wash water. Gypsum occurring in thin plates is difficult to separate with the refuse. From these considerations it is believed that the sulfate sulfur of the White County, Tenn., raw coal resulted from the oxidation of pyrite, rather than from the presence of calcium sulfate, since it is almost entirely absent in the clean washed coal. The organic sulfur content of the various products obtained by washing the same coal remains fairly constant as was expected. It appears that by washing a coal high in clean shale a concentration of organic sulfur in the washed coal, and a corresponding reduction of organic sulfur in the refuse would occur.

Attention is directed to the constancy of the values for organic sulfur in Table 4 for the same coal. The lot of White County, Tenn., coal weighed 700 lb. and the samples of Williamson County, Ill., coal represented altogether about 1200 tons, yet the organic sulfur content of all the products for the same coal is practically the same. This tends to show that organic sulfur is uniformly distributed in the coal substance, and that it is in chemical combination with the coal. Certainly the organic sulfur content of coal from the same mine is much more uniform than the pyritic and total sulfur values.

## Specific Gravity

A specific gravity analysis showing the percentages of material of different densities in a raw coal crushed to the size at which it is to be washed is of considerable value for determining, in a preliminary way, what one may expect to accomplish by washing. Analyses of this kind are described by Pascal,[3] commenting on the methods of testing used by Messrs. Beers of Liége, Belgium; and by David Hancock,[4] who used analyses of this nature to determine the efficiency of jigs.

The graphs here presented show the analysis of a washable coal before and after washing, and of this washable coal compared with a raw coal

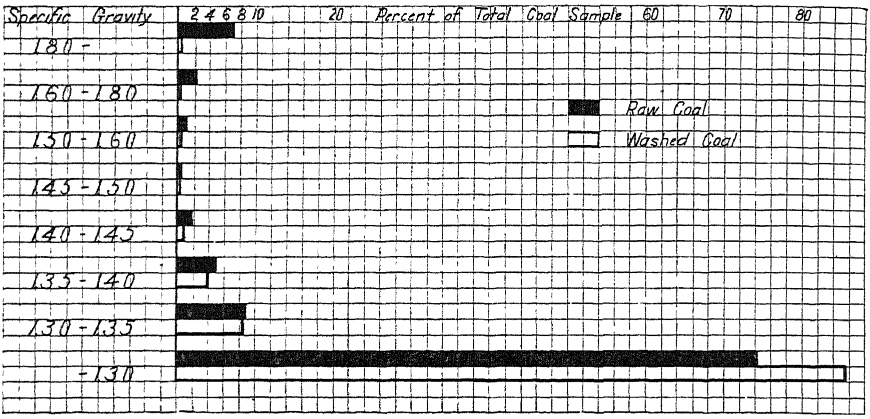


FIG. 2.—COMPARATIVE PERCENTAGES OF MATERIAL OF DIFFERENT DENSITIES IN A COAL BEFORE AND AFTER WASHING AT 0–¼ IN SIZE.

which is difficult to wash. The first graph, Fig. 2, shows very clearly what class of material is removed by washing. While particles heavier than 1.60 in specific gravity were practically all removed, particles between 1.30 and 1.60 in specific gravity are not appreciably affected by washing. The analyses given in Table 5 shows that this material is higher in ash and sulfur than is desirable in the clean coal, but lower than is desirable in the refuse. This represents the class of impurities described under "Physical Form of Impurities" as difficult to remove, and is the product that appears at the washery as "true middling." If the specific gravity analysis of a raw coal shows a large percentage of this material, it is very difficult to wash successfully. This condition is illustrated in the second graph Fig. 3, comparing the raw coal of Fig. 2 with a coal much more difficult to wash. The total percentage between

[3] *Colliery Guard.* (Aug. 10, 1917) **94,** 252

[4] Coal Washing in Alabama. Ala. Geol. Sur. *Bull.*

1.3 and 1.6 specific gravity on the non-washable coal is 40 as compared with only 17 for the washable coal.

TABLE 5.—*Analyses of Coal Represented in Fig. 2*

| Specific Gravity | Raw Coal | | | Washed Coal | | |
|---|---|---|---|---|---|---|
| | Per Cent of Total Sample | Ash, Per Cent | Sulfur, Per Cent | Per Cent of Total Sample | Ash, Per Cent | Sulfur, Per Cent |
| –1 30 | 73 35 | 4 64 | 1 72 | 85 50 | 4 77 | 1 63 |
| 1 30 to 1 35 | 8 74 | 11 27 | 2 14 | 8 30 | 11 8 | 2 06 |
| 1 35 to 1 40 | 4 93 | 17 78 | 2 39 | 3 70 | 17 9 | 2 13 |
| 1 40 to 1 45 | 1 82 | 20 32 | 2 52 | 0 88 | 18 5 | 2 36 |
| 1 45 to 1 50 | 0 39 | 24.60 | 2 62 | 0 27 | 23 6 | 2 55 |
| 1 50 to 1 60 | 1 12 | 29 90 | 2 80 | 0 54 | 28 3 | 2 84 |
| 1 60 to 1 80 | 2 13 | 49 53 | 3 43 | 0 34 | 48 8 | 3 76 |
| 1 80– | 7 52 | 84 04 | 13 63 | 0 57 | 80 3 | 7 07 |

TABLE 6.—*Analyses of Coals Represented in Fig. 3*

| Specific Gravity | Washable Coal | | | Non-washable Coal | | |
|---|---|---|---|---|---|---|
| | Per Cent of Total Sample | Ash, Per Cent | Sulfur, Per Cent | Per Cent of Total Sample | Ash, Per Cent. | Sulfur, Per Cent |
| –1 30 | 73 35 | 4.64 | 1 72 | 55 9 | 10.1 | 2 91 |
| 1.30 to 1 35 | 8 74 | 11.27 | 2 14 | 20 5 | 13 3 | 3 35 |
| 1 35 to 1.40 | 4 93 | 17 78 | 2 39 | 11 8 | 15 4 | 3.45 |
| 1 40 to 1 45 | 1 82 | 20.32 | 2 52 | 3 8 | 19 1 | 4 39 |
| 1 45 to 1 50 | 0 39 | 24.60 | 2 62 | 1 8 | 22 5 | 6 18 |
| 1 50 to 1 60 | 1 12 | 29.90 | 2.80 | 2 1 | 27.6 | 9 29 |
| 1 60 to 1.80 | 2 13 | 49 53 | 3.43 | 1 1 | 42.7 | 13 30 |
| 1.80– | 7.52 | 84.04 | 13 63 | 3 0 | 60 5 | 34 12 |

The ideal coal for washing would be represented by a graph showing all the material concentrated in the parts heavier than 1.60 and lighter than 1.30. Results of a washing test on the coal represented in Fig. 2 are given in Table 7. Table 8 gives the results of a test on the non-washable coal of Fig. 3. The specific gravity determinations on these samples were made by separating the sample at 1.30 specific gravity with the Delameter sink and float machine and treating the sink in a succession of heavier solutions in beakers. The effect of the conditions described on the results attainable by washing is shown by washing tests on some typical coals.

## A WILLIAMSON COUNTY, ILL., COAL

This coal is represented in Fig. 2 and Table 5, and is the washable coal of Fig. 3. The visible impurities consisted of pyrite bands and lenses

as much as 1 in. in thickness; thin shale bands that hold together well in water; some fireclay; thin plates of pyrite in joint fissures; and an unusually large percentage of calcite and gypsum in the form of thin sheets. This coal was washed at 0 to ¼ in. size on a table. The results are given in Table 7.

TABLE 7.—*Washing Test on a Coal from Williamson County, Ill.*

| | Feed, Per Cent | Ash, Per Cent. | Reduction in Ash, Per Cent | Sulfur: Pyritic, Per Cent. | Sulfur: Reduction in Pyritic, Per Cent. | Sulfur: Organic, Per Cent | Sulfur: Total, Sulfur, Per Cent | Sulfur: Reduction in Total Sulfur, Per Cent |
|---|---|---|---|---|---|---|---|---|
| Raw coal ...... | 100 | 14 2 | | 1 94 | | 0 76 | 2 70 | |
| Washed coal ... | 85 | 7 2 | 49 | 1 09 | 44 | 0 76 | 1 85 | 32 |
| Middlings . . . | 6 6 | 19 8 | | 1 80 | | 0 76 | 2 56 | |
| Washed coal and middling combined. ..... | 91 6 | 8.1 | 43 | 1.14 | 41 | 0 76 | 1 90 | 30 |
| Refuse .... | 7 3 | 72.12 | | | | | 10 75 | |
| Loss ......... | 1 1 | | | | | | | |

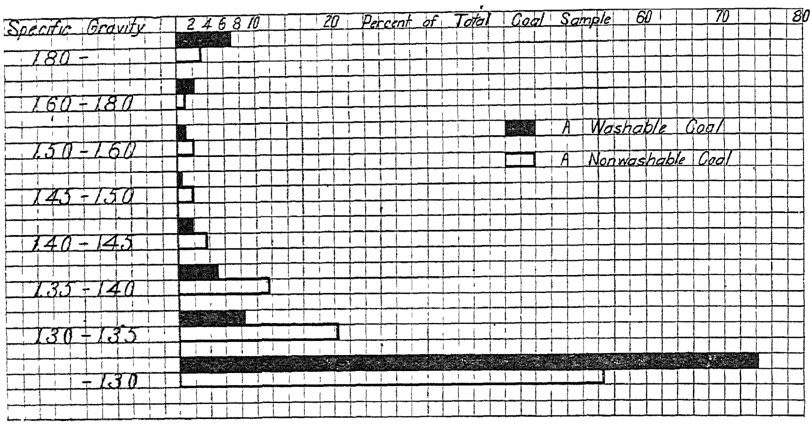


FIG. 3.—COMPARATIVE PERCENTAGES OF MATERIAL OF DIFFERENT DENSITIES IN A WASHABLE COAL AND A NON-WASHABLE COAL, BOTH CRUSHED TO ⅜ IN. MAXIMUM SIZE.

## A COAL FROM WHITE COUNTY, TENN.

This is the coal represented in the graph of Fig. 3 as non-washable. A visual examination showed it to contain very little clean shale or slate coarse enough to be liberated by crushing to the size at which coal is ordinarily jigged. Visible pyrite was present both as thin plates and as coarser bands or lenses. The difficulties in washing this coal were, as indicated by the graph, due to an exceptionally high percentage of material of intermediate density, indicating that the impurities are so fine that even when crushed to ⅜ in. size they are not liberated, and the exceptionally high ash and sulfur content of the lightest coal. The ash

content of the part of this coal that was lighter than 1.30 in specific gravity was 10.10 while the corresponding increment of the washable Illinois coal analyzed 4.64 per cent. ash.

The Tennessee coal was crushed to ⅜ in. maximum size and treated on a washing table. Although a good reduction in sulfur in the clean coal was secured, it was made possible only by taking a very large middling product and a large refuse low in ash and sulfur. For these reasons the washing of this coal would not be profitable.

TABLE 8.—*Washing Test on a Coal from White County, Tenn.*

| | Feed, Per Cent | Ash, Per Cent | Reduction in Ash, Per Cent | Sulfur | | | | |
|---|---|---|---|---|---|---|---|---|
| | | | | Pyritic, Per Cent | Reduction in Pyritic, Per Cent. | Organic, Per Cent | Total, Per Cent | Reduction, in Total Sulfur, Per Cent |
| Raw coal | 100 0 | 15 15 | | 3 60 | | 1 17 | 4 87 | |
| Washed coal | 54 6 | 11 30 | 25 | 1 85 | 51 5 | 1 17 | 3 02 | 38 |
| Middlings | 32 0 | 17 90 | | 4 04 | | 1 17 | 5 22 | |
| Washed coal and middling combined | 86 6 | 13 80 | 9 | 2 65 | 29 00 | 1 17 | 3 82 | 21 5 |
| Refuse | 8 2 | 36 39 | | | | | 17 74 | |
| Loss | 5 2 | | | | | | | |

## SUMMARY

To sum up the conditions that characterize an easily washed coal, the excess undesirable sulfur and ash should be present in form of shale or pyrite particles large enough to be detachable from the coal, without crushing finer than ¼ in. in size. The coal, when crushed to the proper size for washing, should be separable by a sink-and-float test into an increment heavier than 1.6 specific gravity and an increment lower than 1.30 in specific gravity and low in ash and sulfur content with only a small percentage of intermediate density between these increments. The impurities that make a coal difficult to wash are thin bands of friable shale; bony coal; carbonaceous shale; thin filmlike flakes of pyrite, calcite, or gypsum in joint fissures; finely divided impurities intimately mixed with the coal, and organic sulfur.

The chief value, in coal-washing investigations, of the determination of organic sulfur by extraction of the sulfate and the pyritic sulfur, lies in finding a value below which there can be no reduction of sulfur content by mechanical processes. For example, if the coal from a given mine contains 3 per cent. of total, and 1 per cent. of organic sulfur, it would of course be impossible to expect a washed product carrying less than 1 per cent. of sulfur. Although this is a self-evident fact, it is of such importance in determining the washability of a coal that attention is directed

to it. It would be inadvisable to give here a definite figure for the reduction in pyritic sulfur that can be expected with the best modern coal washing machinery. The data given indicate that, in some coals, one-half of the pyritic sulfur may be removed, but the percentage reduction would vary markedly with different coals, depending on the physical form in which the pyritic sulfur occurs. In any case, the minimum sulfur content that may be obtained in the clean coal is well above the organic sulfur content because some pyrite occurs in a very finely divided state intimately mixed with the coal. For practical purposes in coal washing this, in addition to the organic sulfur, may be considered as fixed sulfur.

### Acknowledgment

The experimental work was carried out under the general direction of E. A. Holbrook, Acting Chief Mining Engineer, U. S. Bureau of Mines.

## DISCUSSION

Ernst Prochaska, Bonne Terre, Mo. (written discussion*).—The special importance of this paper is the fact that it calls special attention to the existence of organic sulfur in coal. Many writers have denied the existence of a chemical compound of carbon with sulfur in coal, but later investigations, and especially the paper by Fraser and Yancey, prove undeniably its existence. As we become more fully convinced of its existence and as the present methods of washing, either in a jig or on tables, are unable to separate the organic sulfur, it remains for an inventive genius to perfect a chemical washing process whereby the sulfur will be dissolved out of the coal, or we must abandon entirely the washing of such coal.

Another point that deserves special attention is the description of a systematically carried on investigation into the physical and chemical characteristics of the raw coal previous to the installation of a washery. This method of preliminary investigation was introduced by R. H. DeHoll and later exhaustively carried on by David Hancock, who originated the "efficiency chart," which in my opinion gives a clearer graphical picture of the physical composition of coal than the graphs by Messrs. Beers.

A good many costly washeries have proved total failures, not on account of faulty machinery or methods used but solely on account of lack of preliminary investigation. Such an investigation would have proved the coal to be non-washable and would have saved not only much money but also much disappointment, bad feeling, and reputations. A very simple, inexpensive investigation, preceded by a careful

* Received Sept. 20, 1919.

and thorough sampling of the mine, is the basis not only for the proper laying out of the flow sheet but also for the final decision as to whether or not a washery should be built. Therefore the installation of a physical laboratory should be the first step when considering the building of a coal washery. Coal washing is entering into that period of development where the familiar jig man, with a stick to determine the thickness of the slate bed, must make place for the technically trained washery superintendent assisted by a competent research chemist. Coal washing demands just as much the service of a scientist as do all other metallurgical refining processes.

Carl A. Wendell, New York, N. Y. (written discussion*).—I have never seen in print anything that even remotely has approached this article in clearness and important conclusions in conjunction with work done on the sink-and-float test. I have had many hundreds of these tests made and on different specific gravity solutions and the resultant product analyzed, but I have never grasped nor been able to show what this article sets forth. This article brings out one factor very clearly to my mind; what is to take the place of the solutions of different specific gravities used to obtain the laboratory results, when it comes to washing on a large scale, since water has one constant weight only? The answer is, sizing of coal. And as we know the importance of size versus specific gravity of the material to be washed, perhaps defined (practical) lines can ultimately be gotten up so that the sizing, if any, may be determined at the same time as the laboratory work on the coal is being done.

G. R. Delamater,† Steelton, Pa. (written discussion‡).—The authors' deductions are well founded and although they disclaim any wide experience in commercial coal washing, practically all of the data presented are applicable to coals of all parts of this country. The paper clearly sets forth the great necessity of careful preliminary investigation of each contemplated washery undertaking and contains the following very true statements. "It is not safe to assume that any coal which can be successfully washed on coarse coal jigs can be cleaned more completely by washing at a finer size on tables," and "The effect of finer crushing upon the completeness with which impurities may be separated from coal depends upon two opposing tendencies. First, that the more finely a raw coal is crushed, the more completely will the particles of impurities be detached from the particles of clean coal. Second, the finer a coal is crushed the more difficult it becomes to separate the pieces of clean refuse from the pieces of clean coal."

The use of tables is quite new in coal washing; but because they may be successful in some instances is no proof that they will be in all. Their

* Received Oct. 8, 1919.

† Coke Oven Dept., Bethlehem Steel Co. ‡ Received Oct. 28, 1919.

use should be governed by their adaptability to the various conditions found in any washing proposition. As we near the day when every coal washery proposition will be carefully studied before the plant is designed, coal washing will lose the stigma it has carried so long and its cost will reach a level that will overcome much of the prejudice now existing against it. With the hit-and-miss method of the past, too many plants have failed as first built, because of the necessity of extensive alterations. Had a thorough study been made at first, the plant would have produced with greater efficiency and lower operating cost than is possible with the made-over plant; and in many instances first cost would have been reduced.

The prospective washery owner should not be satisfied with a stereotyped washery layout but should be prepared to spend the necessary money to obtain the following information about his coal:

1. A careful study of conditions within the mine, to be sure that unnecessary expense is not placed against the washery by improper mining practices. Included in this would be an effort to determine the advisability of including such parts of the seam as have not been mined on account of quality. Coal-washing costs may frequently be higher than at other washeries, with reason, if such increased cost results in greater return per acre of that particular coal land.

2. Thorough investigation to determine the proper crushing of the coal and the yield of washed coal that may be expected.

3. Screen tests to determine in what sizes the impurities are carried. This combined with float-and-sink tests and analyses and observation of the resulting products will assist greatly in the determination of:

4. Whether the coal can be cleaned by washing with any degree of success, advisability of sizing for washing, character of machines for washing, advisability of by-passing fines around the washery, suitability to drying by centrifugal driers or other systems, and probable water clarifying equipment necessary.

5. Proper plant arrangement to avoid excessive cost of up-keep and repairs, labor, power, and water consumption and features affecting coal losses and expensive operation delays.

C. A. Meissner, Brooklyn, N. Y.—The question of the organic sulfur in coal is one that has puzzled us quite a little, especially in our Illinois coal, and just how much we can expect those Illinois coals to contain, well, say 3 to 3.5 per cent. in sulfur to be washed down, is dependent to a very large extent on how much organic sulfur there is in them. There is a question in some chemists' minds whether there is much organic sulfur in coal and what this organic sulfur is. The Illinois University, and, I think, the U. S. Bureau of Mines, have been working on it also.

As yet I do not think we have a very clear understanding of what organic sulfur really is, yet we know it is present as sulfur and I think,

when it comes to the blast furnace, whatever may remain in the coke as organic sulfur will ultimately go into the iron or slag and be as deleterious as the pyritic sulfur. On the other hand, Mr. Yancey is quite right, we cannot wash it out; I would like to ask Mr. Yancey whether he is quite positive of the results which he has obtained on organic sulfur; in other words, that there is not sufficient iron present in the coals to take care of the sulfur as pyritic sulfur?

Another question that comes up in connection with washing is the proper sizing of coal. Can you wash your coal without sizing it, say in at least three sizes, or can you take a middle size from ¾ in. down to, say, ⅛ in. and expect to get good results on the washing of that coal? In Europe they size very carefully. Where we have a great deal of fine sulfur, it would seem necessary to get down to pretty close sizing and then do our work either in the jigs or on tables, whichever proves to be the better method.

H. F. Yancey.—Replying to Mr. Meissner's question regarding the determination of the different forms of sulfur in coal; I believe that this is the subject of a paper to be presented by Professor Parr and Doctor Powell. The work which we have done indicates that we approach very closely the true value for organic sulfur. The pyritic iron is extracted along with pyritic sulfur and the determination of iron in this extract serves as a check on both pyritic and organic sulfur. Some iron is present in forms other than pyritic iron, and therefore the calculation of total iron to pyritic sulfur would not be entirely correct. The methods have proved quite satisfactory for the coal-washing work described in our paper.

A. R. Powell.—In regard to the forms of sulfur in coal that come off in coking and what proportion of them stay in coke, I will say there is not a very decided difference in the relative amounts of the forms that come off, although there is a little greater percentage of the organic sulfur than of the inorganic, judging from the majority of the coals studied.

# A Use Classification of Coal*

BY GEO. H. ASHLEY,† WASHINGTON, D C.

(Chicago Meeting, September, 1919)

THE present critical state of the supply, distribution, and utilization of coal and the necessity for pooling and zoning coals calls renewed attention to the lack of any fully adequate classification of coal. In the past coals have been classified as anthracite, semianthracite, semibituminous, bituminous, subbituminous coals, and lignite, and a few special types, such as splint coal and cannel coal. But under the term bituminous coal are included a great variety of coals differing markedly both physically and chemically. The term subbituminous coal covers as great a range of chemical differences. In order to distinguish these varieties of bituminous coals it has been customary to designate coal by names derived from the place where mined, or by the name of the bed from which it is mined. Thus, in the market reports, coals are quoted as Pocahontas coal, New River or Sewell coal, Moshannon coal, Ohio coal, Williamson County coal, Jellico coal, and so on indefinitely, no limit being set as to the boundaries of the area to which a given name is applied. Nor is there any scale by which the coal from one place may be compared with the coal from some other district or some other bed. On the one hand, coals from the same district may be quite distinct and sell at very different prices or coal from the same bed may differ greatly in different districts; on the other hand, coals from different states may be so similar physically and chemically that one could replace the other in practical use without any appreciable difference in service.

The following classification, arrived at after a careful review of a large number of systems of classification and of all of the recognized characteristics of coal, brings together all coals that so nearly resemble each other physically, chemically, and in heating qualities that any coal of a given type could replace any other coal of the same type and grade in use. Thus, if coals *A* and *B* are classed as of the same type and grade they have, so far as known to the writer, approximately the same heating value when burned under the same conditions, and the same character, length of flame, and properties affecting transportation and stocking; and if of the same grade, approximately the same percentage of impurities.

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The classification proposed is intended primarily to be practical, that is of use to the miner, transporter, and user of coal, and only secondarily scientific, that is to bring out resemblances and differences based on resemblances or differences of origin and history of the coal. It is therefore based on obvious physical differences and proximate analyses. This paper is a preliminary statement to bring the matter to the attention of coal men and students of coal, for the purpose of inviting constructive criticism, with the hope that it may then be possible to give the classification fairly permanent form, or as permanent a form as can be done with the present available knowledge. The classification is the result of a comprehensive study of the many thousands of analyses made by the Bureau of Mines, supplemented by many State analyses, combined with studies of the physical properties of the coals of the United States, as made by the writer and others during the past twenty or more years, or as described in tests of coal made by the U. S. Geological Survey, the Bureau of Mines, the Navy, and others.

The available material is still far from complete and the basal studies have not been exhaustive, as such studies will doubtless occupy many men many years in the future, so that the classification cannot lay claim to finality.

The study is based on a series of what are here called "standard" types, which have been arrived at as the result of a long series of pick-and-try tests, and which it has been decided are of sufficient difference to warrant recognition. These types are based on "standard analyses;" that is, an analysis of the coal as "received" or "as fired" but reduced (if necessary) to a standard of impurities. The impurities of the coal, the things not characteristic of the types, are ash, sulfur, and, to a lesser extent, nitrogen. For the standard analysis, there has been selected as a fair average 6 per cent. of ash, 1 per cent. of sulfur, and a percentage of nitrogen varying from 0.75 per cent. in anthracite to 1.5 per cent. in most of the intermediate types, and decreasing to 0.75 per cent. with lignites. A careful study of the moisture of coal has convinced the writer that the moisture content, within certain limits, is a characteristic of the coal. The fact that dried coals of different types subjected to the same conditions of temperature and vapor tension will reabsorb different but characteristic amounts of moisture, as brought out clearly by the experiments of Porter and Ralston,[1] is just one line of evidence of the truth of this. As far as possible, types have been chosen that do not vary greatly in moisture content at the mines (as shown by analyses of mine samples) and at the point of delivery (as shown by large numbers of analyses of coals taken at points of delivery on Government contracts).

[1] H. C. Porter and O. C. Ralston: Some Properties of Water in Coal. U. S Bureau Mines *Tech. Paper* 113.

It may be asked why "air-dried" analyses are not used. A study of the air-drying results shown in Table 1 seems to indicate that, as yet, our air-drying methods have not been standardized, so as to give results consistent with themselves or with the analyses of the coals "as fired," which latter are, after all, the data of most value for the buyer and the engineer.

TABLE 1.—*Air Drying of Low-rank Coals*

| Lignites | | | Sheridan Coals | | | Iowa Coals | | | Rock Springs Coal | | | Indiana Coals | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| As Received | Loss | Air-dried | As Received | Loss | Air-dried | As Received | Loss | Air-dried | As Received | Loss | Air-dried | As Received | Loss | Air-dried |
| 34.5 | 21.0 | 14.5 | 22.7 | 6.6 | 16.1 | 17.1 | 9.4 | 7.7 | 8.5 | 2.3 | 6.2 | 15.3 | 11.3 | 4.0 |
| 35.7 | 23.3 | 12.4 | 21.4 | 6.6 | 14.8 | 16.1 | 8.6 | 7.5 | 9.7 | 2.8 | 6.9 | 15.9 | 10.4 | 5.5 |
| 35.4 | 17.0 | 18.4 | 20.3 | 7.1 | 13.2 | 14.0 | 4.5 | 9.5 | 10.9 | 1.5 | 9.4 | 16.9 | 13.1 | 3.8 |
| 43.7 | 35.3 | 8.4 | 23.2 | 10.0 | 13.2 | 18.5 | 7.1 | 11.4 | 14.4 | 4.2 | 10.2 | 10.9 | 7.0 | 3.9 |
| 29.7 | 22.4 | 7.3 | 24.7 | 15.0 | 9.7 | 14.2 | 10.4 | 3.8 | 14.5 | 4.1 | 10.4 | 12.9 | 7.6 | 5.3 |
| 43.5 | 32.6 | 10.9 | 24.7 | 10.4 | 14.3 | 16.9 | 15.5 | 1.4 | 13.5 | 3.8 | 9.7 | 13.5 | 5.1 | 8.4 |
| 35.9 | 12.7 | 23.2 | 22.8 | 8.7 | 14.1 | 12.0 | 6.6 | 5.6 | 12.4 | 4.0 | 8.4 | 13.9 | 7.8 | 6.1 |
| 32.0 | 19.3 | 12.7 | 23.5 | 15.8 | 7.8 | 15.8 | 10.4 | 5.4 | 13.1 | 4.4 | 8.7 | | | |
| 32.4 | 23.1 | 9.3 | 19.8 | 7.0 | 12.8 | 14.4 | 9.6 | 4.8 | 11.6 | 6.0 | 5.6 | | | |
| 42.6 | 35.6 | 7.0 | 23.5 | 6.9 | 16.6 | 15.4 | 11.0 | 4.0 | 13.0 | 3.4 | 9.6 | | | |
| 35.3 | 23.6 | 11.7 | 22.0 | 5.0 | 17.0 | 11.3 | 7.9 | 3.4 | 11.5 | 7.1 | 4.4 | | | |
| 32.6 | 10.4 | 22.2 | 21.4 | 4.5 | 16.9 | 12.0 | 8.0 | 4.0 | 14.9 | 9.1 | 5.8 | | | |
| 42.3 | 38.5 | 3.8 | 23.5 | 6.9 | 16.6 | . | . . | | 13.1 | 6.1 | 7.0 | | | |
| 36.6 | 12.0 | 24.6 | 23.5 | 14.6 | 8.9 | | | | | | | | | |
| 42.3 | 35.8 | 6.5 | 25.3 | 16.5 | 8.8 | | | | | | | | | |
| ... | ... | . | 23.9 | 7.8 | 16.1 | | | | | | | | | |

It will be noted from the table that on the "air-dried" basis, some of each of the coals have a moisture content of about 8 per cent. and the air-dried analyses of those samples, on the air-dried basis, as might be expected, can hardly be distinguished.

The term "rank" of the coal is here used to designate the extent to which a coal has advanced in its progress from peat to graphite. The term "grade" is here used to designate the purity of the coal with reference to the content of ash, sulfur, or other specific impurities or deleterious action. Most of the coal types are separated by a difference of 750 British thermal units on the basis of the "standard" analysis. To have selected a smaller difference would have increased the number of types and the difficulty of classifying a given coal; and to have increased the difference would have enlarged the range of a coal so that two coals falling within the same type might give an appreciably different service.

Coals differ in three ways: in origin, rank, and grade. These differences may be revealed by either or both the physical and chemical characters of the coal. A notable illustration[2] of coals differing in origin is

[2] G. H. Ashley: Cannel Coal in the United States. U. S. Geol. Survey *Bull* 659

shown by a comparison of cannel and bituminous coals. Here there are both pronounced physical and chemical differences. The differences between "block" coals and similar bituminous coals are due to origin and, so far as present studies have gone, are revealed in their physical characters only.

The rank of a coal is revealed in both its chemical and physical character. Chemically, coal in changing from peat or lignite to graphite shows a progressive elimination of its volatile constituents and a corresponding increase in the proportion of the uncombined carbon and ash. As ordinarily analyzed, fresh peat contains from 80 to 94 per cent. of moisture, from 3 to 7.5 per cent. of volatile matter, from 1 to 4 per cent. of fixed carbon; the rest is ash. Clean peat (not muddy) will have between 90 and 94 per cent. moisture. Fresh lignite has between 40 and 45 per cent. of moisture and about 25 per cent. each of volatile matter and fixed carbon; the rest is ash. There has, then, been a reduction of the proportion of both moisture and volatile matter.

A typical analysis of peat from Beaver Marsh, near Hartford, Conn., shows: 91.2 per cent. moisture, 6.6 per cent. volatile matter, 1.8 per cent. fixed carbon, and 0.3 per cent. ash. If this is freed of ash and the analysis generalized, it might read: moisture 91.5 per cent., volatile matter 6.5 per cent., and fixed carbon 2 per cent., which figures may be assumed to represent the number of pounds of each in 100 lb. (45.359 kg.) of peat. A typical analysis of lignite from Bainville, Valley County, Mont., is: moisture 42.8 per cent., volatile matter 25.7 per cent., fixed carbon 26.8 per cent., ash 4.6 per cent. This, freed from ash and generalized, might read: moisture 45 per cent., volatile matter 27 per cent., fixed carbon 28 per cent. If it be assumed that this lignite had been derived from the peat just described and that in the process there had been no change in the actual amount of fixed carbon, then the 28 per cent. of fixed carbon in the lignite equals 2 lb. (0.9 kg.) comprising all there was in the peat, the 27 per cent. of volatile matter equals 1.93 lb., a loss of 4.57 lb. of volatile matter from that in the peat. Likewise the 45 per cent. moisture represents a loss of moisture from 91.5 lb. to 3.26 lb. If a similar comparison is made between North Dakota lignite and Sheridan, Wyo., subbituminous coal, Iowa coal, and so on up the list, it may be noted that while the amount of moisture in the coal steadily decreases the percentage of volatile matter keeps about even with the percentage of fixed carbon through all of the lower rank coals until the moisture has reached a stabilized minimum, beyond which the percentage of volatile matter is rapidly reduced.

As a matter of fact it can be shown that the amount of fixed carbon does not remain constant but decreases from one type to the next higher so that the actual loss of volatile matter and moisture is greater than indicated, and a study of the ultimate analysis shows that the character

of the volatile matter also undergoes a change. However, for the purpose of distinguishing coals by rank, the simplest system is to assume that the fixed carbon remains stationary and that there is a steady loss of moisture and volatile matter. This may be expressed as a ratio or curve as shown in Table 2.

TABLE 2.—*Ratio of Fixed Carbon to Volatile Matter and Moisture Combined*—$\frac{F.C.}{V.M. + H_2O}$

| Coal | Ratio | Coal | Ratio |
|---|---|---|---|
| Anthracite | 10 7+ | Saint Clair County, Ill , coal | 0 96 |
| Bernice coal | 6 8 | Sangamon County, Ill , coal | 0 84 |
| Brushy Mountain, Va , coal | 4 8 | Grundy County, Ill , coal | 0 78 |
| Pocahontas coal | 3 7 | Sheridan, Wyo., coal | 0 68 |
| Sewell, New River, coal | 2 8 | Carney, Wyo., coal | 0 62 |
| Connellsville coal | 2 0 | Gillette, Wyo , coal | 0 56 |
| Pittsburg coal | 1 60 | Wood County, Tex , lignite | 0 50 |
| Beaver River, Pa , coal | 1 2 | Houston County, Tex., lignite | 0 43 |
| Gallatin County, Ill , coal | 1 09 | Williston, N. Dak., lignite | 0 37 |

Remembering, however, that in the high-rank coals the moisture is stationary and the loss appears to be entirely in the volatile matter, while in the lower rank coals the volatile matter losses appear stationary, with reference to the fixed carbon, it is possible to arrange a double ratio table of which the higher rank coals are distinguished, as now, by the ratio of the volatile matter to the fixed carbon or by the well-known "fuel ratio" and the lower coals by the ratio of moisture (as received) to fixed carbon. A table so constructed appears as shown in Table 3.

TABLE 3.—*Fuel Ratio and Fixed Carbon Moisture (or as here designated F.C.M.) Ratio*

| Coal | Fuel Ratio | Carbon Moisture | Coal | Fuel Ratio | Carbon Moisture |
|---|---|---|---|---|---|
| Anthracite | 10+ | 10+ (30±) | Saint Clair County, Ill. | 1.4– | 4 0–6.0 |
| Bernice | 7–10 | 10+ (27±) | Sangamon County, Ill | 1.4– | 2 5–4.0 |
| Brushy Mountain, Va | 5–7 | 10+ (26+) | Grundy County, Ill | 1.4– | 2 0–2 5 |
| Pocahontas | 3.5–5 | 10+ (24.5) | Sheridan, Wyo. | 1 4– | 1.7–2.0 |
| Sewell | 2 5–3 5 | 10+ (23) | Carney, Wyo. | 1.4– | 1 4–1 7 |
| Connellsville | 1.85–2 5 | 10+ (21.5) | Gillette, Wyo. | 1.4– | 1.0–1.4 |
| Pittsburg | 1.4–1.85 | 10+ (19 5) | Wood County, Tex. | 1 4– | 0 85–1 00 |
| Beaver River, Pa. | 1.4– | 10+ (17) | Houston County, Tex. | 1.4– | 0 65–0.85 |
| Gallatin County, Ill. | 1.4– | 6.0–10.0 | Williston, N. Dak | 1.4– | 0.5–0.65 |

After a careful study of various proposed methods of distinguishing the rank of coal, as well as of many new ones, the writer believes the above method the best yet found, involving, as it does, only proximate

analysis results. According to the table the lignites are found to fall between 0.5 and 1 in the fixed carbon moisture ratio; the principal subbituminous coals fall between 1 and 2; and the so-called bituminous coals between 2 and 10+, beyond this point they are separated on the basis of fuel ratio, as now.

In preparing a scheme to cover all coals full account must be taken of the physical as well as the chemical properties, as many coals having similar chemical composition, as shown by the usual analysis, may differ greatly physically, due to difference of origin or subsequent history, and therefore should not be put into the same class. No scheme based entirely on present known chemical differences may be used to differentiate all types of coal.

In Table 4 attempts have been made to cover all of the common varieties of coal known in the United States. The coals are divided into those of compact texture and those of woody, fibrous, or earthy texture. As a matter of fact, woody texture occurs and may be distinguished with the microscope in coals ranging all the way to moderately high-rank bituminous coals. Where, therefore, doubt exists as to whether a coal has compact, woody, fibrous, or earthy texture, the second test given in the table is used. If the moisture, as received, exceeds the fixed carbon, as received, the coal is classed as lignite or peat. If it does not, the coal is classed as bituminous or higher rank.

Geographic names corresponding to those now in common use in designating coals are proposed for the several types of coals, except that the type name is made to end in *ite* to correspond to the endings of graphite, carbonite, anthracite, and lignite, already in use. Thus, Pocahontas coal is called Pocahontite. Pocahontas coal may continue to mean, as now, coal from the Pocahontas district of West Virginia and Virginia; Pocahontite will mean coal of Pocahontas rank of any grade from any part of the world.

Following the key and omitting discussion of graphite and carbonite (native coke), the coals with compact texture are divided into anthracite and bitumite classes. The first class has a fuel ratio of 7 or more and a non-luminous flame. The second class has a fuel ratio of less than 7 and a luminous flame. The latter class includes the bituminous and subbituminous coals. The luminous flame indicates the presence of hydrocarbons in the volatile matter, and their presence is taken to indicate the bituminous character of the coal. The anthracites may then be divided into the true, or hard, anthracite, with a conchoidal fracture, high specific gravity, and submetallic luster, and the soft anthracite, with semicubic fracture and low specific gravity. The type of the soft anthracites is found at Bernice, Sullivan Co., Pa. The line between the two is drawn at a fuel ratio of 10.

Bitumites are first divided into those having a British thermal unit (B.t.u.) value of over 14,300 and those having a British thermal unit value of under 14,300 on the coal free of moisture, ash, and sulfur. As previously stated, in the change of the coal from peat to anthracite there is a more or less steady change in the composition of the volatile matter. The volatile matter of peat contains from 35 to 60 per cent. of oxygen. In the lower rank post-Carboniferous coals, this high percentage of oxygen is maintained and tends to hold the heat value of the volatile matter to a point well below the heat value of a similar amount of fixed carbon. In the coals of Carboniferous age and in a few of the post-Carboniferous coals, the oxygen in the volatile matter is commonly less than 25 per cent. and the heat value of the volatile matter correspondingly increased. By drawing the line at 14,300 British thermal units it is believed that practically all of the Carboniferous coals, certainly of the United States, are included above the line, and a majority of the post-Carboniferous coals placed below the line.

This British thermal unit figure has been obtained by the formula:

$$\text{B.t.u. (ash, moisture, sulfur free)} = \frac{\text{B.t.u. (coal as received)} - 40S}{100 - (\text{moisture} + \text{ash} + \text{sulfur})}$$

It may be noted that due to climate, or some other cause, some of the western post-Carboniferous coals with a low B.t.u. value of coal on

TABLE 4.—*Classification of Coals, with Key,*

**Key to Classes** (Texture) — **Classes** — **Key to Groups**

**COALS and affiliated hydrocarbons**

- Foliated / Cellular (Both of above metamorphosed)
  - Graphite
  - Carbonite (native coke)
- Compact. $\frac{\text{Fixed carbon}}{\text{moisture}} > 1$
  - Anthracites. Fuel ratio > 7. Flame non-luminous
    - Hard," fracture conchoidal.
    - Soft " fracture cubic
  - Bitumites. Fuel ratio < 7. Flame luminous
    - B.t.u. "Standard Pure" over 14,300 (Includes all coals of carboniferous age)
      - Fuel ratio 7–2.5 (Smokeless and semi-smokeless coals) Flame short to medium
        - Virginites: Non-caking "Admiralty"; Caking: Flame short; Flame medium
        - Fuel ratio 2.5–1.4: Pennaites
      - Fuel ratio < 2.5. Flame long
        - Caking coals "Steam coals": Fuel ratio < 1.4: Ohioites, F C M ratio > 6; Illinoites, F C M ratio < 6
        - Non-caking coals "Household coals"
          - Xyloid (Containing woody parts as do all caking coals. Laid down where found): Splintites
          - Cubical fracture: Fuel ratio > 1
          - Canneloid (Transported spores, etc.) Conchoidal fracture: Canneloid coals; Fuel ratio < 1, Cannels
    - B.t.u. "Standard Pure" under 14,300 (Post-carboniferous age)
      - Weather-resisting: Montanites
      - Non weather-resisting (subbituminous): New Mexites, "Standard Pure" > 14,000 B.t.u.; Wyomites, "Standard Pure" < 14,000 B.t.u.
- Woody fibrous, earthy. F C M ratio < 1
  - Lignites (non-surficial): Xyloid (woody); Intermediate; Canneloid (cannel like)
  - Peat (surficial)
  - Wood

| Key to Types: Fuel Ratio = $\frac{\text{Fixed Carbon}}{\text{Volatile Matter}}$ | Key to Types: F C M Ratio = $\frac{\text{Fixed Carbon}}{\text{Moisture (as rec'd)}}$ | Types Major (Based on differences recognized in the market quotations or equivalent differences. Except for high rank coals average difference of 750 B.t.u.) | Type Localities (Many of coals at type localities not of standard grade and therefore of lower heat value than quoted for the type.) | Code Letters |
|---|---|---|---|---|
| | | Graphite | | |
| | | Carbonite | Midlothian Va | |
| 10+ | 10+ | Anthracite | Wyoming Valley, Pa | A |
| 10.0–7.0 | 10+ | Bernicite | Bernice, Pa | B |
| 7.0–5.0 | 10+ | Brushymontite | Brushy Mts., Va | C |
| 5.0–3.5 | 10+ | Pocahontite | Pocahontas Va | D |
| 3.5–2.5 | 10+ | Sewellite | Sewell W Va | E |
| 2.5–1.85 | 10+ | Connellsite | Connellsville Pa | F |
| 1.85–1.4 | 10+ | Pittsite | Pittsburgh Pa | G |
| <1.4 | 10+ | Belmontite | Belmont Co Ohio | H |
| <1.4 | 10.0–6.0 | Hockingite | Hocking Val Ohio | I |
| <1.4 | 6.0–4.0 | St. Clairite | St. Clair Co., Ill | J |
| <1.4 | 4.0–2.0 | Sangamite | Sangamon Co Ill | K |
| 2.0–1.4 | 10+ | Coalburgite | Coalburg W Va | GG |
| <1.4 | 10.0–5.0 | Kennilworthite | Kennilworth, Utah | II |
| <1.4 | 5.0–2.5 | Brazilite | Brazil Ind | KK |
| <1.4 | 2.5–2.0 | Mendotite | Mendota Mo | LL |
| 5.0–3.5 | 10+ | Altizite | Altizer mine Va | DDD |
| 3.5–2.8 | 10+ | Phillipsite | Phillipsburg Pa | EEE |
| 2.8–2.0 | 10+ | Richmondite | Richmond Pa | FFF |
| 2.0–1.4 | 10+ | New Bethlehemite | New Bethlehem, Pa. | GGG |
| 1.4–1.0 | 10+ | Canfieldite | Canfield, Ohio | HHH |
| 1.0–0.5 | 10+ | Cannelite | | M |
| <0.5 | 10+ | Bogheadite | Boghead, Ky | N |
| <2.0 | 10.0–6.0 | Sand Coulite | Sand Coulee Mont. | O |
| <2.0 | 6.0–4.0 | Bridgerite | Bridger Mont | P |
| <1.4 | 4.0–2.0 | Maxite | Maxey mine Mont. | Q |
| <1.4 | 3.0–5.0 | Gallupite | Gallup N M | R |
| <1.4 | 5.0–3.0 | Hannite | Hanna, Wyo | S |
| <1.4 | 3.0–2.0 | Kirbyite | Kirby Wyo | T |
| <1.4 | 2.0–1.5 | Sheridanite | Sheridan, Wyo | U |
| <1.4 | 1.5–1.0 | Inezite | Ines Wyo | V |
| <1.4 | <1.0 | Wiltonite | Wilton N D | W |
| <1.4 | <1.0 | Hoytite | Hoyt, Tex | X |
| <1.4 | <1.0 | Camdenite | Camden Ark | Y |
| | | Peat { Air dried | Orlando Fla. | Z |
| | | Peat { Fresh | New Haven Conn | |
| | | Wood { Dry / Seasoned } ($C_6H_9O_4$ ±) | | |
| | | Lignocellulose ($C_{12}H_{18}O_9$) | | |
| | | Cellulose ($C_6H_{10}O_5$) | | |

TABLE OF LETTER ABBREVIATIONS TO EXPRESS GRADE

| Letters | a: Ash, Per Cent. | s: Sulfur, Per Cent. | f: Fusibility of Ash, °F |
|---|---|---|---|
| vl = very low | 0–4 | 0.00–0.75 | <2200 |
| l = low | 4–8 | 0.75–1.50 | 2200–2400 |
| m = medium | 8–12 | 1.50–2.50 | 2400–2600 |
| h = high | 12–16 | 2.50–4.00 | 2600–2800 |
| vh = very high | >16 | >4.0 | >2800 |

COALS ADAPTED TO

| | | |
|---|---|---|
| Navy coal | C-E la, ls, bf | |
| Household smokeless | A, B, C. | Coking coal |
| Household semi-smokeless | D E | Gas coal |
| Household block grate | GG N | |
| Smithing coal | D-E la, ls | |
| By product coal | F-G la, ls | Steam coal |
| Stoker non-clinkering coal | C-K ma, ls | |
| Fuel-oil distillation coal | M N | |
| | | Railroad coal |

ash, moisture, sulfur free basis have a lower percentage of moisture than some of the coals of Carboniferous age, and therefore a higher British thermal unit value on the "as received" basis, notwithstanding the low rank and character of the volatile matter. The coals below 14,300 British thermal units on the ash, moisture, and sulfur free basis include, in general, the coals commonly called brown coals in Europe, following the classification of Zincken[3] and others.

The higher rank bitumites are then divided into the Virginites or so-called smokeless coals having a fuel ratio between 2.5 and 7, a short to medium flame, and coals having a fuel ratio below 2.5 and a long flame. The Virginites are divided into three types, those having a fuel ratio of from 5 to 7, from 3 5 to 5, and from 2.5 to 3.5. The first is a non-caking coal with a fuel ratio between 5 and 7. This coal is not abundant in this country, though found in the Brushy Mountain field of Virginia, the Coal Hill field of Arkansas, and a few other places. This is the type of the best "Admiralty" coals of Wales, so long used by the British Navy. The other two types are both caking coals, that is they swell up on burning and run together into a cake and if other characters are favorable may be used in the making of coke. They are separated into two types, according to the length of flame. The short-flame type is typified by the Pocahontas coal of eastern-central McDowell Co., W. Va., and

*Analyses, and Various Data and Ratios*

| Heat Value in British Thermal Units (B t u): As received but Standard coal (Reduced to 6 per cent ash 1 per cent sulfur 0 75-1 50 per cent nitrogen as beyond) | | | Analyses (Standard Reduced to 6 Per Cent. Ash S 1 Per Cent N as Below) from 'As Received' Basis: Proximate | | | | | | Ultimate | | | | | | Standard Pure (Ash, S and N Free) | | | Volatile Matter: Per Cent. of Whole | | | Per Cent. of Vol Mat. | | | | Carbon Ratios | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Range | Aver | Coal (Ash sulfur moisture free) | Moisture Range | Moisture Aver | Volatile Matter | Fixed Carbon | Ash | Sulfur | Hydrogen | Carbon | Nitrogen | Oxygen | 100/Oxygen | Hydrogen | Carbon | Oxygen | Volatile Hydrogen | Volatile Carbon | Volatile Oxygen | Volatile Hydrogen | Volatile Carbon | Volatile Oxygen | B t u of 1 Per Cent. Volatile Matter | Carbon/Oxygen | Carbon/Hydrogen | Carbon/Volatile Carbon |
| 13,850-13 250 | 13 700 | 15 250 | <4 | 3 0 | 5 0 | 86 0 | 6 0 | 1 0 | 2 75 | 85 0 | 0 75 | 4 5 | 22 2 | 3 0 | 91 6 | 5 4 | 2 45 | 1 (?) | 1 8 | 46 6 | 20 9 | 34 3 | 220 | 18 9 | 31 7 | 85 (?) |
| 14 000-13 650 | 13 850 | 15 750 | <4 | 3 0 | 9 0 | 82 0 | 6 0 | 1 0 | 3 75 | 83 5 | 1 0 | 4 75 | 21 6 | 4 0 | 90 0 | 6 0 | 3 45 | 1 5 | 2 0 | 49 6 | 21 5 | 28 7 | 204 | 17 5 | 22 2 | 55 0 |
| 14 600-12 000 | 14 500 | 15,750 | <4 | 3 0 | 13 0 | 78 0 | 6 0 | 1 0 | 4 0 | 83 0 | 1 0 | 5 0 | 20 0 | 4 3 | 89 4 | 6 3 | 3 7 | 5 0 | 2 0 | 33 6 | 45 4 | 20 9 | 160 | 16 6 | 20 7 | 16 6 |
| 14 850-14,250 | 14 650 | 16 000 | <4 | 3 0 | 17 0 | 74 0 | 6 0 | 1 0 | 4 5 | 82 0 | 1 25 | 5 25 | 19 0 | 4 7 | 88 8 | 6 5 | 4 2 | 8 0 | 2 5 | 28 6 | 54 4 | 17 0 | 210 | 15 5 | 18 3 | 10 2 |
| 14 650-14 000 | 14 250 | 16 000 | <4 | 3 0 | 22 0 | 69 0 | 6 0 | 1 0 | 4 75 | 81 0 | 1 5 | 5 75 | 17 4 | 5 0 | 88 0 | 7 0 | 4 45 | 12 0 | 3 0 | 22 8 | 61 7 | 15 4 | 200 | 14 0 | 17 0 | 6 7 |
| 14 350-13 750 | 14 000 | 16 000 | <4 | 3 0 | 27 0 | 64 0 | 6 0 | 1 0 | 5 0 | 80 0 | 1 5 | 6 5 | 15 3 | 5 3 | 87 1 | 7 6 | 4 7 | 16 0 | 3 8 | 19 2 | 65 3 | 15 5 | 170 | 12 3 | 16 0 | 5 0 |
| 14,250-13 700 | 14 000 | 15 750 | <4 | 3 0 | 33 0 | 58 0 | 6 0 | 1 0 | 5 2 | 78 0 | 1 5 | 8 3 | 12 0 | 5 7 | 84 5 | 9 8 | 4 9 | 20 0 | 5 6 | 21 9 | 59 8 | 18 3 | 166 | 9 3 | 15 0 | 3 9 |
| 13 750-13,000 | 13 400 | 15 500 | <4 | 3 0 | 40 0 | 51 0 | 6 0 | 1 0 | 5 4 | 76 0 | 1 5 | 10 1 | 9 9 | 5 9 | 83 3 | 10 9 | 5 1 | 25 0 | 7 4 | 13 7 | 66 6 | 19 7 | 148 | 7 5 | 14 0 | 3 0 |
| 13 000-12 250 | 12 750 | 15,250 | 4- 8 | 6 0 | 39 0 | 49 0 | 6 0 | 1 0 | 5 4 | 72 0 | 1 5 | 14 1 | 7 1 | 5 9 | 79 0 | 15 3 | 4 7 | 23 0 | 8 7 | 13 6 | 63 1 | 23 9 | 143 | 5 1 | 13 4 | 3 2 |
| 12 250-11 500 | 12 000 | 14 750 | 8-12 | 10 0 | 38 0 | 46 0 | 6 0 | 1 0 | 5 4 | 69 0 | 1 5 | 17 1 | 5 8 | 5 9 | 75 5 | 18 6 | 4 8 | 23 0 | 8 2 | 12 2 | 64 7 | 23 1 | 139 | 4 0 | 12 8 | 3 0 |
| 11,500-10 750 | 11 250 | 14 400 | >12 | 14 0 | 37 0 | 43 0 | 6 0 | 1 0 | 5 5 | 66 0 | 1 5 | 20 1 | 5 0 | 6 0 | 73 2 | 20 8 | 4 0 | 23 0 | 7 5 | 11 7 | 66 6 | 21 7 | 133 | 3 3 | 12 0 | 2 8 |
| 14,000-13 000 | 13 750 | 15,300 | 1- 3 | 3 0 | 35 0 | 56 0 | 6 0 | 1 0 | 5 3 | 77 7 | 1 0 | 9 0 | 11 1 | 5 7 | 84 6 | 9 7 | 5 0 | 21 7 | 6 3 | 13 2 | 67 8 | 10 0 | 159 | 8 6 | 14 6 | 3 6 |
| 13 000-12 000 | 12,500 | 14 500 | 3- 6 | 5 5 | 40 0 | 48 5 | 6 0 | 1 0 | 5 5 | 70 5 | 1 0 | 16 0 | 6 2 | 5 7 | 82 2 | 13 0 | 5 0 | 22 0 | 8 0 | 13 5 | 65 0 | 21 5 | 145 | 4 3 | 12 9 | 3 2 |
| 12 500-11 500 | 11 750 | 14 800 | 12-15 | 15 0 | 33 0 | 46 0 | 6 0 | 1 0 | 5 6 | 66 4 | 1 0 | 20 0 | 5 0 | 6 5 | 67 8 | 26 2 | 5 0 | 20 4 | 6 9 | 11 3 | 63 1 | 21 3 | 151 | 2 5 | 10 2 | 3 8 |
| 11,500-10 000 | 11,000 | 14 750 | 15-20 | 17 5 | 35 0 | 41 5 | 6 0 | 1 0 | 6 0 | 60 0 | 1 0 | 26 0 | 3 9 | 6 5 | 65 2 | 28 3 | 4 4 | 18 0 | 11 5 | 13 1 | 39 5 | 33 9 | 135 | 2 3 | 10 2 | 3 4 |
| 14 650-14 200 | 14 400 | 16 000 | <10 | 3 0 | 19 0 | 72 0 | 6 0 | 1 0 | | | | | | | | | | | | | | | | | | |
| 14 650-14 000 | 14 250 | 16 000 | <10 | 3 0 | 22 0 | 71 0 | 6 0 | 1 0 | | | | | | | | | | | | | | | | | | |
| 14 350-13 750 | 14,000 | 16 000 | <10 | 3 0 | 30 0 | 61 0 | 6 0 | 1 0 | | | | | | | | | | | | | | | | | | |
| 14 200-13 700 | 14,000 | 15 750 | <10 | 3 0 | 36 0 | 55 0 | 6 0 | 1 0 | | | | | | | | | | | | | | | | | | |
| 13 750-13 000 | 13 400 | 15,500 | <10 | 3 0 | 42 0 | 49 0 | 6 0 | 1 0 | | | | | | | | | | | | | | | | | | |
| 14 250-12 500 | 13,500 | 16,000 | <10 | 3 0 | 53 0 | 38 0 | 6 0 | 1 0 | 7 0 | 76 0 | 1 0 | 9 0 | 11 1 | 7 6 | 82 6 | 9 8 | 6 6 | 38 0 | 8 4 | 12 4 | 71 7 | 15 9 | 130 | 8 4 | 10 8 | 10 4 |
| | | | <10 | 3 0 | 64 0 | 27 0 | 6 0 | 1 0 | 7 5 | 74 0 | 2 5 | 9 0 | 11 1 | 8 3 | 81 7 | 10 0 | 7 2 | 37 0 | 19 8 | 11 2 | 57 8 | 31 0 | | 8 2 | 9 9 | 11 6 |
| 12,250- 9 750 | 12,000 | 14 250 | 4- 8 | 7 0 | 31 0 | 56 0 | 6 0 | 1 0 | 5 0 | 69 0 | 1 0 | 18 0 | 3 5 | 5 4 | 75 0 | 19 6 | 4 3 | 13 0 | 11 7 | 14 9 | 44 8 | 40 3 | 122 | 3 6 | 13 8 | 5 3 |
| 11,600-11 000 | 11 500 | 13,900 | 8-12 | 10 0 | 35 0 | 49 0 | 6 0 | 1 0 | 5 5 | 65 5 | 1 0 | 21 0 | 4 7 | 6 0 | 71 2 | 22 8 | 4 5 | 16 0 | 12 5 | 13 8 | 48 4 | 37 8 | 121 | 3 1 | 11 9 | 4 1 |
| 11 000-10 000 | 10 500 | 13 250 | 12-18 | 16 0 | 35 0 | 43 0 | 6 0 | 1 0 | 5 5 | 58 0 | 1 0 | 29 0 | 3 4 | 6 0 | 63 0 | 31 0 | 3 6 | 16 0 | 13 8 | 10 9 | 47 8 | 41 3 | 124 | 2 0 | 10 5 | 3 6 |
| 12,000-11,250 | 11,800 | 14,200 | 9-15 | 12 0 | 39 0 | 43 0 | 6 0 | 1 0 | 5 8 | 61 0 | 1 0 | 22 2 | 4 5 | 6 3 | 69 6 | 24 1 | 4 5 | 18 0 | 11 5 | 13 3 | 52 9 | 33 8 | 133 | 2 7 | 10 5 | 3 3 |
| 11,500-10 500 | 11,000 | 13,500 | 9-15 | 12 5 | 39 5 | 42 0 | 6 0 | 1 0 | 6 0 | 62 0 | 1 0 | 24 0 | 4 1 | 6 5 | 67 4 | 26 0 | 4 6 | 20 0 | 13 0 | 12 2 | 53 2 | 34 6 | 123 | 2 5 | 9 7 | 3.1 |
| 11 000- 9,750 | 10 250 | 13 250 | 15-18 | 17 5 | 36 5 | 40 0 | 6 0 | 1 0 | 6 2 | 57 5 | 1 0 | 28 8 | 3 4 | 6 7 | 62 5 | 30 8 | 4 2 | 17 5 | 12 6 | 12.5 | 50 4 | 37 1 | 116 | 2 0 | 9 3 | 3 3 |
| 9 750- 8 750 | 9,400 | 13 000 | 20-25 | 22 5 | 34 0 | 37 5 | 6 0 | 1 0 | 6 25 | 53 0 | 1 0 | 33 0 | 3 0 | 6 9 | 57 5 | 35 6 | 3 75 | 15 5 | 13 0 | 11 7 | 48 0 | 40 3 | 115 | 1 6 | 8 4 | 3 3 |
| 9 000- 8,000 | 8,750 | 13,000 | 25-30 | 27 5 | 33 0 | 36 0 | 6 0 | 1 0 | 6 25 | 50 0 | 0 75 | 35 0 | 2 8 | 6 9 | 54 3 | 38 8 | 3 5 | 14 5 | 12 7 | 11 5 | 47 2 | 41 3 | 105 | 1 4 | 8 0 | 3 4 |
| 8,000- 5 500 | 7 000 | 13 000 | 30-45 | 37 5 | 28 5 | 28 0 | 6 0 | 1 0 | 6 8 | 40 0 | 0 75 | 45 45 | 2 2 | 7 4 | 43 3 | 49 3 | 2 6 | 12 0 | 10 0 | 15 6 | 47 8 | 36 6 | 110 | 0 87 | 5 9 | 3 3 |
| 8 000- 6,000 | 7,500 | 13 000 | 30-40 | 35 0 | 29 0 | 30 0 | 6 0 | 1 0 | 6 7 | 43 0 | 0 75 | 42 55 | 2 3 | 7 3 | 46 6 | 46 1 | 2 9 | 13 0 | 11 6 | 10 5 | 47 5 | 42 2 | 110 | 1 0 | 6 4 | 3 3 |
| | | | 30-40 | 35 0 | 41 0 | 18 0 | 6 0 | 1 0 | | | | | | | | | | | | | | | | | | |
| 9,000- 2,000 | 8,000 | 9 800 | 10-80 | 20 0 | 52 0 | 22 0 | 6 0 | 1 0 | 6 5 | 50 0 | 0 50 | 36 0 | 2 7 | 7 1 | 54 0 | 38 9 | 4 3 | 28 0 | 18.2 | 8 5 | 55 4 | 36 1 | 99 | 1 3 | 7 7 | 1 7 |
| 2 000- 400 | 800 | 9,800 | 80-94 | 90 0 | 7 0 | 2.5 | 0 5 | 0 05 | 10 35 | 5 0 | 0 10 | 84 0 | 1 2 | 10 4 | 5 0 | 84 6 | 0 35 | 2 5 | 4 0 | 5 1 | 36 5 | 58 4 | | 0 06 | 0 48 | 2 0 |
| 9,500- 7 500 | 8,500 | 8 600 | | | 74 0 | 25 0 | 1 0 | ? | 6 0 | 49 0 | 0 10 | 43 9 | 2 3 | 6 3 | 49 2 | 44 5 | 6 25 | 24 0 | 44 5 | 8 3 | 32 5 | 59 4 | | 1 1 | 8 1 | 2 0 |
| 7,200- 5,500 | 6,500 | | 15-30 | 25 0 | 55 5 | 18 75 | 0 75 | ? | 7 25 | 36 75 | 0 10 | 55 15 | 1 8 | 7 4 | 37 0 | 55 6 | 4 5 | 11 75 | 32 9 | 9 1 | 23 3 | 67 6 | 67 | 0 66 | 5 1 | 3 1 |
| | | | | | | | | | | | | | | 5 89 | 47 06 | 47 05 | | | | | | | | 1 00 | 8 0 | |
| | 7 480 | | | | | | | | | | | | | 6 22 | 44 43 | 40 35 | | | | | | | | 0 90 | 7 1 | |

VARIOUS USES

| High grade. | Medium grade. | Low grade. |
|---|---|---|
| D-G la, ls. | D-G ma, ls | D-G ha, ls. |
| G-H la, ls | G-H ma, ls.<br>I la, ls. | G-H ha, ls.<br>I ma, ls. |
| A-E ma, ls hf<br>F ma, ls hf<br>G ma, ls, hf | A-G ha, hs lf<br>H-K ma, ls, hf | H-K hs hs, lf<br>O-Q ma, ls.<br>R-V la ls. |
| A-E ma, ls, hf<br>F ma, ls, hf<br>G ma, ls, hf | A-G ha ha, lf<br>H-K ma, ms hf | H-K ha, ha.<br>O-Q ma, ma. |

EXPLANATORY NOTES

The classification here proposed is based on both the physical and chemical characters of the coal. The use of the key requires only proximate analyses, on the 'as received' basis. Of two characters that given first should be used if possible Thus a coal with fibrous structure is a lignite If the structure is not clear as between earthy and compact the classification is determined by the F C M ratio A coal 'cakes' if when pieces through ¼ in screen and over 10 mesh screen are heated as in determining volatile matter they lose shape and run together A caking coal is not necessarily a commercial coking coal A non-weather-resisting coal is one that when exposed to the weather for a month loses shape and breaks up with an irregular or hackly fracture Analyses of volatile matter obtained by subtracting fixed carbon from total carbon, and hydrogen and oxygen in moisture from total hydrogen and oxygen.

[3] C. F. Zincken: "Die Physiographie der Braunkohle," 5. Leipsic, 1867.

the medium-flame type by the Sewell coal below Thurmond on the New River. The line between them and on either side is drawn on the basis of the fuel ratio. The Pocahontas type is limited by a fuel ratio of from 3.5 to 5 and the Sewell type by a fuel ratio of from 2.5 to 3.5. In accepting the lowest fuel ratio just given, account was taken of the fact that Sewell coals with a fuel ratio of 2.8, as sampled by the writer, have been on the Navy accepted list.[4]

The long-flame coals are divided into the caking, or steam, coals, and the non-caking, or household, coals. The caking long-flamed coals are then divided into two groups according as their fuel ratio is above or below 1.4. The former are here called the Pennsites, from their well-known occurrence in Pennsylvania ("Pennsy"). The Pennsites include two types, according as the fuel ratio is above or below 1.85. Those above 1.85 are termed Connellsite, from their typical occurrence in the Connellsville basin in Fayette and Westmoreland counties, Pa. They are commonly suited to the making of coke in beehive ovens if of proper grade. The lower group, having a fuel ratio of between 1.4 and 1.85, are called Pittsites and are typified by the Pittsburgh coal south of Pittsburgh. They are good steam coals and, if of proper grade, are suitable for making gas and by-product coke.

The coals having a fuel ratio below 1.4 are then divided into the Ohioites, characteristically developed in Ohio, having a fixed carbon moisture ratio of more than 6; and Illinoites, high-moisture coals having a fixed carbon moisture ratio of less than 6. There is a marked change in calorific value in Ohio coals from east to west, due to increasing moisture content toward the west. On this basis, the coals are treated as of two types, Belmontites, as found in the Belmont field, having a fixed carbon moisture ratio of over 10, and Hockingites, as found in the Hocking Valley field, with a fixed carbon moisture ratio between 6 and 10.

The coals of Illinois change markedly in rank from southeast to northwest. In southern Gallatin County, the coal is of Belmontite rank. North and west of that, the coal as received contains from 4 to 8 per cent. of moisture. These coals have a British thermal unit value of from 12,250 to 13,000. West of that and typically developed in St. Clair County, just east of St. Louis, is found the type of coal here called St. Clairite. This type has a fixed carbon moisture ratio of 4 to 6 and a British thermal unit value of between 11,500 and 12,250. Farther north and centering about Sangamon County is a type of coal with high moisture, the fixed carbon moisture ratio being 2.5 to 4, and the British

[4] Attention should be called to the statements on page 29, *Bulletin* 22, of the Bureau of Mines, regarding the unreliability of the determination of volatile matter in analyses bearing laboratory numbers between 5147 and 9120. In this study, those analyses have been discarded in so far as they bore on fuel ratio or content of volatile matter or fixed carbon.

thermal unit value between 10,750 and 11,500; it is here called Sangamite. In the northern coal counties of Illinois, the coal commonly has over 15 per cent. moisture.

The non-caking, or household, coals differ in origin from the caking coals and are divided into two groups, here called the splintites, or splint coals, which have a laminated structure and cubic or tabular fracture; and the cannelites, or cannel coals, which have a massive structure and conchoidal fracture. The splintites are characteristically high-moisture coals, as compared with other bituminous coals of the same region. This is explained by the writer as being due to the absorption, or holding, of the moisture in the mineral charcoal layers of the coal. The cannel coals are characteristically low-moisture coals but the typical cannel coals are high in volatile matter. Four types of splintite have been picked, which differ both physically and chemically and in heat value. It is not necessary to describe them in detail, as probably all are well known to coal men. Coalburgite is a typical West Virginia "splint" coal named from Coalburg, W. Va., which appears to have been the first point from which that type of coal was extensively shipped. Kennilworthite is a low-moisture non-caking coal found at Kennilworth, Utah; it is fundamentally a little lower in rank than Brazilite, but probably climatic conditions have reduced its moisture content so that according to the plan of classification it stands higher. Brazilite is the type known for over half a century as Indiana block coal or Brazil block coal. Mendotite is a type of block coal, found at Mendota, Mo., that differs from the last mainly in its lower heat value.

The cannelites, including cannel and canneloid coals, are differentiated by differences of fuel ratio. Canneloid anthracite and semi-anthracites are not listed here as their canneloid character does not affect their use. Altizite, named from the Altizer mine, ½ mi. (804.6 m.) north of Faraday, Tazewell Co., Va., is a non-coking coal in contrast with the Pocahontas coal, which is a coking coal, and therefore may possibly find a different use. The other types down to Canfieldite are lean cannels or canneloid coals having the physical properties of cannels but not the chemical properties. True cannels have been defined as coals of bituminous rank having a fuel ratio of less than 1.

The lower rank bituminous coals having a fuel value of less than 14,300 B.t.u. on the ash, moisture, sulfur-free basis are divided into two groups: those that resist weathering and may be stocked or shipped long distances and those that, when exposed to sun and rain, tend to break down rapidly. If necessary to draw a definite line between these groups, it is suggested that coals, lumps of which, free of pyrite, exposed to alternate wetting and drying break down and lose their shape within one month shall be classed as non-weather-resisting. The weather-resisting group is divided into three types, according as they are low, medium, or high in moisture

or in the reverse order in difference of fixed carbon moisture ratio. These coals are called Montanites, as the three types are all taken from Montana.

The low rank non-weather-resisting bituminous coals, or subbituminous coals, are characterized by their lightness and tendency to break down as they lose moisture, the fracturing commonly following irregular, or zigzag, lines. The types are all drawn from Wyoming with one exception, and are therefore called Wyomites. The one exception is Gallupite, of which the type locality is Gallup, N. Mex., and which is therefore grouped as New Mexite. Gallupite and Hannite have many points of resemblance, but judging by the character of the volatile matter Gallupite is of a considerably higher rank than Hannite.

The term lignite, though properly applied only to coals having a woody structure, is in this country applied to coals of all kinds in the first stages of anthracitization. A study of the coals that have been classed as lignite in this country, not including those formerly called black lignite and more recently subbituminous, reveals that almost without exception they contain in the "as received" sample over 30 per cent. moisture, while the black lignites or subbituminous coals almost without exception contain less than 30 per cent. moisture. It is proposed, therefore, that coals in which the woody, fibrous, or earthy texture is obvious shall be called lignite, regardless of their moisture content, but that of coals in which the woody texture is not obvious only those having a fixed carbon moisture ratio of less than 1 on the "as received" basis shall be classed as lignite. As so grouped, the class lignite includes coals that range from those obviously 75 to 85 per cent. wood to those that do not contain any wood, such as the canneloid coals derived from accumulation of algæ, spores, and spore cases and other non-woody vegetal material accumulated in water to which the name "sapropel" has been given. If the term "xyloid," meaning woody, is applied to the woody lignites, the class lignites may be divided into three subclasses, xyloid lignites and sapropel, or canneloid lignites, and an intermediate class.

Lignites differ greatly in percentage of moisture and thus in fixed carbon moisture ratio, in fuel ratio, and in heating value. Unfortunately the coal from the same mine differs so greatly that it is not possible to classify the coal closely on these characters. Thus, the fuel ratio of the coal from Hoyt No. 3 mine in Wood County, Tex., varies from 0.46 to 1.14. Coal from the Snyder mine 8 mi. (12.87 km.) north of Glendive, Mont., varies in fuel ratio from 0.31 to 1.0. In like manner, the moisture content will vary according as the coal is fresh or has had time to dry out. Thus mine samples from the Lehigh mine, Starke County, N. D., all have over 42 per cent. moisture, but samples taken from carload lots from the same mine range from 32 to 35 per cent. moisture. A classification on the fixed carbon moisture basis would put the same coal in one type as it came

fresh from the mine and in another at the point of delivery, the coal meanwhile having had time to dry out. Therefore classification is restricted to differences of texture as obvious to the naked eye.

Following the key and table of types are given first the locality from which the names are derived, then the range and average British thermal unit value of the coal reduced to standard analyses on the "as received" basis; then the British thermal unit value on the ash, sulfur and moisture free basis The next column gives the range and average moisture content of the coals when reduced to standard ash and sulfur. Then follow proximate and ultimate analyses on the standard basis, that is including moisture as received but reduced to 6 per cent. ash, 1 per cent. sulfur, and a varying content of nitrogen. Following the analyses are given a number of ratios that may prove of interest to any desiring to study further into the classification proposed. These include the inverse ratio of oxygen to the whole of the coal, the ratio of the carbon to the oxygen, of carbon to hydrogen, including the hydrogen of the moisture in the standard analyses, of carbon to the volatile carbon. There is also given an ultimate analysis on what is designated "standard pure" basis, that is freed of ash, sulfur, and nitrogen, but including the moisture. Two columns are devoted to the volatile matter, giving first the percentage of the three elements, volatile hydrogen, volatile carbon, and volatile oxygen in terms of all the coal, and in the second column in terms of the volatile matter. The percentages of the elements of the volatile matter are not determined by actual analyses of the volatile matter but in the usual method of subtracting the fixed carbon from the total carbon to obtain volatile carbon, by subtracting the hydrogen and oxygen of the moisture from the total hydrogen and oxygen to obtain the volatile hydrogen and volatile oxygen. There is also inserted, as of possible interest, the heat value of 1 per cent. of volatile matter derived by subtracting from the total B.t.u. value of the coal the B.t.u. value of the fixed carbon and dividing by the percentage of volatile matter. It may be of interest to note that the figure for any coal is not far from three times the percentage of fixed carbon; in fact, in an initial study of this topic, using individual analyses, the result was even more striking than is brought out in the generalized table.

In these days, so much business is done by wire that it is often desirable to have, in addition to names, some letter, symbol, or word that may be used to designate the various ranks of coal. The system given is to give a capital letter to each type, except that non-caking splint coals are given the same letter as the corresponding caking coal, but doubled, while three letters are used to designate the canneloid coals and at the same time indicate the type of caking coal to which they correspond. If canneloid coals corresponding to types *I*, *J*, and *K* are found, they will take the corresponding letter tripled.

The classification presented is based, as stated, on a standard analysis reduced to contain 6 per cent. ash, 1 per cent. sulfur, 0.75 to 1.50 per cent. nitrogen. But coals differ in grade as well as in rank and an economic classification should provide for their classification by grades as well as by rank. Aside from the presence of phosphorus in coals in too large amounts for the making of steel, the three factors commonly affecting the grade of coal are ash, sulfur, and fusibility of ash.

Little attention has, in the past, been paid to the fusibility of ash but engineers are beginning to realize that a coal with ash with a low fusing point may, by blocking the grate bars, give a much lower duty than other coals that by the analyses alone are of lower grade and heat value. To present these facts, two methods may be adopted: the designation of the rank of the coal by name or letter may be followed by a brief descriptive statement of the coal, as—Pocahontite, 7 per cent. ash, 8 per cent. sulfur, high fusibility. Or, to facilitate brief description by wire or cable, certain letters to which are assigned definite range of meaning may be used. Thus, it is suggested that *a*, *s*, and *f* stand respectively for ash, sulfur, and fusibility, and that these be prefixed by small letters which should have the following limited meanings:

*Table of Letter Abbreviations to Express Grade of Coal*

| | Ash, Per Cent. | Sulfur, Per Cent | Fusibility, Degrees F. |
|---|---|---|---|
| *vl* = very low .... | 0–4 | 0–0.75 | Less than 2200 |
| *l* = low . . . . ...... | 4–8 | 0.75–1 5 | 2200 to 2400 |
| *m* = medium . . . . . | 8–12 | 1 5 –2 5 | 2400 to 2600 |
| *h* = high.... ... . . | 12–20 | 2 5 –4 | 2600 to 2800 |
| *vh* = very high. . . . ....... | Over 20 | Over 4 | Over 2800 |

Using these letters and those that indicate the rank of the coal it is possible to describe a coal fully, as follows: *D*, l*a*, l*s*, *hf*, stands for a smokeless coking coal with a fuel ratio between 3.5 and 5, an ash between 4 and 8, a sulfur content between 0.75 and 1.50, an ash with a fusing point of between 2600 and 2800° F.

Should this proposed plan of coal classification meet with definite approval, it is proposed to prepare an extended paper discussing the whole problem in detail, with detailed descriptions of each type and possibly maps showing the occurrence of the several types over the country.

## DISCUSSION

R. D. Hall, New York, N. Y.—In view of the fact that we are now going into the foreign markets and query is being made into what kind

of coals we have to supply, it is a good thing that we should have such a use classification as Mr. Ashley has suggested, so we can bind together in one class all the coals of a similar character. Heretofore, it has been customary to describe coals not by their nature so much as by their localities. Mr. Ashley's idea is to show the coals particularly suited for certain purposes. He would bind like coals in one state with like coals in another so that all the coal available would be readily ascertained as soon as one knew the exact character of the coal wanted. Mr. Ashley has very carefully made a classification of all the coals of the United States. Perhaps the names that he has given them will occasion much dissatisfaction; that is altogether to be expected, but he has started something we all need and we may arrive, finally, at a knowledge as to which of the various kinds of coals we are justified in grouping together. He has called, for instance, one kind of coal "Pocahontite;" that may or may not be satisfactory to the Pocahontas people. He includes in that class a large quantity of coal in Pennsylvania which has similar characteristics. Any one who wants coal similar to Pocahontas can find coals of that quality grouped under the head "Pocahontite."

E. N. Zern,* Pittsburgh, Pa.—I am wondering how far Doctor Ashley expects to go into this scheme of classification. In order to be highly useful, commercially, it will require that all coals mined in the United States be classified in accordance with their physical and chemical qualities and take their place under a type name. Such an arrangement would prove very useful, indeed, to the consumer of coal, especially to those who require a coal for such particular purposes as byproduct coking or for the burning of fine ceramic ware.

George H. Ashley (author's reply to discussion).—In the preparation of the classification presented, graphic studies were made of all the analyses of coal that had been made by the Bureau of Mines, as well as of a large number of other analyses both domestic and foreign. This involved, in part, the preparation of maps, some of which have since been published.[1]

It is the author's plan, after learning whether such proposed classification meets in some measure the needs of the industry, to prepare a detailed report with extensive maps showing just how the classification works out, as applied to the coal fields of the United States. It is his hope that later, if desirable, its application may be extended to the coal fields of the world, in order that comparison may be made between domestic and foreign coals. When first written, the types and type localities were drawn from foreign as well as domestic fields. Unfortunately, the

* Editor, Keystone Cons. Pub. Co.

[1] *Coal Age* (Dec. 23, 1919).

lack of uniform analyses of foreign coals with the analyses of the coals of this country, made by the Bureau of Mines, made such type determination hazardous and led to basing the whole scheme on the Bureau of Mines' analyses and to the omission of a large group of these.

It is the author's plan to make a complete classification survey of the coals of the United States as far as the information available warrants, provided the classification proposed appears, to those connected with the industry, to meet the needs of the industry and to warrant such a detailed report.

# Height of Gas Cap in Safety Lamp*

BY C. M. YOUNG,† E. M, URBANA, ILL.

(Chicago Meeting, September, 1919)

THE safety lamp is the most common and convenient apparatus for detecting inflammable gases in mines, the presence of gas being shown by a blue flame, called the cap, if the wick has been lowered to suppress the luminous flame. The height of the cap increases with the proportion of gas and with the temperature of the lamp flame. Lamps of the Wolf type, the fuel of which is benzine, have a hotter flame than lamps in which vegetable oils are used, such as the Davy, and the Clanny, and are more sensitive than the latter and their many modifications. The Piehler alcohol lamp gives a still higher cap while the Clowes hydrogen lamp is most sensitive of all. While the effect of the temperature of the source of ignition is well known, the writer knows of no previous attempt to correlate the change of this temperature with change of the height of the cap produced.

In order to determine the height of the cap at various temperatures, it was necessary to have a source of ignition the temperature of which could be accurately controlled over a considerable range. This condition was met by a coil of wire of high fusing point heated by an electric current controlled by a variable resistance. The use of a coil offered the further advantage of freedom from complications due to the flame of a burning fuel. Such a flame, being itself of variable height, would to some extent raise or lower the elevation of the base of the cap and therefore add to the difficulty of determining the height of the cap. Besides this, the presence of a small cap from the lamp fuel, commonly known as the "fuel cap," would interfere with the observation of small gas caps.

After some experiment, a coil was made of platinum wire, No. 26, diameter 0.016 in. The coil was 17/64 in. (6.746 mm.) in outside diameter and had eight full turns in a length of ½ in. (12.69 mm.). This was connected to an electric circuit through an adjustable resistance, as shown in Fig. 1. In order to measure the temperature, a thermocouple was inserted in the center of the coil; the leads passed out at the ends and were conducted to a millivoltmeter. It is probable that the current of air passing upward through the coil resulted in the indication of a tempera-

* Work done in ventilation laboratory of Department of Mining Engineering, University of Illinois.

† Assistant Professor of Mining Research, University of Illinois.

ture somewhat below that of the coil, but the error was not of a nature to change the general conclusion drawn from the observations. Partly because of the fact just mentioned and partly because the impossibility of measuring the exact height of the cap rendered refinements in other parts of the work unnecessary, no cold junction was used in connection with the thermocouple. Moreover, there was no object in ascertaining the exact temperature at which a given percentage of the particular gas used would give a cap of certain height. The height of cap varies

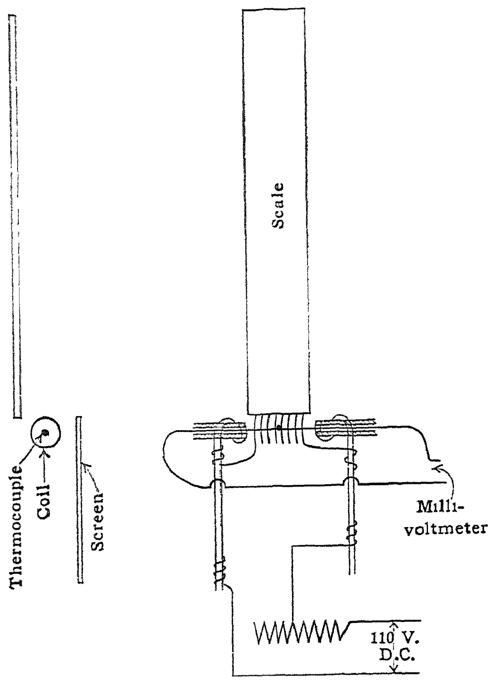


Fig. 1.

with the composition of the gas and the results obtained could not be checked with those obtained with another gas unless that gas were measured in accordance with its cap-forming power instead of its composition and percentage. The object of the experiment was not to find the exact height of cap given with a certain percentage of gas at a given temperature of the coil, but to show that the height of the cap varies with the temperature of the source of ignition. Points such as the relation between the exact temperature of the source of ignition and the height of the cap in mixtures of pure methane and air, and the relation between the height of the cap and the size of the igniting body, are reserved for future investigation.

A scale was placed back of the coil and in front of it a screen to cut off the light from the coil; the base of the scale, the top of the coil, and the

top of the screen were at the same elevation. The apparatus was blackened to prevent reflection of light from the coil as far as possible. For the proper proportioning of gas and air, an Oldham gas-testing machine was used. As methane was not available, gas from the city lines was employed. No attention was paid to the composition of the gas or to the percentage present, but the apparatus was arranged to give a mixture approximately equivalent in cap-forming power to certain percentages of methane.

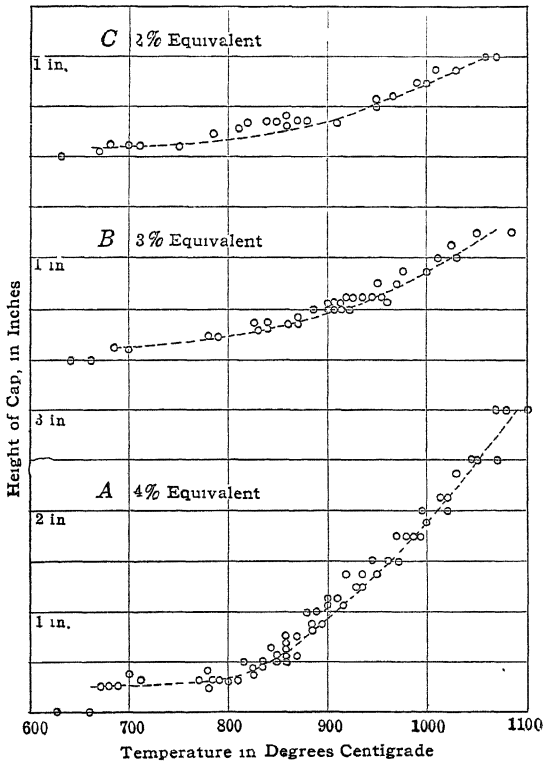


FIG. 2.

The apparatus was calibrated with an unbonneted Clanny lamp and the mixtures used were approximately equivalent in cap-forming power to mixtures of methane and air containing 4, 3, 2, and 1 per cent. of methane. No higher percentage was used because of the approach to the explosive limit. The relations between the temperature of the igniting coil and the height of the cap produced with these mixtures are shown by graphs *A*, *B* and *C*, Fig. 2.

With a gas mixture equivalent to 4 per cent. methane, no cap was observed at 660° C., but a cap about 3/8 in. (9.52 mm.) high appeared at 662° C.; thus roughly indicating this as the ignition temperature of the

gas. The composition of the gas is variable but the following analysis is fairly representative of its composition at the time when the tests were made: $CO_2$, 6.7 per cent.; $O_2$, 0.2 per cent.; Illuminants, 12.1 per cent.; $H_2$, 20.8 per cent.; CO, 8.8 per cent.; $CH_4$, 18.4 per cent.; $N_2$, 33.0 per cent. The ignition temperature of such a mixture is uncertain, but that of methane is commonly given as about 650° C. while that of hydrogen is somewhat lower. It may be judged from the appearance of the cap at 662° that the temperature indication was not seriously in error.

The abrupt appearance of a cap 3/8 in. high at 662° C. is somewhat unexpected, as it might be supposed that the cap would increase gradually from 0. The measurement of the height of the cap was difficult because of the lack of a quite definite summit, but none of many observations with a 4 per cent. methane equivalent showed a cap less than 1/4 in. (6 mm.) high and this small cap appeared and disappeared abruptly with very small temperature changes.

Graph *A* is a composite of the readings of four sets of observation, graph *B* is a composite of two sets, and graph *C* represents only one set. In each case there is a nearly horizontal portion, while the temperature is lower than about 800° C. At these lower temperatures, changes in temperature made very little difference in the height of the cap. The explanation of this fact is not certain but it may be due to the burning of the constituents of the mixture having lower ignition points than methane. It seems quite possible that the heavy hydrocarbons that constitute 12.1 per cent. of the gas would be thus burned. Beginning at about 850°, increase of temperature produced a more marked effect on the height of the cap. This is especially noticeable in the case of the richest mixture.

When an attempt is made to draw curves through the points, it is found that there is some evidence of irregularity in the region between about 825° and 925°, where some points are found lying above the position of a smooth curve. This is shown most plainly in graph *C* but is found also in *B* and *A*. It occurs at a lower temperature in *C* than in *A* while in *B* it occurs at about the same temperature as at *A*. No explanation for this peculiarity is at present apparent.

With gas equivalent to 4 per cent. methane the cap was plain and while low could be measured with some accuracy. High caps ranging above about 1 1/2 in. (38 mm.) were hard to measure because they fluctuated somewhat, and also because the top faded out without having a definite limit.

With gas equivalent to 3 per cent. methane, the cap was less plain than with 4 per cent.; and with a 2 per cent. equivalent the cap was hard to see. Experiment with a mixture equivalent to 1 per cent. methane showed a faint cap, about 1/8 in. (3 mm.) high, when the coil was bright red. Raising the temperature did not apparently increase the height of the

cap, but the cap was so faint that it was obscured by the light from the coil and no measurement was attempted.

The experiments show that there is a fairly definite relation between the temperature of the source of ignition and the height of the cap formed in a mixture of a combustible gas and air, the height of the cap increasing with the temperature. More rigid investigation might reveal a connection between these variables capable of mathematical expression, but the conditions of the experiments made do not warrant an attempt at such expression.

## DISCUSSION

E. B. Wilson, Scranton, Pa. (written discussion*).—Prof. Young's paper shows another application of electricity in solving problems in coal mining, and suggests that it may be possible to utilize the data he has collected in developing an electric firedamp detector.

If electric lamps are to be adopted in gassy coal mines, the use of an electric firedamp detector becomes almost imperative, for safety lamps and electric lamps in the same mine are not always conducive to safety, and Liveings electric firedamp detector is too cumbersome to meet with general approval.

Three well understood facts are verified in a new way by Prof. Young's experiments:

1. The temperature at which methane ignites.

2. The more inflammable gas present in an atmosphere, the hotter will be the flame, and the longer the cap in a safety lamp.

3. As the temperature at the source of ignition varies, the length of the gas-cap flame varies.

Prof. H. B. Dixon collaborating with Mr. H. F. Coward, some years ago, made experiments on the ignition of various gases to find the temperature of their kindling points.[1] From the results of their experiments, the writer has culled the ignition temperatures of some hydrocarbon gases, and has tabulated them with other data, in Table 1.

Table 1.—*Ignition Temperatures of Hydrocarbon Gases Met with in Mines, together with their Heat Units*

| Gas | Formula | Ratio $\frac{C}{H}$ | Temperature of Ignition Degrees C. | B t u. per Cubic Foot |
|---|---|---|---|---|
| Acetylene | $C_2H_2$ | 1 : 1 | 406–440 | 1,555 |
| Ethene | $C_2H_4$ | 1 : 2 | 500–519 | 1,673 |
| Ethane | $C_2H_6$ | 1 : 3 | 520–630 | 1,858 |
| Methane | $C H_4$ | 1 : 4 | 650–750 | 1,065 |
| Hydrogen | H | | 580–590 | 348 |

* Received Aug. 25, 1919

[1] *Jnl.* Chem. Soc., *Trans.* (1909) **95**, **514**.

It will be noticed that the less carbon and the more hydrogen a gas contains the higher will be its temperature of ignition, also that the density of a gas has a direct bearing on the number of heat units evolved by its combustion.

Evidently the wide range between the first and second temperatures is due to the varying conditions accompanying the tests, therefore it appears that Young's curves are quite uniform within the possibilities of temperature differences, even assuming that the composition of the gas varied slightly. It is possible that the nearly horizontal extension of Prof. Young's curves, while the temperatures of the gases were below 800°, was due to the molecules having reached a distance from the source of heat where vibrations failed to increase, and thus limited the transmission of heat, but as the heat intensity increased the vibrations increased sufficiently to extend the flame.

This speculation is based on the assumption that gas, like any other substance, must be raised to the temperature of ignition before it will burn. This being true, then the cap flame which originated at the point of ignition is extended by those molecules of gas burning which have never been in direct contact with the source of heat.

Mr. Force, chief chemist of the Delaware, Lackawanna, & Western R. R., recently made experiments in which mine air containing a small percentage of methane was used under boiler furnaces. He found that so small a quantity as 1 per cent. of methane was able to replace a certain amount of coal for steam raising. If, then, 1 per cent. gas furnished a certain amount of heat, it is reasonable to suppose that 2-per cent. gas would furnish double the amount. Prof. Young's graph follows this line of reasoning when the difference in the mixture of gas is 1 per cent., but when the gas is doubled from 2 to 4 per cent. the height of the flame is trebled, thus showing that the heat conditions about the point of ignition are materially changed.

This offers an explanation why the same lamp with different illuminants gives different heights of caps, and why different types of lamps with the same illuminants give different heights of caps; also why the same types of lamps with the same illuminants give slightly different caps.

An interesting proof of the mechanical mixture of gases is developed by the safety lamp, which under certain conditions will show caps in an atmosphere that contains but $\frac{1}{250}$ part of gas. It would be unreasonable to expect that an atmosphere containing but one part of inflammable gas in two hundred and fifty parts would produce and transmit as much heat as another volume of air that contained one part of gas in one hundred parts, or one part of gas in fifty parts, etc. Evidently, then, the height of the gas cap is not entirely due to the heat at the point of ignition, and some credit must be given to the extension of the flame

through the combustion of the gaseous molecules. Liveings firedamp indicator is based on the comparative brightness of two glowing platinum wires, one enclosed in an airtight vessel and the other exposed to the atmosphere to be tested. Firedamp coming in contact with the exposed red-hot platinum wire raises its temperature, and causes it to glow more brightly. The greater amount of firedamp present, the greater the heat of combustion and the brighter the glow. It is possible that the trebled length of cap flame appearing in Prof. Young's graph is due to the increase of sensible heat which is measured by the thermophile and the combustion of the molecules of gas which have never been in touch with the coil. The sensitiveness of the experimental apparatus is not equal to that of the tri-burner Ashworth lamp, or to the Pieler lamp using alcohol for the initial flame. The former lamp gives a 3-in. flame when 1 per cent. of gas is present; the Pieler lamp furnishes a flame over 3 in. long with 1 per cent. gas, and over 5 in. long with 2 per cent. gas in the atmosphere. Since the platinum metals seem to have an affinity for hydrogen, the use of platinum sponge rather than platinum wire would increase the sensitiveness of the apparatus.

James Ashworth[2] gives the following order of sensitiveness for the fluids used in safety lamps:

1. Benzine, benzolene, colzalene, and naphtha.
2. Petroleum and paraffin.
3. Mixtures of vegetable or fish oils with petroleum.
4. Vegetable and fish oils.

As a broad rule, the nearer a substance is to the gaseous condition, the higher the temperature to which it must be exposed before kindling. Ignition temperatures of some oils are: petroleum, 380° C.; gas oil, 350° C.; benzine, 415° C.; benzole, 520° C.; tar oil, 580° C. (The latter is presumed to be the drip oil from illuminating gas.)

In the same article, Ashworth demonstrated that the larger the lamp flame the longer would be the cap for a given percentage of gas and also that the larger the lamp flame, the more heat is produced. The difference in the heat of the testing flame is shown in three separate sets of caps in a height of 3 in. This was accomplished by altering the height of the flame in an Ashworth-Clowes hydrogen-gas testing lamp, and thus changing the temperature at the source of ignition.

H. G. Davis,* Wilkes-Barre, Pa. (written discussion†).—The only practical method of testing mine air for gas, until recent years, was by the effect of the gas on the flame of the ordinary Davy Lamp, and it is quite surprising how nearly the correct percentage of gas can be determined by many men by observing the behavior of the safety-lamp flame

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[2] *Iron & Coal Tr. Rev.* (1906) **72**, 293, 375.

* Lehigh and Wilkes-Barre Coal Co. † Received Sept. 11, 1919.

in the different mixtures. Mine foremen frequently guess the percentage of gas to within 0.1 per cent, making certain allowances to cover the dust-laden atmosphere, in which the test was made. It is not meant by this that anthracite dust is explosive, but that its properties increase the length of a fuel cap on an ordinary oil-burning lamp, which, as Prof. Young contends, increases in height with the proportion of gas and increased temperature.

Naphtha or benzine-burning lamps have a hotter flame, and, therefore, are more sensitive so that a smaller percentage of gas can be detected. Some of the fire bosses in this region some years ago preferred the Koehler lamp, while others did not care to use it in their morning examinations.

In 1911, when the experimental mine of the Bureau of Mines at Bruceton, Pa., was being prepared to be exploded, a large number of mining men went through it to examine the entries, which were lighted by electricity. While so doing, they saw in the face-return-airway a Koehler or Wolfe lamp, that seemed to be burning with difficulty. An investigation showed that the lamp had been filled to overflowing with naphtha and was generating this surplus gas. This shows how easily we can be deceived so that it is more difficult to decide by observation the percentage of gas contained in the atmosphere by the length of a cap of a naphtha-burning lamp, than of the cap of an ordinary oil-burning lamp.

Professor Young states that the object of his experiment was not to find the exact height of a cap given with a certain percentage of gas at a given temperature of the coil. He also states that with a gas mixture equivalent to 4 per cent. methane no cap was observed at 660° C., but that a cap of about ⅜ in. was observed with a 4 per cent. mixture at a temperature of 662° C. To the practical miner, this means little, as the only method of detecting gas at his command is the flame of the safety lamp, which provides the necessary temperature at all times.

It is interesting, however, to learn from the experiments conducted by Prof. Young that if the temperature of gaseous mines can be kept down the danger from gas explosions is somewhat decreased: the one practical way of keeping down this temperature is by keeping the mine well supplied with an adequate quantity of good fresh air.

The United States Bureau of Mines some time ago prepared a chart on the properties of gases, from which the following is quoted.

| Cap, in Inches | Percentage | Cap, in Inches | Percentage |
|---|---|---|---|
| 0.20 | 1 5 | 0 47 | 3.0 |
| 0.15 | 2 0 | 0.75 | 3.5 |
| 0.35 | 2.5 | 1.15 | 4.0 |

I readily agree with Prof. Young that to determine the heights of gas caps with any degree of accuracy is, indeed, a rather difficult problem,

especially if there is any velocity or movement of the ventilating current. It never occurred to me that temperature was such an important factor in the composition of our mine gases; this, however, is very satisfactorily demonstrated by the paper.

While the writer was superintendent of the D. L. & W. mines, a miner left an ordinary Davy lamp in the face of a gaseous place and, later, on returning to the place, found the burning lamp filled with gas and red hot. He at once ran out of the place and sent word to the surface, as he was on the night shift. The writer and the mine foreman were notified and hurried to the mine expecting to find the place blown up Entering the mine we advanced very cautiously until we reached a point from which the lamp could be observed. We had been told that the 16-ft. canvas in the face had fallen and that a considerable amount of gas had accumulated. This we later found was not true. After a thorough study of the situation, it was suggested that a wet feed bag be wrapped around the burning lamp, which could then be removed to the mouth of the gangway. This was successfully accomplished. When the burlap, which was steaming hot, was unwrapped, it was found that the gauze had been burnt so that when touched it fell to dust: very much like that of a gas mantle.

A fire boss in one of the mines of this region had a similar experience. A miner, whose place was only a short distance from the gangway road, forgot to take down his safety lamp when leaving for home and the driver, after passing through the canvas with the gangway car, forgot to replace the same, with the result that gas quickly gathered near the lamp, which, strange to say, burned until found by the fire boss when he made his morning round. After a study of the situation, he climbed the pitch until he could reach the lamp with his pole, when he carefully removed it. The gauze in this lamp also was found to have been burnt to a white ash.

These cases are cited to show that a safety-lamp gauze will not necessarily pass the flame and ignite the surrounding gas, as we have always been taught. I will admit, however, that these cases are not common. The movement of the lamps themselves, or the surrounding atmosphere would, no doubt, result in passing the flame.

James Ashworth, Livingstone, Alberta, Can. (written discussion*) — About the year 1878, the writer commenced to experiment on safety lamps, the results of which will be found in the Transactions of the North of England Institute of Mining and Mechanical Engineers 1879–1880, the Transactions of the Manchester Geological and Mining Society, and in the technical press. He has also made a careful investigation of the flame caps produced by chemically prepared methane when mixed with air in accurately measured percentages. Before the conclusion of these

* Received Oct. 29, 1919.

experiments he personally fitted the hydrogen gas test to a special pattern of the Ashworth-Gray patent safety lamp.

Later, as pure hydrogen gas was not always available, the writer constructed a safety lamp that would separately burn oil and alcohol; and still later, a somewhat similar lamp to burn alcohol only for specially testing low percentages of methane. This lamp was known as the Ashworth alcohol safety lamp.

The Ashworth alcohol lamp, having a conical glass 4 in. high, gives just room enough for the cap from 1 per cent. of methane, which is 3 in. high above the cone over the wick flame. The cap for ½ per cent. of methane is 1¾ in. high, and for ¼ per cent. 1¼ in. high. Each of these caps is well defined.

The sketch of the apparatus used by Mr. Young does not show that he adopted any means to bring the methane mixture into close contact with his eight coils of platinum wire. It would, therefore, appear to the writer that in this respect his tests were not as accurate as they might have been and that this fact will account for the non-uniformity in the height of the caps produced. Those who are acquainted with the caps produced by the Ashworth-Clowes hydrogen flame know that the tips of the flame caps are very distinct and that the tips of the caps form a straight and not a curved line, due to the increasing percentages of methane. This is, in either case, due to the increase in the temperature of the methane flame resulting from the increase in the volume of combustible gas.

In the Ashworth-Clowes, hydrogen, gas-testing, safety lamp the influence of the temperature of the testing flame is made use of by making two scales of flame caps. The flame for testing percentages above 1 per cent. may be made with a 10-mm. hydrogen flame cap and those under 1 per cent. with a flame 15 mm. high; in this way the cap produced by ¼ per cent. of methane can be easily seen.

Mr. Young states that he "knows of no previous attempt to correlate the changes of the temperature with change of the height of the cap produced." In this regard it is evident that the use of two or more heights of the hydrogen flame in the Ashworth-Clowes lamp effects this in a simple and practical way, but without registering the actual heat of the flame; although the heat of each size of flame may become a standard one, and in practice is so. The writer regrets that he cannot see the novelty claimed by Mr. Young; viz., that he has discovered that there is a fairly definite relation between the temperature of the source of ignition and the height of the cap formed in a mixture of a combustible gas and air, the height of the cap increasing with the temperature. In this regard the height of the 10-mm. hydrogen flame, say, for 1 per cent. of methane with the height of the cap produced by a 15-mm. flame may be compared.

If Mr. Young were to use a conical glass, blackened (smoked) over its inner surface behind the testing flame, and with a chimney a few inches high on the top of it, he would find the tip of his flame caps sharply defined and easy to measure. All safety lamps used by fire-bosses are greatly improved in their firedamp indications if, say, one-third of the inner surface of the glass part is either colored a dead black, or is smoked so as to deaden the reflection on the usual bright surface.

# Engineering Features of Modern Large Coal Mines in Illinois and Indiana

BY C. A. HERBERT,* E. M., VINCENNES, IND., AND C. M. YOUNG,† E. M., URBANA, ILL

(Chicago Meeting, September, 1919)

WITHIN the past few years, considerable development has been made in the coal-mining industry in Illinois and Indiana and it is the purpose of the authors to record its most important phases. Perhaps the two most striking features are the entry into the producing fields of certain large consumers of coal and the magnitude of some of the new operations. Mines are now being equipped by the Chicago & Northwestern R. R., the Chicago, Burlington & Quincy R. R., the Standard Oil Co., of Indiana, and the Union Electric Light & Power Co. of St. Louis. Besides, the U. S. Steel Corpn. has increased its coal-mining activities by commencing operations on its large holdings northeast of Benton, Franklin Co., Ill.

Until recent years, the supply of coal has commonly been adequate and prices have generally been favorable, especially to large consumers, as the magnitude of their purchases enabled them to obtain satisfactory quotations from producers. In fact, most consumers of coal have believed that they could buy their fuel more satisfactorily in the open market than they could produce it themselves. The entry of the United States into the war was accompanied by various disarrangements of industrial conditions, and a demand for coal in excess of the supply seemed likely to be experienced for an indefinite period. Under these circumstances it seemed that the greatest assurance of a supply of fuel was its production by the operation of mines. In the case of the Chicago & Northwestern R. R. and the U. S. Steel Corpn. the production of coal has been carried on for many years with satisfactory results, but the C. B. & Q. R. R., the Union Electric Light & Power Co. and the Standard Oil Co. of Indiana had not been producers of coal, at least in this district.

Apparently the most impressive feature of the newest mines is their capacity, for in some cases it is planned that production shall be in the neighborhood of 1000 tons per hr. There are mines now in this field having nearly that output, but these were not planned for such output; in fact, when the shafts were sunk and the machinery installed it was generally supposed that the daily outputs would be only in the neighborhood of 4000 to 5000 tons. Incidentally, there has risen

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an interesting rivalry among some of the mines of large output. For a few years, the record for a day's output from a shaft mine in the bituminous-coal districts of the United States, and probably in the world, was held alternately by the No. 3 mine of the Superior Coal Co., at Gillespie, and the No. 1 mine of the New Staunton Coal Co. at Livingston. Subsequently the contest passed to the south and the record was held alternately by the Orient mine of the Chicago, Wilmington & Franklin Coal Co. at Orient, Ill., and the No. 1 mine of the American Coal Mining Co., at Bicknell, Ind. While the output of the latter mine is generally above 5000 tons per day, the record for a day's output is now held by the former mine, at which there was hoisted, on Mar. 6, 1919, 6776.75 tons in 1501 hoists.

## Mines Considered

No attempt has been made to chronicle any but the most important phases of recent events. Many new mines have been sunk and equipped in the past few years but these have not been discussed unless some new features of engineering practice were involved. Six mines have been selected as exhibiting the most striking recent developments. There is also given a short discussion of the safety work of the Old Ben Coal Corpn. The six selected mines are the No. 2 mine of the Standard Oil Co., the No. 4 mine of the Superior Coal Co., the Kathleen mine of the Union Colliery Co., the No. 2 mine of the Bell & Zoller Mining Co., the Valier mine of the Valier Coal Co., and the No. 2 mine of the American Coal Mining Co. These mines are located as follows:

Standard Oil Co. of Indiana, Mine No. 2, at Schoper, 8 mi. northeast of Carlinville, Macoupin Co., Ill.; Superior No. 4, 7 mi. southwest of Gillespie, Macoupin Co., Ill.; Kathleen mine of the Union Colliery Co., 5 mi. south of DuQuoin, Perry Co., Ill.; Bell & Zoller No. 2, 1½ mi. southwest of Zeigler, Franklin Co., Ill.; Valier mine of the Valier Coal Co., 3 mi. north of Christopher, Franklin Co., Ill.; No. 2 mine of the American Coal Mining Co., 2½ mi. southeast of Bicknell, Ind.

The mines are the property of companies unrelated to each other. In each case the plans have been made by engineers of experience in coal mining and of perfect familiarity with the conditions to be met. These mines, therefore, embody the best knowledge and experience obtainable and represent the highest type of coal mine engineering in the district at the present time.

The Standard Oil Co. of Indiana is a new operator. To provide fuel for its refineries at Wood River, Kansas City, and Whiting, which together consume about 7000 tons of coal every day of the year, the company purchased the old mine of the Carlinville Coal Co. at Carlinville, with about 2300 acres of coal lands, and also about 22,000 acres of undeveloped coal lands lying to the north and east of Carlinville.

The Superior Coal Co. is a subsidiary company of the Chicago & Northwestern Railroad. It first entered the Illinois coal fields about 14 years ago and already had three mines in operation near Gillespie, the combined output being about 12,000 tons a day. The coal rights of the company include nearly 45,000 acres in the Gillespie field.

The Union Colliery Co. is a subsidiary company of the Union Electric Light & Power Co. of St. Louis, which in turn is a subsidiary company of the North American Co. The latter company is already interested in coal mining in Ohio and Kentucky. Coal from the Kathleen mine will be supplied to power plants in St. Louis and Milwaukee and to the open market.

The Bell & Zoller Mining Co. is a producer and marketer of coal for steam and domestic use. The company has mines at Centralia and Zeigler, the latter mine being the largest and best known.

The Valier Coal Co. is a subsidiary company of the Chicago, Burlington & Quincy R. R. and its coal will go entirely to that road.

The American Coal Mining Co. of Indiana is, like the Bell & Zoller Mining Co., a producer and marketer of coal for steam and domestic use. It already owns and operates its No. 1 mine which is one of the most productive mines in the bituminous-coal fields of the country.

TABLE 1.—*Data of Mines Discussed*

| Name of Operator | Name of Mine | Approximate Output Expected per Day, in Tons | Thickness of Coal, in Feet | Depth of Hoisting Shaft to Bottom of Coal, in Feet | Inside Dimensions of Hoisting Shaft, in Feet | Inside Dimensions of Air Shaft, in Feet |
|---|---|---|---|---|---|---|
| Standard Oil Co. of Indiana | No 2 (Schoper) | 7000 | 6 to 8 | 317 | 7 by 17 | 14 by 31 |
| Superior Coal Co | No 4 | 7000 | average 7⅛ | 313 | 11 by 21 | 11 by 17 |
| Union Colliery Co | Kathleen | 7000 to 8000 | average 8¼ | 261 | 11 by 19 | 12½ by 26⅓ |
| Bell & Zoller Mining Co | No. 2 | 6000 | average nearly 11 | 310 | 12 by 21⅔ | 12⅙ by 22⅚ |
| Valier Coal Co | Valier | 7000 to 8000 | 9½ to 12⅙ | 605 | 11 by 18$\frac{11}{12}$ | 13 by 30 |
| American Coal Mining Co | No 2 | 7000 | average 6½ | 248 | 12 by 20 | 12 by 18 |

## COAL

In the case of each of the Illinois mines considered, the coal developed is the well-known No. 6, which is the principal coal worked south of Springfield. The characteristics of this coal are not the same at all of the mines considered.

The two mines lying farthest north, Standard Oil No. 2 and Superior

No. 4, are in coal that, at the former, varies from 6 to 8 ft. in thickness and at the latter averages 7 ft. 1½ in. with about the same limits. In this northern part of the district, the coal is nearly horizontal and there is no reason for anticipating serious trouble from grades, as the mines already developed have not encountered serious trouble from this cause. As far as the experience in the district indicates, there will be no trouble from gas, at least at the Standard mine, but the Superior No. 4 approaches so close to the Staunton natural-gas field that inconvenience from gas is possible. The roof conditions in this district are, in general, very good and it is the practice of the Superior Coal Co. to drive its entries and crosscuts 21 ft. (6 4 m.) wide in order to avoid payment for narrow work. In some cases this is not possible and the present development of the No. 4 mine indicates that the entries in part of the mine cannot be made more than from 15 to 18 ft. wide.

The Kathleen mine is situated near the bottom of the monoclinal fold, commonly known as the DuQuoin anticline, which passes a little east of north across the eastern side of Perry County. The shaft of the Security mine on the west side of the fold is 90 ft. (27 m.) deep, while those of the Paradise and Majestic mines on the east side are respectively 365 and 409 ft. (111 and 124 m.). Heretofore operations on the steep part of the slope have been avoided, though there have been some small workings.

After considerable exploratory work the Union Colliery Co. decided to develop a property so situated that about one-third of the coal will be taken from the plateau on the west side of the fold, where the depth is about 90 ft., about one-third from the slope of the fold, and about one-third from the east side where the average depth is about 250 ft. (76 m.). With such a topography of the coal bed, the location of the shaft required careful consideration. Since the coal east of the steepest part of the fold continues to dip slightly to the east, the only position of the shaft that would have allowed a general down-grade from west to east would have been on or near the east boundry of the plot. This, however, would have required the development of a one-sided mine and the surface conditions would have been unfavorable. Study of conditions led to the location of the shaft as far down the slope of the monocline as the surface conditions would permit. Development underground has shown that the bottom of the monocline was not reached but that the steep grade, approximately 5 per cent., extends for a short distance to the east of the shaft. The larger part of the coal will travel down grade but some will have to be hauled up an adverse grade. The grade on this monocline has been found to average from 5.2 to 5.3 per cent. but is not constant, appearing rather as a series of steps in which flatter and steeper parts alternate. It will be necessary to use mechanical means to control the movement of the cars approaching the shaft when uncoupled from the

locomotive. The average thickness of all sections of the coal thus far made at the Kathleen mine is 8 ft. 3½ in. (2.5 m.), the thickest coal being on the eastern side of the monocline. It is not expected that much water will be encountered as the other mines in the vicinity have no trouble, at least unless the roof is broken by the removal of coal. Little trouble from gas has been experienced in the neighboring mines.

In the case of the Bell & Zoller mine No. 2, the No. 6 coal is reached a little south of the famous Zeigler mine, which was the first opened in Franklin County and the one in which the thickest coal is found. It is now thought that the coal in the No. 2 mine will not have as great an average thickness as that at Zeigler, where it is close to 11 ft. (3.3 m.) and where as much as 15 ft. is found in places and the average thickness taken out is about 8½ ft. The coal at the new mine will, however, be thicker than most of the other coal of Illinois and a thickness of 8½ ft. can be mined as at Zeigler. The depth at the Bell & Zoller mine No. 2 is 310 ft. (94 m.) and the dip to the north is indicated by a depth of 450 ft. (137 m.) at the Zeigler mine of the same company about 2 mi. north of No. 2.

At the Valier mine, the depth to the bottom of the coal in the main shaft is 605 ft. (184 m.), this being one of the deep mines in the state. The shaft was located with an idea of avoiding an unusually thick layer of water-bearing material. Success was attained in this respect but the topography of the coal bed at the shaft bottom was found to be somewhat rough and considerable grading will be necessary. The thickness of the coal is somewhat variable, ranging from 9 ft. 6 in. to 12 ft. 2 in. (2.8 to 3.7 m.) so far as known. It is expected that a thickness of 8½ ft. (2.5 m.) will be taken out in the rooms. The mine gives off considerable gas, as do all of the mines in this part of the district. Electric lamps are used exclusively.

The coal being developed at the American No. 2 is the No. 5 bed of the Indiana series. The coal varies considerably in thickness in various parts of the field, averaging about 6½ ft. (1.9 m.) at the No. 2 mine. The coal bed lies generally horizontal, but some trouble is experienced from local hills and swamps. The coal contains many impurities and is not well suited to domestic purposes; it is a most excellent steam coal and for this purpose finds its most ready market. The No. 5 bed has little or no water and has an excellent roof, but gives off large quantities of methane.

## Shafts

The high yield of coal planned for these mines has necessitated the sinking of shafts not only large enough to accommodate the necessary hoisting but to transmit the large volumes of air required for ventilation when the mines become fully developed. At the Standard mine, the

main shaft is 7 by 17 ft. (2 by 5 m.) inside and 317 ft. (96 m.) deep. The lining is 3 by 10 in. (7.6 by 25 cm.) timber laid flat above the solid rock, and 3 by 5 in. timber laid flat through all the rock, nailed with very heavy spikes. The lining inside the timber will be of reinforced concrete 17 in. thick from top to bottom.

At all of the mines the concrete was mixed on the surface and lowered through pipes to the bottom, and in all cases, but one, the concreting was done from the bottom upward. At the Standard, the concrete was distributed from the bottom of the pipe to the periphery of the shaft through a flexible pipe similar to those used for handling grain, while at the Valier mine a rubber hose was used. At the Kathleen mine the concrete was lowered through a 6-in. (15-cm.) flanged-joint pipe and discharged into a metal telegraph, or baffle, to which was attached a flexible metallic elbow from which the concrete was carried to the forms through wooden chutes. At the Superior No. 4, wooden chutes, floored with metal under the pipe, were used for distributing the concrete.

Various kinds of shaft forms were used at these mines. Those at the Standard consisted of planks laid on edge and bound by corner pieces devised by the engineers for this work. The buntons are of concrete, poured in position as the lining was brought up. The forms for the lining at the Superior No. 4 were made of planks set on end and held by horizontal strips of 2 by 6-in. timber (5 by 15-cm.) bolted together at the corners. A joint in the middle of each timber permitted the form to fold in at the middle when the braces were removed. At the Kathleen mine, the forms were made of 2 by 8 shiplap fastened vertically to horizontal studdings held in place by a framework of 3 by 3-in. angle irons bolted at the corners and braced by easily removable 4 by 4-in. buntons. Five or six sets of forms were kept in use, the lower sets being removed and used at the top after each pouring.

The air shaft of the Standard No. 2 is probably the largest shaft in cross-sectional area in the coal fields of the United States, being 19 by 36 ft. (5.7 by 10.9 m.) outside dimensions at the surface, and 14 by 31 ft. (4.2 by 9.4 m.) inside. It is also timbered in the same way as the main shaft and is lined in the same way. This shaft has four compartments: air, stairway, main cage, and counterweight.

The use of a counterweight cage in the airshaft has been adopted at two of the Illinois mines considered—Standard and Valier. In each case the main cage is large, that of the Standard being of sufficient size to accommodate a 15-ton locomotive without disassembling, and the one at Valier being of approximately the same size. In each of these cases the counterweight cage is small and has two decks; the number of men hoisted on the two cages will be the same. At the Valier mine 30 men will be hoisted per trip, while the main cage at Standard No. 2 will accommodate 45 men, allowing over 2 sq. ft. of area per man, though the number hoisted

per trip will probably be less. In each case counterweight cars will be used on the small cage when heavy loads are handled on the main cage.

At the Superior No. 4, the main shaft is 313 ft. (95 m.) deep and is 11 by 21 ft. (3.3 by 6 4 m.) inside. The lining is of reinforced concrete 2½ ft. (0.75 m.) thick for 100 ft. (30 m.) and from 15 to 18 in. (38 to 45 cm.) thick for the remainder of the way. The buntons are 8-in. (20-cm.) I-beams riveted to 30 ft. (9 m.) lengths of channel iron. These were fabricated on the surface and lowered into slots left in the lining to receive them, later being wedged into alinement and anchored with concrete. At this mine 65-lb. (29 kg.) railroad rails are used for guides. The air shaft is 11 by 17 ft. (3.3. by 8.2 m.) inside with the same lining as the main shaft. This shaft has only two compartments, airway and stairway, separated by a solid 12-in. reinforced-concrete partition. No hoisting is done at the air shaft. The stairway at this shaft is different from any other in the state. The landings, of concrete reinforced with triangular wire mesh, were cast in place as the lining was cast. The stringers and treads, also of reinforced concrete, were cast on the surface and lowered into position; the stringers were put into place and the treads slipped into notches left for them. Both shafts were concreted in sections as the sinking progressed. At the bottom of the air shaft is a deflector of concrete so curved as to direct the air in both directions along the airway with the least eddying.

At the Kathleen mine, the main shaft is 261 ft. (79.5 m.) deep to the bottom of the coal and is 11 by 19 ft. 11 in. (3.3. by 5.9 m.) inside. The lining is of concrete approximately 1 ft. thick. The buntons are of 6-in. (15-cm.) 23.8-lb. (10.5-kg.) H-beams set on 5-ft. centers. Guides are 85-lb. (38-kg.) steel rails. The air shaft is 230 ft. deep, 12½ by 26⅓ ft. inside, and the lining like that of the main shaft is 12-in. reinforced concrete. Both shafts were temporarily lined with 3 by 12-in. pine curbing. The shaft is divided into four compartments—hoisting compartment 9 by 12½ ft., counterweight compartment 3 by 12½ ft., stairway compartment 4 by 12½ ft., and air compartment 8 by 12½ ft. The air compartment is separated from the remainder of the shaft by a 12-in. reinforced-concrete partition. In this mine only one cage is used, but this has two decks each of which will accommodate 25 men. The counterweight is of concrete with a scrap-iron aggregate. The buntons in the air shaft are 9-in., 21-lb. I-beams. The hoisting capacity of the air shaft is about 800 tons of coal in 8 hours.

At the Bell and Zoller mine No. 2, the hoisting shaft is 12 by 21 ft. 8 in. (5.4 by 9.9 m.) inside. The lining is a wooden cribbing of 6 by 12-in. (15 by 30-cm.) timber laid on the sides for a depth of 60 ft. (18 m.) and on edge for the remainder of the depth. The upper 60 ft. has a lining of 6 in. of reinforced concrete and the remainder has 2 in. of gunite with

wire-mesh reinforcement. There are three rows of 6 by 12-in. wooden buntons with 8 by 10-in. yellow-pine guides. The air shaft is 12 ft. 2 in. by 22 ft. 10 in. and is divided into three compartments, a stairway 3 ft. 2 in. by 12 ft. 2 in. an air compartment 10 ft. 7 in. by 12 ft. 2 in. and a cage compartment 8 ft 1 in. by 12 ft. 2 in. The partition between the air and hoisting compartments will consist of 6 in. of gunite and the buntons in both shafts will be coated with gunite to make them fireproof in compliance with the Illinois law. The counterweight for the air-shaft cage runs on wire-rope guides in the air compartment.

At the Valier mine, the main shaft is 605 ft. (184 m.) deep to the bottom of the coal with a 52-ft. (15.8-m.) sump below this. The inside dimensions are 11 by 18 ft. 1 in. (3.3 by 5.5 m.). The lining down to the rock, a depth of 30 ft. (9 m.) is 24 in. (61 cm.) of reinforced concrete and for the remainder of the way is from 12 to 15 in. thick. The reinforcing consists of ¾-in. (19-mm.) bars set on 6-in. (15-cm.) centers vertically and 1-ft. (30-cm.) centers horizontally. The concrete mixture was 1 to 4. The buntons are reinforced concrete cast on the surface and lowered into place and anchored with concrete. The air shaft is 13 by 30 ft. (3.9 by 9 m.) inside and has a lining similar to that of the main shaft. This shaft has four compartments—airway 9 ft. 9 in. by 13 ft., stairway 4 ft. 3 in. by 13 ft., cage-way 8 ft. 4 in. by 13 ft., and counterweight compartment 4 ft. 4 in. by 13 ft. The air compartment is separated from the other compartments by 12 in. of reinforced concrete. The guides are 8 by 10 in. wood. The excavation for the air shaft amounted to 12,000 cubic yards.

The American No. 2 hoisting shaft is 12 by 20 ft. (3.6 by 6 m.) inside dimensions and is 248 ft. (75.5 m.) in depth. From the surface to the rock, a distance of 52 ft. (15.8 m.), it is lined with reinforced concrete, the balance of the distance with 4 by 10-in. (10 by 25-cm.) yellow pine placed on edge. There are three rows of buntons; steel I-beams through the concrete, and below this 6 by 10 in. wood. The guides are 6 by 10-in. yellow pine. The air shaft is 12 by 18 ft. (3.6 by 5.4 m.) inside dimensions, with two compartments, air and cage. The counterweight for the cage runs in guides in the air compartment and is made up of sections or blocks of cast iron so designed that it may be made heavier or lighter by the addition or removal of these sections. The lining of the air shaft is similar to that of the hoisting shaft.

## Hoisting

In the mines developed according to the common practice of the present day in this district, the output is limited, first, by the capacity of the shaft or tipple and, second, by the layout of the underground workings. Both of these features have been given careful attention in the new developments under consideration.

Two lines have been followed in the increase of the hoisting capacity of the shafts. In three of the mines considered, capacity is being increased by the use of large mine cars and of hoisting appliances adequate to handle these cars, whereby the number of hoists necessary for large output is decreased somewhat by increasing the amount of coal hoisted at each trip. In the case of the other mines the coal is hoisted in skips. As the skips hold about two mine-car loads of coal the hoisting speed can be reduced by about one-half.

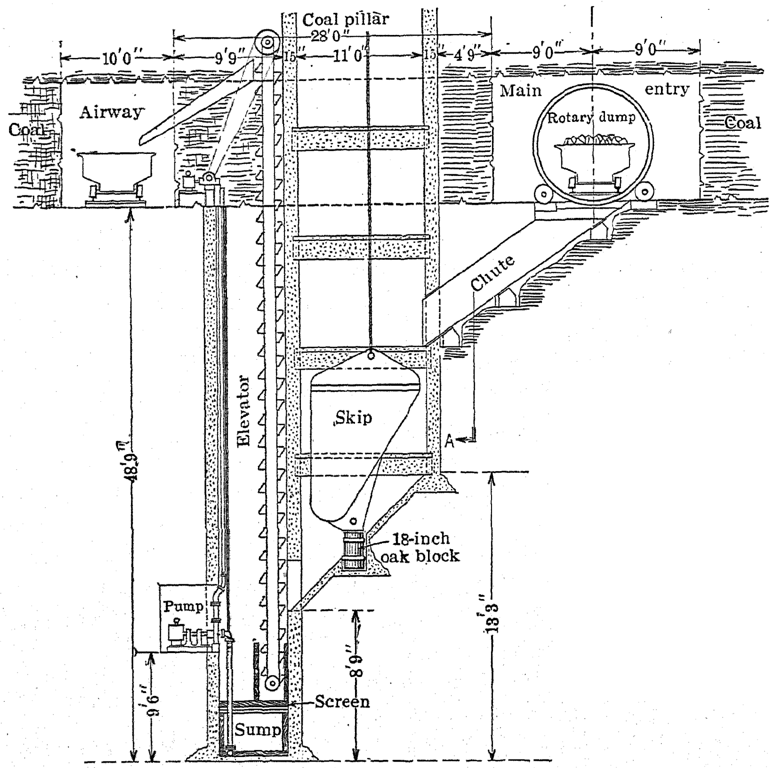


FIG. 1.—ELEVATION OF SHAFT BOTTOM AT VALIER.

Another very considerable advantage of hoisting in skips is the fact that tight cars can be used. A large part of the trouble encountered in handling coal in cars is due to the end doors necessary when the car is hoisted in a self-dumping cage. Moreover, such cars are always more or less leaky and a considerable amount of coal is scattered along the haulage roads, where part of it is ground into dust, part of which is suspended in the air and part deposited on the ribs, adding considerably to the chance of explosion.

The principal advantage of hoisting in cars is that the product of each miner is kept separate from that of other miners until it has reached

the surface where it can be inspected for impurities. When coal is dumped from the mine cars into skips at the bottom of the shaft, this last advantage is sacrificed to the advantages of lower hoisting speed and smaller number of hoists per minute. It is this advantage of hoisting in cars which has determined the retention of this system in the three prominent cases mentioned.

At three of the mines, the Superior No. 4, the Bell & Zoller No. 2, and the American No. 2, the old method of hoisting coal in end-dump cars on self-dumping cages is followed, while at the other mines the coal is hoisted in skips. The No. 1, or Zeigler, mine of the Bell & Zoller Mining Co., is equipped with skips, but these were installed by the predecessors of the present operating company before this method of hoisting coal had reached its present development. This company has not adopted skips in the new mine.

Apparently there are no new developments as far as the use of cages is concerned, except in connection with the size of cars, which, at all three mines where cages are used, hold approximately 5 tons. No new arrangements were required at the shaft bottoms in connection with these, though all have been planned to make the rapid movement of cars easy. Where skips are used the main shaft is set at the side of the entry. In all cases the cars are handled in rotary dumps, the coal going into a chute from which it passes either directly into the skip or into a hopper. The former arrangement is used at the Kathleen and Valier mines, see Fig. 1. In each of these cases two cars are to be dumped at the same time and the trip is to be passed through the rotary dump without uncoupling. In each case the chute branches and coal is directed to the proper side by a vane.

At the Valier mine, all operations in connection with dumping and hoisting will be performed by one man at the bottom. The position of the vane in the chute and of the skips will be indicated by lights. The bottom man, observing which skip is at the bottom, will direct the vane into the proper position to discharge coal into the skip. These positions are indicated by the simultaneous lighting of two lamps in the same column. The rotary dump is then operated and the hoist started. The operation of the hoist is to be entirely automatic, only the starting being performed by the bottom man.

At the Standard No. 2, two rotary dumps are set side by side, see Fig. 2. From the cars, the coal goes into a hopper of, approximately, 40 tons capacity and cars can be dumped without regard to the position of the skip. In this case the cars are uncoupled and dumped singly. From this storage hopper the coal will go to a measuring hopper holding one skip load of about 12 tons, then into the skip, as soon as the latter arrives at the shaft bottom. As the empty skip descends, it will close the discharge gate of the storage hopper and open that of the measuring

hopper. As it ascends it will close the gate of the measuring hopper and open that of the storage hopper. The chute from the storage hopper is self-sealing, the gate being used only to prevent coal from passing into

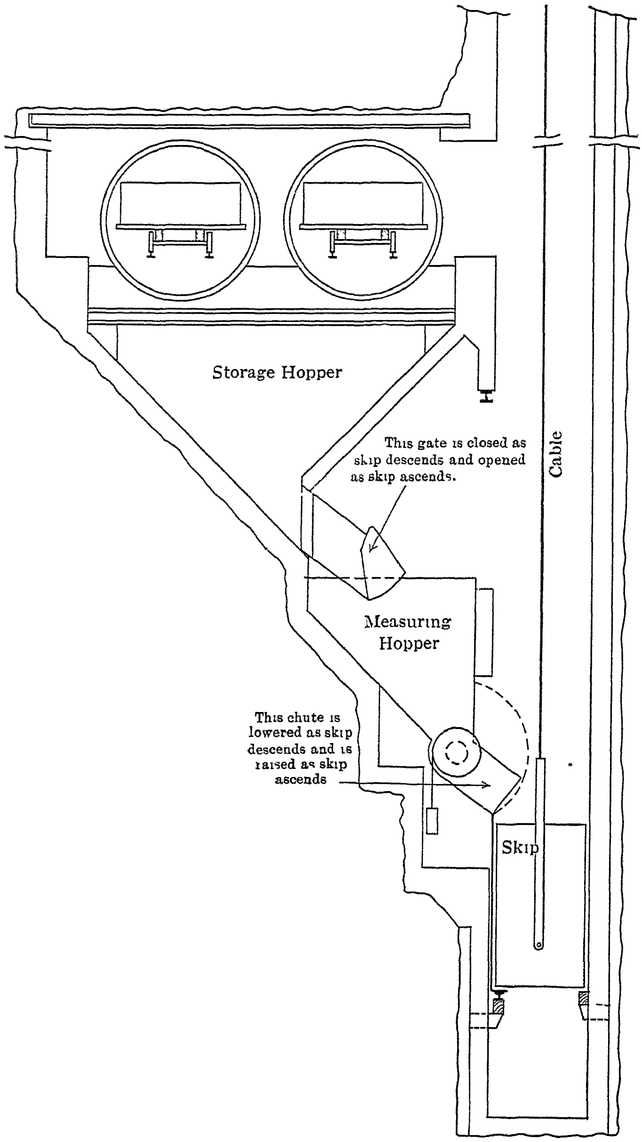


FIG. 2.—SHAFT BOTTOM DUMPING ARRANGEMENT, STANDARD OIL CO.

the measuring hopper during the loading of the skip. The amount discharged into this hopper can be somewhat varied by raising and lowering a dam in the upper part of the hopper, in this way changing the point at

which the accumulation of coal seals the chute. The breakage resulting from this method of loading is not objectionable as all coal from this mine is to be crushed.

At the Kathleen and Valier mines the skips have curved bottoms, while at the Standard No. 2 the bottoms are square.

Various means have been adopted for cleaning the sumps. At the Valier mine an extra compartment is excavated at the side of the shaft to the depth of about 10 ft. (3 m.) below the shaft bottom. An elevator in this compartment will raise the spilled coal and load it into cars.

At the Standard No. 2, it is thought that the method of handling coal from the hoppers will largely prevent the spilling of coal into the sump and that little besides dust will accumulate. For the removal of this material a square box is fitted in the sump. When this is full it will be attached to the bottom of the skip and raised high enough to be dumped into one of the hoppers.

At the two Illinois mines at which coal is to be hoisted in cars, steam-hoisting engines will be used, while an electric hoist will be used at the Indiana mine. These engines have cylinders 28 by 42 in. (71 by 106 cm.) and 8-ft. cylindrical drums. The hoisting equipments of these two mines are nearly the same, as are the depths of the shafts and the loads to be raised.

At the other three Illinois mines, electric hoists will be used. The load is about 9 tons at the Kathleen and Valier and about 12 tons at the Standard. The most important difference in conditions is the depth of the mines, the Valier shaft being 605 ft. (184 m.) deep, while the Standard No. 2 is 317 ft. (96 m.), and the Kathleen 261 ft. (79 m.). These depths are from the shaft collar to the bottom of the coal and approximately 100 ft. (30 m.) will be added in each case by the depth of sump and height to dump circle.

The air-shaft hoist at the Standard No. 2 will run at a rope speed of 600 ft. (182 m.) per min. and the main shaft hoist at 900 ft. per min. The hoisting cycle at the main shaft will be 48 sec. and it is expected that the average amount of coal hoisted will be 20 tons per minute. At this mine, the hoists at both main and air shafts are geared hoists operated by alternating-current slip-ring motors with liquid-rheostat control on the secondary. These motors will run on 2200-volt current. The main-hoist motor is 900 hp. and the air-shaft motor 250 hp.

At the Valier mine, the maximum rope speed of the air-shaft hoist will be about 700 ft. (213 m.) per min., and that of the main hoist about 1600 ft. (487 m.) per min. The hoisting cycle at the main shaft will be 35 sec. for starting, accelerating, hoisting, and stopping, and 8 sec. for loading.

The main hoist at the American No. 2 has a 7 to 10-ft. (2 to 3-m.) combined cylindrical and conical drum, directly connected to an 800-

hp., 500-volt, direct-current motor that receives its current from a flywheel motor-generator set operated by a 500-hp., 2200-volt, motor, driving a 750-kw., 500-volt direct-current generator. It is expected to make four hoists per minute. In addition to the 800-hp., direct-current motor, the drum is connected at the opposite end through herring-bone gears and clutch to a 350-hp., 2200-volt, alternating-current motor, which will be used at night and on idle days to avoid the necessity of using the flywheel set for intermittent operation. The air-shaft hoist is driven through clutch and gear by a 250-hp., 2200-volt, alternating-current motor at one end, and at the other by a double 10 by 12 geared steam engine. Should the electric power fail, it will be possible to operate the air-shaft hoist by steam in order to take the men out of the mine. The fan, which is ordinarily motor driven, is also equipped with an auxiliary steam engine to be used in case of failure in electric power. Steam for these auxiliaries and for heating the surface buildings will be furnished by a 150-hp. boiler.

At the Kathleen mine, the combined cylindrical and conical drum is driven through a Francke flexible coupling by a 600-kw., direct-current motor, with full voltage speed of 235 r.p.m. Current will be supplied to this hoist motor at 500 volt by a flywheel motor-generator set having a 500-hp., alternating-current induction motor, taking current at 2200 volt and running at 900 r.p.m. The direct-current 500-kw. generator is separately excited. The flywheel weighs 20,000 lb. (9071 kg.). This set is equipped with a speed-limit switch. The hoist is equipped with air-operated brake, a Royer & Zweibel over-winding device and mechanical slow-down. The air-shaft hoist is operated by a geared slip-ring induction motor operating on 2200-volt, 60-cycle, three-phase current, and having Cutler-Hammer reversible magnetic control.

At the Valier mine, the main hoist will be driven by a direct-connected, direct-current motor of 1350 hp. A cylindrical drum is used, 9 ft. (2.7 m.) in diameter, and the full speed will be 55 r.p.m., giving a rope speed of approximately 1600 ft. per min. Current will be supplied by a motor-generator set having an 1100-hp. motor and a generator capacity of 1000 kw. with voltage of 0 to 600. This set will run at 720 r.p.m. The flywheel weighs 33,000 lb. (14,949 kg.). The air-shaft hoist will be operated by a geared alternating-current motor.

## Tipples

The construction of the tipples shows only one striking novelty, the concrete air-shaft tipple of the Kathleen mine, shown in Fig. 3. The use of concrete for the construction of tipples is not in itself a novelty, as it has been used in some other districts. In this case concrete was

adopted, not because a concrete structure was desired, but because at the time of designing it was doubtful whether steel could be obtained. In spite of the fact that a portion of the concrete was poured when the thermometer registered about 20° below zero, the work is sound and the structure is peculiarly attractive. Moreover, it is perfectly rigid, no vibration whatever being felt when the hoist is running. The cost of

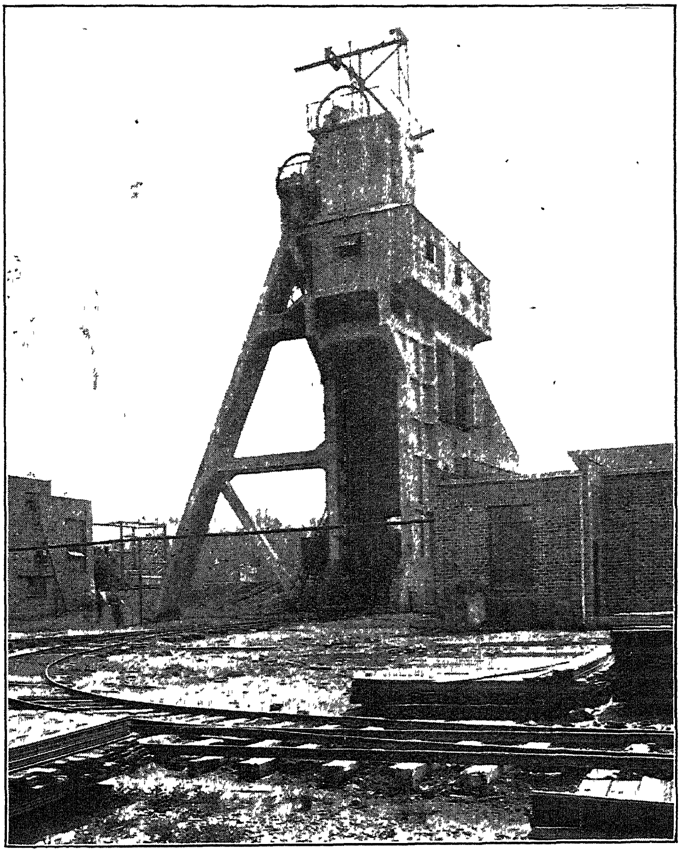

FIG. 3.—CONCRETE TIPPLE AT AIR SHAFT, KATHLEEN MINE.

the concrete tipple, as built, was about $6000 higher than the cost of the steel tipple originally intended. A considerable part of this excess was due to the expense of heating the materials and of pouring the concrete during severely cold weather. At this mine and at the Valier, cars hoisted at the air-shaft will be dumped into Wood single-car rotary dumps. The main-shaft tipple of Superior No. 4 mine is shown in Fig. 4.

Attention may also be drawn to the fact that of the mines mentioned, four have tipples of the three-leg type introduced by Allen & Gracia.

Two of these have bar screens, and the other shaking screens. The tipple at the Bell & Zoller No. 2 was designed and built by the Wisconsin Bridge & Iron Co. The tipple at the Standard No. 2 mine was designed by R. W. Hunt & Co.

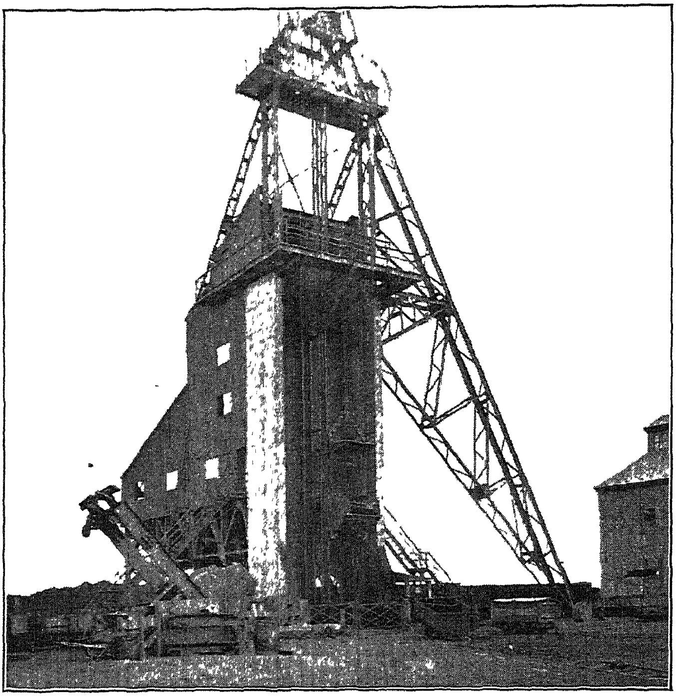

Fig. 4.—Main shaft tipple, Superior No. 4.

## Preparation

The equipment for preparation at the six mines considered is designed to satisfy the needs of the different producers.

*Standard No.* 2.—At the Standard No. 2 mine all coal is to be crushed to from 1¼ to 1½ in. (31.75 to 38 mm.) and the only screening equipment needed is that preceding the crusher for by-passing the small coal. The coal will go from the skips to a receiving bin from which it will pass over two parallel conveying and crushing equipments, each designed to handle 500 tons per hr. The coal passes from the receiving bin to two 54-in. (137-cm.) feeder conveyors, from which it goes over sloping bar screens, the oversize going to crushers from which it drops to conveyors, being joined there by the undersize of the screens. Two 60-in. apron conveyors elevate the coal to the middle of the line of loading bins, which are built in two lines over the two loading tracks and extend for a linear distance of 360 ft. Their capacity is 3500 tons. Each apron conveyor

discharges onto a 72-in. flat-top conveyor, one running from the middle to each end of the line of bins. Adjustable scrapers are used to distribute the crushed coal into the bins, from which it will be discharged into the cars.

*Superior No. 4.*—In point of simplicity of preparation equipment, the Superior No. 4 comes next. The preparation depends on the kind of coal demanded. When coal is being stored, all coal over 2 in. (5 cm.) or over 6 in. goes to storage. The material under 6 in. may be shipped as mine run. The material under 2 in. goes to the washer When coal is not being stored, it may be crushed to pass a 6-in. screen. The screening equipment is designed to meet this simple demand and consists only of gravity bar screens. Material to be crushed is carried by a reciprocating feeder to the crusher.

*Valier.*—At the Valier mine shaker screens with pendulum suspension are used and are so arranged as to make four sizes of coal: lump, egg, 2½-in. screenings, and a fine coal approximately corresponding to No. 5. The two larger sizes are run over picking tables and may either be loaded into cars or fed to a crusher and reduced to pass through a 2½ in. screen. The plant is thus equipped to furnish coal for either hand-fired locomotives or those equipped with mechanical stokers.

*American No. 2.*—As the coal from this mine will be sold almost entirely to steam trade, the screening equipment is very similar to that of the Superior No. 4. The coal will be run over bar screens so arranged as to make either 1¼-in. (31.75-mm.) screenings and standard lump, 1¼-in. screenings and railroad lump, mine run, or the entire output may be run through crushers and made into 1½-in. screenings. Picking tables are being installed for both the mine run and the lump coal.

*Kathleen.*—At this mine the demand will be variable, as part of the coal will go to the St. Louis and Milwaukee plants of the North American Co., which consume 500,000 tons of screenings per year. The remainder of the coal is to be marketed and the tipple is equipped for making lump, 3 by 6-in. (7.6 by 15-cm.) egg, 2 by 3-in. nut, and 2-in. screenings. There are picking tables for the nut, egg, and lump coal, with loading booms for lowering the coal into the cars. Provision has also been made for the installation of crushers so that either the lump or the egg coal may be crushed to screenings if the market conditions make this desirable. Shaker screens are used with pendulum suspension. At this mine and at Valier, the crushers are so placed that the coal may be discharged into them by elevating the loading boom.

*Bell & Zoller No. 2.*—The preparation equipment at this mine is more elaborate than that at any of the other mines. The main screen will produce three sizes of coal; lump, 6-in. egg and 3-in. screenings. The egg and lump sizes from the main screen will be passed over picking tables and discharged over loading booms. Provision is also made for

the installation of crushers to crush either the lump or egg sizes. Means are also provided for loading lump coal into box cars on a separate track, in case these should be available when coal cars are not. The screenings will be elevated by a belt conveyor to a rescreening plant equipped with two sets of screens in parallel, supported on ash boards, which will produce coal of the five standard Illinois sizes; viz., Nos. 1, 2, 3, 4, and 5. The screens are of such length as to permit hand picking if this should be necessary. The rescreening plant is set over the loading tracks and the coal will be discharged from the screens to the bins, from which it will be loaded.

The chutes from these bins are equipped with lip screens for removing the fines produced by handling.

*U. S. Fuel Co.*—In connection with preparation, it should be added that the U. S. Fuel Co., as subsidiary of the United States Steel Corpn., is developing a coal property lying to the northeast of Benton, Franklin Co., Ill. All the coal produced is to be washed, so a large concrete washer has been erected and equipped with jigs, tables, and settling tanks. This is by far the largest coal washer in the central coal field and its equipment has been planned with great care. The company is not yet ready for a discussion of its operation.

## Surface Yards

The system of handling cars on the surface by gravity, common in this district, has been followed at most of the mines. This system sometimes requires extensive grading in a level country and the moving of cars is difficult in winter. The surface layout for the Bell & Zoller No. 2 mine is shown in Fig. 5.

At the Kathleen mine, the nature of the surface is such that the construction of a gravity yard would have required an unusual amount of grading, as the surface at the mine is practically on a level with the main line of the Illinois Central from which the cars will be taken. This difficulty has been avoided by the use of a flat yard, the only grade being that extending from the air shaft on the west, past the main shaft and railroad scales on the east, where there is sufficient grade for the movement of the cars being loaded. Over the remainder of the yard, cars will be moved by a pusher locomotive, running on a 36-in. track between the two main tracks. The pusher locomotive will have arms that can be extended to either side and engage the ends of the cars. A 15-ton locomotive is now being used for this purpose. A 25-ton locomotive will be provided later to move the loaded cars while the 15-ton locomotive will be employed for the empty cars. Each locomotive will be able to handle about ten cars.

At the Standard Mine No. 2, the problem is entirely different as far as the loading of cars is concerned. At all mines in the district having

the ordinary preparation equipment, only one car is loaded at a time on each track. At the Standard No. 2, all the coal is to be crushed and placed in hoppers from which it will be drawn directly into the railroad cars. Only one size of coal will be shipped and only two loading tracks

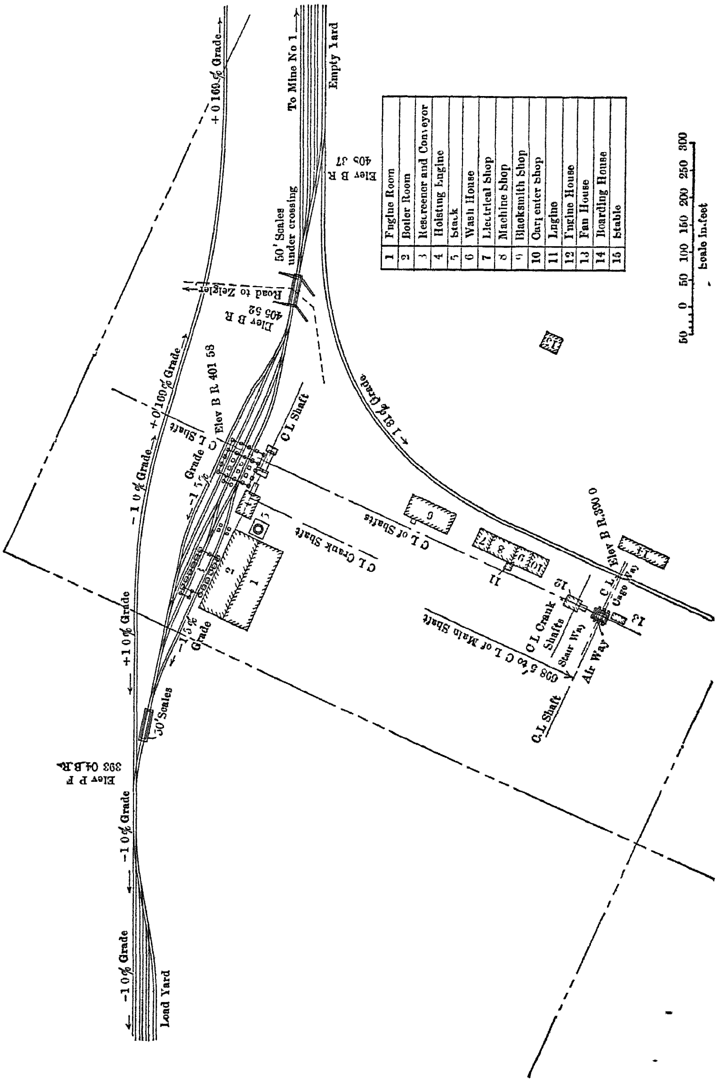


FIG. 5.—SURFACE LAYOUT, BELL & ZOLLER NO. 2 MINE.

will be used. They will extend for the length of about eight railroad cars, over the two loading tracks, and cars will be loaded in groups of eight. Car pulls will be used to allow the spotting of cars of variable lengths under the loading chutes. Under these circumstances the simplest arrangement is to employ a switching engine. The track arrangement

differs from that of the ordinary mine in that there is no provision for the collection of loaded cars beyond the tipple. The empty cars will be brought in from the west to the loading hoppers and after being loaded will be taken back toward the west to be made up into trains.

## Power

At three of the mines, Superior No. 4, Standard No. 2, and Bell & Zoller No. 2, power plants have been erected. The Superior No. 4 will be equipped with five vertical water-tube boilers of 513 hp. which will supply steam at 150-lb. (68-kg.) pressure and 150° superheat. There are two turbo-generators, one having a capacity of 938 kv.-a. and one of 375 kv.-a. both generating current at 2300 volts. All steam piping from the boilers to the engine is in duplicate, so that any interruption of service in the steam lines is guarded against.

At the Standard No. 2, a central power plant is being erected at which four 750-hp. Vogt boilers have been installed. These are expected to produce from 1000–1200 hp. apiece and it is probable that two more boilers will be added in the future. The power plant contains two 2500-kw. turbo-generators, and one 1250-kw. generator both supplying current at 6600 volts. The surplus power will be transmitted to other plants. One of the precautions against interrupted transmission from this plant to the mine is the construction of a double transmission line from the power plant to the main shaft.

The three other mines are supplied by power purchased from public-service corporations. The American No. 2, in case of protracted failure of the power supply, can be supplied by the power plant of the No. 1 mine, which can be enlarged sufficiently to supply both mines.

In the case of the Kathleen mine, it would be possible within 90 days to build a power line from one of the generating stations of the Union Electric Light & Power Co., at St. Louis.

## Mining

During the past 25 years, the method of mining in the central field has been changed from the old room-and-pillar system with long room entries, which in many instances were used as main haulage roads, to what is known as the panel system. In this latter system, no rooms are turned from the haulage and main air-course entries, but these are protected on either side by large barrier pillars. The room entries turned off from these main entries are comparatively short, averaging from 1000 to 1500 ft. (304 to 457 m.) in length. In the typical panel system of mining, each panel, consisting of two room entries with their respective rooms, is a unit in itself connected with the other workings only where the two-room entries pierce the barrier pillar.

This method of mining has come into general use in the central field, principally because of the prevalence of mine fires and the great difficulty of sealing them off under the old system of mining, it being necessary to seal off a large part of the mine in order to control the fire. It is now possible to isolate a fire in any one panel by seals at the points where the room entries pierce the barrier pillar, without the loss of a great deal of territory and at the same time, because the area is small, the fire is usually smothered completely within 3 or 4 weeks. This system of mining is being followed at all the new mines in this field with such variations as to the length and width of the rooms, thickness of pillars, and number of rooms in each panel, as are made necessary by local conditions.

At the Valier mine an attempt will be made to make a complete recovery of the pillar coal in the panels. With this one exception the room pillars will vary in thickness with the depth of the seam and will be only thick enough to allow rooms to be driven to full length without squeezing. The barrier pillars are of sufficient thickness to protect the main entry from any subsequent squeezing. The dimensions of rooms and relative dimensions of rooms and pillars are, however, subject to change and it is not improbable that steps will ultimately be taken for the recovery of pillar coal at more of the mines.

At the Valier mine it is planned that the rooms shall be 25 ft. (7.6 m.) wide on 85-ft. (25.9-m.) centers, leaving 60-ft. (18-m.) room pillars, which will be later recovered. By driving the cross-cuts between the rooms 20 ft. in width, 40 per cent. of the coal will be taken out in the advance, which is nearly as much as the total extraction at the other mines in this field. The room pillars will be left in blocks 60 ft. square, which may be recovered later without the squeezing that would be certain to take place should an attempt be made to recover the narrow room pillars used at the other mines. The ultimate extraction is expected to be about 70 per cent.

The sinking of these new mines with large daily capacities and expected to work out larger areas than ever were contemplated in the past has led to the necessity for driving a greater number of main airways. Until recently two entries in the mines in this field had been thought sufficient.

At the Bell & Zoller No. 2, it is proposed to use the four-entry system throughout the mine for all of the main haulage and airways, with double entries in the panels. At the Valier and at the Superior No. 4, four main entries will be driven, the cross-entries, or secondary mains, from which the room entries are turned, will be on the three-entry system, with two entries in the panels. At the Kathleen mine, the three-entry system will be used for the mains, with two-entry system for the secondary mains and panel entries. At the American mine No. 2, the three-entry system will be used for both the main and secondary main entries with two entries in the panels.

The four-entry system has the advantage of making it possible to put the required amount of air into the mine at a much lower pressure and velocity than where fewer entries are used, and with a marked saving in power required to operate the fan. Also where the entries are driven in pairs with no crosscut between the two central entries, except at points where cross or room entries are turned off, the necessity of expensive brick or concrete stoppings is eliminated, as the air in both entries of

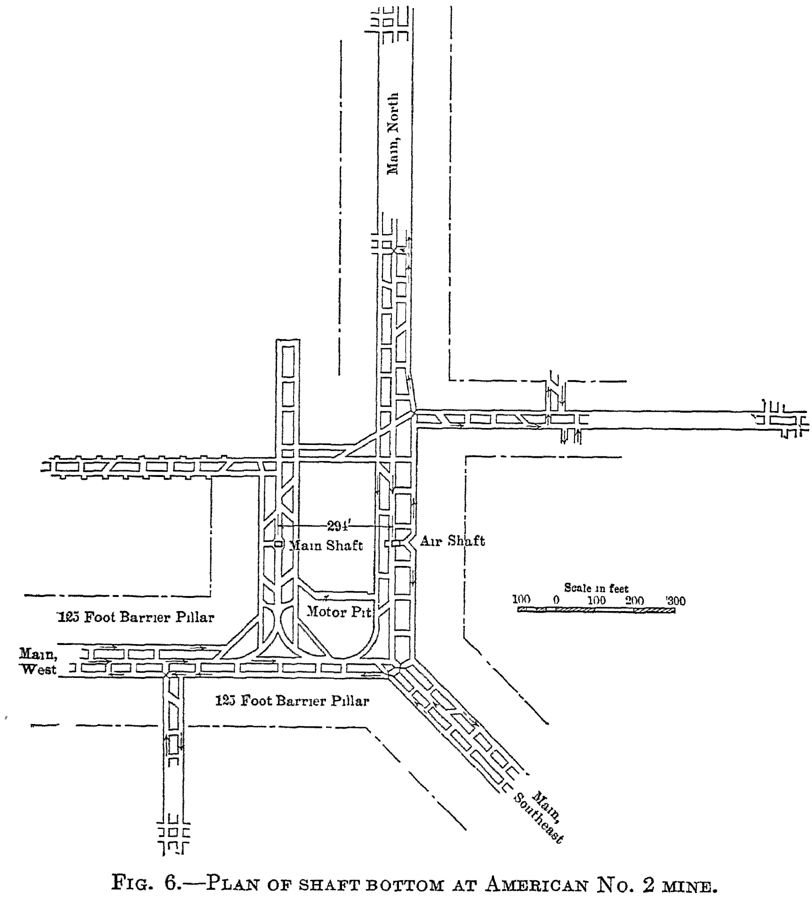


FIG. 6.—PLAN OF SHAFT BOTTOM AT AMERICAN NO. 2 MINE.

each pair travels in the same direction. Moreover, in case of an explosion the recovery work can be carried on much more rapidly and economically, as there will be no delay nor expense in rebuilding stoppings to restore ventilation.

The shortwall mining machine has practically superseded the breast machine at all these new developments. Alternating-current mining

machines will be used at four of the six new larger developments in this field. In every case, it is proposed to take 2300-volt three-phase alternating current into the mine in armored cables to motor-generator sets located at advantageous points in the mine, there to be transformed to 250-volt direct current for haulage locomotives, and for mining machines also where alternating current machines are used for cutting.

At the American mine No. 1, where 240-volt alternating-current mining machines have been in use for several years, the mining-machine transformers are supplied with 240-volt and 275-volt secondary taps, so that when the room entries have advanced sufficiently to make a drop in voltage noticeable at the machines the machine lines are connected to the 275-volt taps This same system will be used in the No. 2 mine.

With the advent of the 5-ton mine car, it has been found that the 6-ton gathering locomotive, which had been almost a standard in this field, is too light and the gathering locomotives at these mines will weigh from 7.5 to 8 tons. At the Valier mine and at the Standard No. 2, combined storage-battery and trolley-gathering locomotives will be used, while at the other mines the trolley-and-reel type of gathering locomotives will be used.

The use of heavier cars and locomotives has also made it necessary to use heavier track with curves of longer radius. In fact, the underground mine tracks in these new mines will approximate very closely, as far as the class of construction is concerned, to the standards of the modern steam or electric railway.

The bottom layout at the American No. 2, shown in Fig. 6, Bell & Zoller No. 2, and Superior No. 4 are all very much alike. At each of these mines self-dumping cages are used. At each what is known as the A type of bottom has been adopted, in which the coal is brought in from either side of the mine to a double-track shaft-bottom entry, at right angles to the main entry.

The mine car, after being automatically fed on to the cage, hoisted, and dumped is returned to the bottom, where it is pushed off the cage by the oncoming loaded car. After leaving the cage, the cars pass down, by gravity, to a kickback switch and are shunted off to either side into entries forming the sides of the A, where they are made up into the empty trips to be taken back inside. Where the natural conditions make it necessary, a car haul is put in between the cage and the kickback switch in order to give sufficient grade to run the cars back into the empty-car storage entries. There are usually a number of cut-throughs between the main bottom and the entries on either side to enable the locomotive coming in with the loaded trip to uncouple at a convenient point and run through to the waiting trip of empty cars.

The bottom layout at both the Kathleen and Valier mines is very

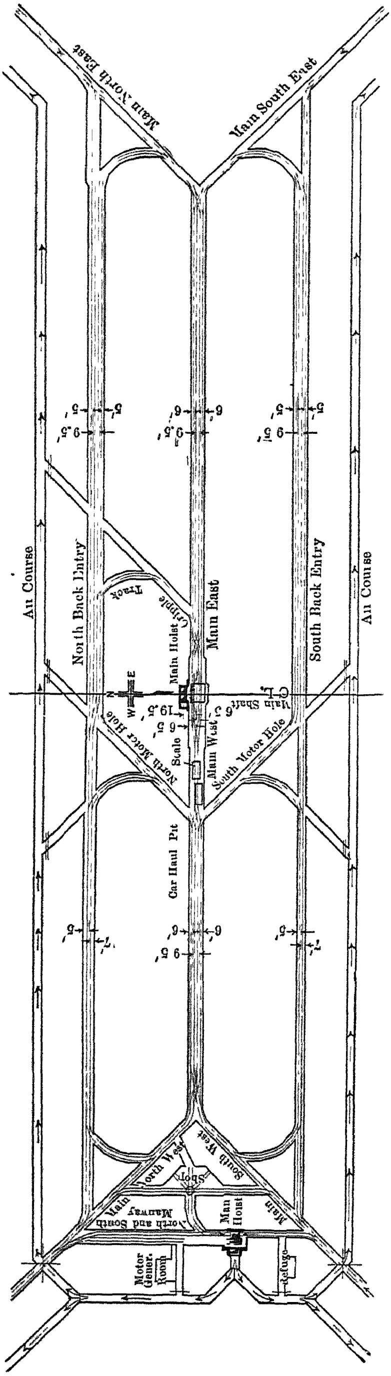


Fig. 7.—Bottom layout, Standard Oil Co. shaft.

similar to that just described except that the cars are run in over a single track to be weighed and dumped and that provision has been made to handle the empty cars in trips instead of singly, as will be done where cages are used. At the Valier mine, the cars will be handled in trips of ten. After leaving the dump, these trips will run a short distance to the foot of a rather steep incline up which they will be hoisted by a single-drum electric hoist. From the top of this incline, the trips will be shunted off to the entries on either side of the main shaft bottom. This method of handling empty cars was made necessary by the natural grades that were encountered. At the Kathleen mine the empties will leave the dump in trips of twenty cars and run by gravity to empty storage entries.

The bottom layout at the Standard No. 2 is rather more elaborate than at any of the others. The mine is divided into four districts, and the coal will be brought to the shaft bottom over four main haulage roads. The bottom is so laid out that the cars from these four roads will be brought to a long double-track bottom entry. An automatic car haul will be used to feed the cars over the automatic scales and into the rotary dump. The movement and dumping of the cars will be controlled by one man. At this mine, contrary to the general custom, the cars will be turned end for end at each trip to the bottom; a new and ingenious coupling has been provided for each end.

*Entry-driving Machines.*—At the Valier mine, entry-driving machines have been introduced to accomplish rapid development work, promote safety through elimination of explosives and prevent the shattering action of explosives on the ribs and roof of the entry. Two B-34 machines, made by the Jeffrey Mfg. Co., have been used. The cutting in this mine is unusually hard, but under ordinary conditions an advance of as much as 150 ft. (45 m.) per wk. was made with each machine, by working three shifts. On single shifts these machines are now making a general average of 300 ft. per mo. With regard to the economy of the use of these machines, it is the opinion of Mr. Carl Scholz, general manager of the company, that the cost of installation and operation will be repaid many times by the saving in timbering because the coal is not affected by the use of explosives when these machines are used. The roof will stand much better than it does when so shattered and timbering will be unnecessary in most parts of the entries. These machines are being used on the main west entries where a possible distance of 3½ mi. can be driven. After an experience of about 1 year in entry driving in this mine, a difference between the standing qualities of these entries and those driven with explosives can be observed.

## Safety Work of the Old Ben Corporation

This company has undertaken a more elaborate program for accident prevention than is being followed by any other coal company in this district. This program follows three lines; in the first place it was decided that only electric safety lamps should be allowed in the future in the No. 11 mine in which an explosion had occurred, and that no one should be allowed to smoke in the mine or even carry matches. At first this ruling met with some opposition but at the present time this has entirely disappeared and the mine is being satisfactorily operated with electric lamps.

In the second place a safety engineer was employed, whose work is a general inspection of the mines of the company with special regard to the prevention of accidents, and the recommendation of changes to be made and policies to be followed. In the pursuit of this work, the safety engineer, Mr. J. E. Jones, formerly state mine inspector, is making a close study of all accidents and their causes, giving attention not only to the immediate physical causes of these accidents but to the classes of labor in which they are most common. Records are also kept covering the accident rate under different headings at the different mines of the company. In addition to this, careful study will be made of all the fatal accidents in the Franklin County field.

The third line followed is the introduction of dust barriers. The general plan promulgated by the U. S. Bureau of Mines was followed, but the details were worked out by the company staff. These barriers are

of two types; a concentrated barrier for the haulage entries and a trough barrier for the airways. The concentrated barrier is commonly known as the Jones barrier, but represents the combined efforts of the company staff.

The Jones barrier consists essentially of a box about 2 ft. deep and extending across the entry, the dimensions being varied somewhat to

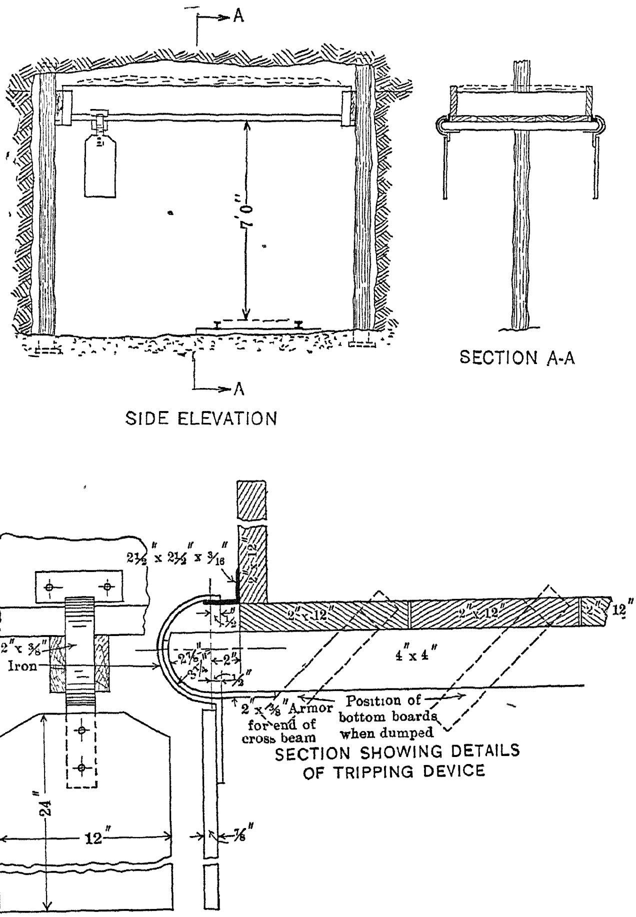


Fig. 8.—Jones dust barrier.

suit its capacity to the width of the entry. The bottom of the box consists of a number of planks running crosswise of the entry, 10 to 12 in. wide, and notched out at the ends so that only a few inches of one edge of the planks rests on the framework of the box proper. When unsupported from underneath, the planks drop down at an angle of about 45° allowing the dust to pour out in a number of thin streams. In their normal position when loaded with dust, these bottom planks are held in position

by cross-pieces, which extend beyond the sides of the box on either side and are held in position by supporting irons. The ends of the cross-piece are cut to a half circle and fit into a semi-circular curve in the supporting irons, and are so adjusted that a slight outward pull on the supporting irons releases the cross-pieces and allows the bottom planks to tilt downward, discharging the rock dust. To the bottom end of each supporting iron is attached a vane about 12 in. (30 cm.) wide and 24 in. long. These vanes will be caught by the advancing wave of an explosion, moving the supporting irons sufficiently to allow the cross-pieces to fall. It is planned also that other vanes shall be placed at some distance from the barrier along the entry and connected with the supporting irons, thus providing for a somewhat earlier operation of the barrier.

As the Jones barrier is up close to the roof of the entry and offers no obstruction either before or after being discharged it is thought to be a marked improvement over the Rice barrier, which when discharged drops into the entry sufficiently to offer a dangerous obstruction to locomotives or cars, in case of accidental discharge.

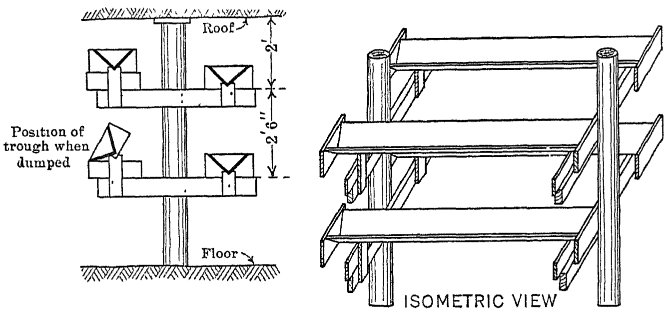


FIG. 9.—TROUGH DUST BARRIER.

The trough barrier, Fig. 9, consists of V-shaped troughs loaded with shale dust, supported on arms attached to posts set in the entry. Each pair of posts carries four troughs set at different elevations. The supporting posts are so arranged that the sets of troughs are staggered along the entry, thus providing a winding path for men to travel but protecting the entire cross-area of the entry. The troughs are made of 8-in. (20-cm.) boards nailed together at right angles and are supported in notches 1 in. deep. With this arrangement they will be easily overturned by any strong blast of air; moreover, the shale dust is heaped up in the troughs so that a considerable amount will be blown off, even by a blast of air too feeble to overturn the troughs.

The company has installed a crusher in which shale from the mines is reduced in size so that 75 per cent. will pass a 250-mesh screen, 92 per cent. a 200-mesh screen, and 95 per cent. a 150-mesh screen. The dust is removed from the crusher by a current of air and only that fine enough

to be raised is used; 94 per cent. of this dust is ash and the remainder mostly moisture.

## Mine Towns

The proper housing of mine labor has always been a serious problem, and one that has been too much neglected until recent years. This field is fortunate in the fact that most mines are situated near established towns in which men can find homes.

There has been only one development in mine housing of sufficient importance to warrant inclusion in a discussion such as this: This is the provision of residences in connection with the work of the Standard Oil Co. of Indiana. Though the No. 2 mine is about 8 mi. from Carlinville, it was decided that no town should be built at the mine, but that houses should be erected in an addition to the town of Carlinville, thus giving the miners the benefit of city schools, lights, water, and sewer systems.

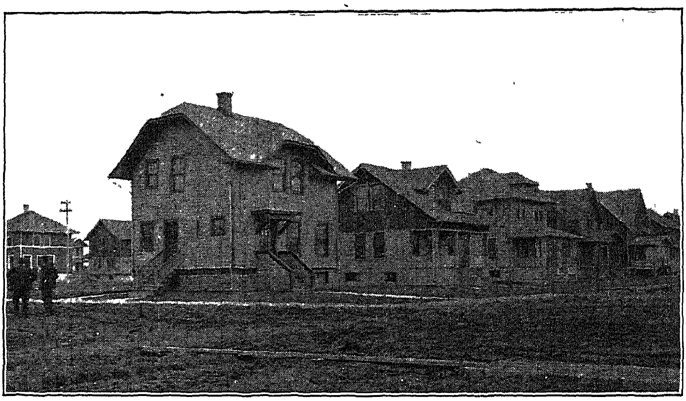

Fig. 10.—Houses at Standard No. 2 mine.

The houses were erected by Sears, Roebuck & Co., of Chicago. One hundred and fifty-six houses were erected in Carlinville, and twelve more at the mine for the accommodation of those employees who must necessarily be constantly at the mine. The plans and materials for these houses were taken from the regular stock of the company, illustrated in their catalog. In order to diversify the dwellings as far as possible, fourteen different floor plans were used and these were modified by different arrangements of porches and roofs so that there are forty-five or fifty different appearances. The difference of design is farther accentuated by painting in different colors, see Fig. 10. The cost of these houses was nearly one-half million dollars not including grading, sidewalks, and plumbing. This addition to the town is provided with sewers, the lots are graded, the sidewalks are in place, and the trees have been

planted to the value of about $50 per lot. This new addition to Carlinville is so different from the ordinary mine town that no connection between the mines and the dwellings would be suspected.

The company expects to operate the mine constantly and desires to attract labor of steady habits and a desire to work, in short, such class of labor as will desire to live in the kind of houses erected and become a permanent part of the population. It is not the intention that these houses shall be rented. The whole undertaking simply represents a desire on the part of the operating company to so house its labor as to promote contentment rather than unrest and attract a class of steady industrious labor.

## Summary

The development of coal mining in the Illinois-Indiana district has been rapid during recent years, and especially since the increased demand for coal from this district incident to the war. An important feature of this development is the production of coal by large consumers.

While many of the new mines present advances in engineering practice, six have been selected for discussion as exemplifying the best practice in coal-mine engineering in the central district at the present time.

These six mines are planned for outputs of 6000 to 8000 tons per 8-hr. day. To make this large output possible, all of these mines are so laid out as to provide an ample number of working places, airways are sufficient for good ventilation, shaft bottoms are laid out for rapid and easy movement of cars, and hoisting equipment is of large capacity.

Two plans have been followed to permit rapid hoisting. At three mines coal is hoisted in end-dump cars of 4.5 to 5 tons capacity; at the other three, coal is hoisted in skips. The skip load at one mine is 9 tons, at another 9.5 tons, and at the third 12 tons.

Steam hoists are used at two of the six mines and electric hoists at four. Power plants have been erected at three mines and electric power is purchased from public service sources at three.

The Valier Coal Co. is the first in this district to lay out a mine with the view to recovery of a large percentage of the pillar coal.

The Old Ben Coal Corpn. has adopted a program of accident prevention involving the employment of a safety engineer. This company has also devised an improved form of dust barrier for limiting the extent of explosions.

The Standard Oil Co. of Indiana has erected a power house at its No. 2 mine equipped with machinery to generate about five times the amount of electric power needed to operate the mine. The No. 11 mine of the Old Ben Coal Corpn. also has a power generating equipment in excess of its needs but on a smaller scale than that at the Standard.

The Standard Oil Co. of Indiana has taken a new step by the erection

of dwellings probably of a better type than have been provided by any other coal-mining company in this country. These are not to be rented but sold at cost plus interest.

Flat surface yards are used at two mines; Kathleen and Standard No. 2. The first has pusher locomotives running on a narrow-gage track between the two loading tracks. The second has a switch engine, and cars are moved and loaded in groups of eight.

Preparation varies from simple bar screens to a re-screening equipment for the production of five sizes of small coal. At the Standard No. 2 all coal will be crushed.

The following statements are in addition to the summary of the preceding article; the Old Ben Coal Corpn. is sinking its No. 15 mine, which will be electrically operated by power from the No. 11 mine. The same company uses pusher locomotives to expedite the movement of cars at the shaft bottom and decrease the amount of labor required.

The Superior Coal Co. has provided a new type of wash-house at the No. 4 mine. Each miner has both hanger and locker, but these are at opposite ends of the building and separated by the shower-bath room, thus preventing contact between clean and dirty clothes.

The Equitable Coal & Coke Co. is recovering pillar coal at its Majestic mine at DuQuoin. Wide room pillars are left and these are split from the crosscuts. At the same mine an emergency stop has been devised for the hoist, by which the bottom man can almost instantly stop the hoist.

The Old Ben Coal Corpn. is making a more complete recovery of barrier pillar coal at its No. 8 mine than has been accomplished elsewhere in the central district.

Practically all mines in the district are provided with track scales on which the loaded cars are weighed.

Storage at the mines has been developed to a considerable extent. The methods vary from simple piling on the ground to piling and repiling with a steam shovel with clam-shell bucket. The latter equipment has been used at the Old Ben Coal Corpn.

Recent progress in mining points to an increase of the percentage of coal extracted. In conclusion it may be stated that present developments show application of the highest engineering skill and thereby indicate the realization of the necessity of engineering in coal production and foreshadow growth of the functions of the coal-mine engineer.

## DISCUSSION

Eugene McAuliffe, St. Louis, Mo. (written discussion*).—When we undertook the development of the Kathleen mine, near DuQuoin,

*Received Sept. 27, 1919.

certain features greatly influenced the construction and underground development. The country is very flat, so that the usual gravity yard movement of empty and loaded coal cars would entail a very heavy fill at the empty-car end of the tracks, with some form of car-pulling arrangement to move the loads where it was practically impossible to provide a gravity movement. So the yard was made level, except for 1000 ft. (304 m.) under the two tipples, where the cars are moved by gravity and controlled under the main-shaft tipple by four Fairmont car retarders. Both tipples are spread sufficiently to admit of introducing a narrow-gage track and locomotive. One 18-ton locomotive is at present employed in moving empties down to a point above the main-shaft tipple and the loads off the track scales down into the storage yard; ultimately a 25-ton locomotive will be employed on the loaded side of the tipple. This form of car movement and control will admit of handling empty and loaded cars promptly in winter weather without the employment of a large number of men, as is commonly practised.

The main-shaft tipple was designed to include ample screen area with picking tables and loading booms on the nut, egg, and lump tracks, insuring the best possible dry cleaning of the screened product. The control of all machinery in the tipple is in the hands of one operator, centrally located. A number of push-button controls conveniently located at different points in the tipple enable any employee to throw off the power instantaneously.

In designing the underground layout, including pit cars with roller-bearing wheels and a capacity of approximately 5 tons, together with a two-car rotary dump, due attention was given to the matter of reducing to a minimum the number of employees required to handle the mine bottom. One operator, centrally located at the shaft bottom, handles the movement of the loaded trip over two pairs of pit car scales, placed tandem, through the rotary dump, thence to the empty-car tracks. All the cars remain coupled throughout; that is, each trip is coupled to the one preceding it, making a continuous train passing through the rotary dump, sufficient empty cars being cut off below the rotary dump to meet the requirements of the outbound empty trip. This reduces the labor of coupling and uncoupling at the shaft bottom to a minimum. The rotary dump and its attendant mechanism, including the trip control, are handled by one operator through the medium of compressed air. It was our idea to reduce as far as practicable the number of men employed at other than coal loading, using the largest possible transportation unit, reducing transportation costs. It was thought that a high hourly hoisting capacity would be desirable in view of a possible reduction of hours per working day.

The matter of providing houses for mine employees was given very serious consideration. The decision finally reached was that the coal

company would confine its effort to insuring the sale of building lots and the construction of houses for its employees under terms that were fair and reasonable, with ample provision for time payments either made to a townsite company separately organized, in which the mining company has no financial interest, or through a building and loan association.

The Illinois mining law requires the construction and maintenance of wash houses for employees. In designing these, an attempt was made to insure absolute cleanliness. Two steel lockers were provided for each employee, one for pit clothes and one for street clothes; also provision for drying damp clothes, so as to insure as far as possible the absence of disagreeable odors so commonly experienced in miner's wash houses. No provision for washing other than through the medium of a shower bath was provided, with the result that 95 per cent. of the employees fully bathe and change their clothes before leaving the wash house. No seats whatever are provided in the locker room, which discourages loafing there.

A. G. Reese,* Cleveland, Ohio (written discussion†).—As the hoists and skips mentioned were installed by the Wellman-Seaver-Morgan Co., some of the principal features of these hoists and skips may be of interest.

All the rotating parts of both hoists are of steel, the drums having machine-turned grooves. All the bearings are of the ring oiling type, the pinion-shaft bearings of the skip hoist being provided with removable shells. Each hoist is provided with a parallel acting post brake; the brake beams are of structural steel and the supporting links, operating levers, and connections are of cast steel and forgings. Both hoists are driven through one reduction of herring-bone gears enclosed in an oil-tight housing of structural steel, supported on the hoist bed plate. The brakes are applied by gravity and released by cylinders operated by compressed air at 90 lb. (39.7 kg.) pressure, the air being supplied by two electrically driven air compressors, each having a piston displacement of 72 cu. ft. (2 cu. m.) of free air per minute. The hoist houses are located about 300 ft. (91 m.) apart with one compressor and receiver in each hoist house, the receivers being connected by piping, which arrangement allows the brakes of both hoists to be operated from either one of the compressors, the other compressor being used as a spare unit.

The cage hoist is at present hoisting 1000 tons of coal per 8 hr., using only one cage, the load being partly balanced by a counterweight of 18,000 lb. The total load on the cage rope is 24,380 lb., which is divided as follows: cage 11,000 lb., coal car 4500 lb., coal 8000 lb., rope 880 lb. The rope speed is 1000 ft. per minute.

The skip hoist is used for hoisting the coal in self-dumping skips oper-

* Engineer, Wellman-Seaver-Morgan Co. † Received Sept. 24, 1919.

ating in balance. The capacity of the hoist is 6400 tons of coal in 8 hr. on the basis of 8 tons per trip, or 8000 tons on basis of 10 tons per trip, the load being as follows:

| | Average Condition, Pounds | Maximum Condition, Pounds |
|---|---|---|
| Skip | 20,000 | 20,000 |
| Coal | 16,000 | 20,000 |
| Rope | 1,580 | 1,580 |
| Total | 37,580 | 41,580 |

The average rope speed is 980 ft. per min The flywheel of the motor-generator set has sufficient capacity for completing the hoisting cycle from any point in the shaft, after acceleration is completed, in case of failure of main-line current.

The calculated running light losses of the motor-generator set is 32 kw. The calculated power consumption under normal hoisting conditions is 0.485 kw.-hr. per ton.

The power for both hoists is purchased from the Central Illinois Public Service Co. at 33,000 volts, three-phase, 60-cycle, and is transmitted by wooden pole construction from a steam plant located at Christopher, Ill., a distance of about 20 miles.

The skips at the Standard mine have a capacity of 414 cu. ft. or 12 tons of coal and are designed to operate in a shaft compartment having T-rail guides. The body of skip, 6 ft. by 6 ft. 3 in. by 12 ft., is made up of 3/8-in. plates, 2 1/2 by 2 1/2 by 7/16 in. angles and four 1 by 6 in. stiffener bars extending around four sides of the skip. The sides of the guide frame are made of 3 by 2 1/2 by 3/8 in. angles riveted to a 7/8 by 16 in. plate running the entire length of guide frame. The skip is provided with spring drawbar and steel safety catches. Total weight of each skip is 19,000 pounds.

Frank F. Jorgensen, Gillespie, Ill.—We use the Litchfield hoist in all our mines as it is manufactured only 12 mi. away and we can get repairs very quickly. We have never been delayed over two days in getting the mine in operation after an accident; but it takes much longer to have our electric machinery repaired. However, I am very much in favor of the electric hoisting. We have found that in our Iowa mines we can hoist coal more cheaply with electricity than with steam.

In all operations below ground we use direct current altogether. We generate alternating current at 2300 volts and put the rotary converters right in the plant. We have been criticized for this plan but the old mines now are beyond the limit of carrying 250 or 275-volt current. We are at the point where we must do something. My plan is that when the new plant gets out, say 3/4 mi. or 1 mi. from the bottom, a substation will be placed at the face. For that reason the two converters we have now

are of small capacity but they are the size that we intend to take to the face when we carry the alternating current into the mine.

R. V. NORRIS, Wilkes-Barre, Pa.—It seems to me that 70 per cent. output is weak, compared to the practice of West Virginia and Virginia where they are getting a little over 90 per cent. output; it would seem as though a little of the detail of the inside work might well be added to this paper, which treats largely of the outside and hoisting end. In regard to the electrification, why not do as they have in many places, put the converters on the surface and take the current through bore holes near the face? It is safer than carrying your high current into the mines and does not cost appreciably more.

THE CHAIRMAN (CARL SCHOLZ, Chicago, Ill.).—We have to meet a situation that does not exist in West Virginia. Our coal ranges from 8 ft. 6 in. (2.58 m.) to 12 ft. 2 in. (3.7 m.). That coal is overlaid with a tender slate, therefore the top coal must be kept up. This is one of the worst features in the recovery of coal because, in advancing the rooms, we only drive them 7 ft. high. That brings down the percentage of recovery from what might be 85 per cent. to 70 per cent.

I do not believe that the plan of taking in the power through bore holes is feasible, but it is feasible and practical, I have found, to take high-tension voltage into the mines by armored cables. We are going to generate the power for our main road locomotives with four 100-kw. sets in the four quarters of the mine; that will eliminate great expense in cables. For the total output of our mines, we take inside three 300,000 circular-mils lead-covered cables; for that same amount of power, at 250 volts direct current we would have to put in 2,000,000 circular mils for each side. The 300,000 circular mils lead-covered cables are as large as my thumb; the 2,000,000 would be as large as my arm. This cable is protected by steel ribbons and is fool-proof.

The mining machines are operated by alternating-current motors, thus eliminating flames. The transformer stations are very small. In large transformer stations the secondaries soon become too long and the loss in voltage is too great. Our units are three, 25-kw. transformers; each operates two mining machines. That brings our secondaries down to a maximum length of 700 ft. which is the maximum distance we can carry. Small transformers and high-tension voltage will overcome the power trouble to a very appreciable extent; besides the power is brought to the face at full pressure and there is no dragging of the motors, so common where direct current is employed. This plan is a check on the activity of the mine superintendents in keeping the power at the face.

The Illinois law prohibits the use of over 275 volts on exposed wires but that law does not pertain to protected wire. The cables we use are fool-proof and are no more costly than cables for direct-current operation.

H. I. Smith.—What wire do you contemplate using from the transformers up to the mining machine?

Chairman Scholz.—The secondaries are ordinary weatherproof wires. They are carried on insulators suspended from the roof 3 or 4 in. (7 to 10 cm.) apart. As yet we have not experienced the least trouble by the use of ordinary double-braid weatherproof wire.

Mr. Parker.—What is the result of the deterioration from the use of the skips, the storage hopper, and the weighing hopper? Some Illinois coal breaks badly, I understand, in handling. It takes two or three handlings there instead of one at a dumping.

Chairman Scholz.—We have not had any experience in that line yet. I did operate a mine in Northern Illinois where we had a skip; in that case we found no difference in the breakage. The result was about the same in both mines.

We have found that if you transfer a large mass of coal lump by lump, the breakage is considerable. If you transfer a shovel full at the same time, your breakage is less. If you pick up 10 tons and dump it, the breakage is still less because of the cushioning effect. Therefore in skip dumps, where the coal flows into the skip, the breakage is reduced to a minimum. The shape of the skip provides the minimum fall in pouring into the skip and the minimum breakage of pouring out of the skip. In our case, though, breakage does not matter because all of the coal is to be crushed any way.

L. V. Rice, Chicago, Ill.—In the Standard Oil Co. mines, which we are developing, we expect to crush all of the coal so we are using a 12-ton skip. We dump first into a hopper that holds about 40 tons; from that into a measuring hopper, then into the skips, which we hoist at the rate of 900 ft. (274 m.) a minute. They are then dumped into a hopper containing 150 tons, and from that fed over a grizzly into a crusher. After the coal is crushed to 1¼ or 1½ in., it is distributed into large hoppers holding about 4000 or 4500 tons and from these it is loaded into the railroad cars.

In the mine we have two rotary dumps, which can be worked together or separately. Each rotary dump we expect can handle a car in about 20 sec. The mine cars are a little larger than most of the mine cars in the state; they will hold about 6 tons each. We are using 48-in. gage track, which is a little wider gage than that used elsewhere.

Mr. Parker.—Are your miners paid by the car?

L. V. Rice.—They are paid by the ton.

Mr. Parker.—Are the cars weighed in the mine?

L. V. RICE.—Yes. Each locomotive brings in thirty or thirty-five cars, which are then attached to a car haul which elevates them and throws them over the scale, which is self-registering. The cars then pass to the turn-over dump. All the cars will be numbered and the records kept from the numbers.

MR. MURPHY.—Are these gassy mines? In Columbia, our hoisting could not be done by electricity because of the gassy nature of the mines.

C. M. YOUNG.—As a general thing, the mines of Illinois are not sufficiently gassy to prohibit the use of trolley, so electric haulage gives very little trouble. There have been a few cases where a spark from the trolley caused trouble, but these were under unusual conditions. Except in a few cases, the mines are not operated with safety lamps.

L. V. RICE.—While we cannot get along very well in mines rated as non-gaseous without trolley haulage, in mines that are naturally dusty we must look out for the dust problem as well as the gas. There have been at least two great disasters in this country that, without question, were caused by the direct ignition of coal-dust clouds by the breakage of a trolley wire. In Illinois, generally, the dust is mixed with clay and other materials so that the hazard is not as great as in some mines, still that possibility must be borne in mind and coal dust guarded against.

Perhaps the first use of skips in coal mines, in this state at least, was in the Wilmington district where Richard Ramsey had installed a skip, which I saw in operation about 1892. At that time it was handling the coal in a very gentle manner. It was poured out as the skip came up and slid off the edge of the skip down the chute. I understood that there was practically no greater breakage than in the neighboring mines where they used the car dump.

In connection with the splendid tipples which are here, I recall it is only in 1899 that I had occasion to design the third steel tipple. The first tipples were built with light material, compared with the present practice, and the progress and advance in the art have been very marked.

Mr. Scholz said that one reason for the great losses of the coal was the top coal and the necessity of keeping the draw slate covered because of the weathering if the top coal is not left up. He might also have added another factor, which necessitates leaving very large amounts as pillars, namely, the very heavy limestone stratum above that. As many of the operators in this field know, there was great trouble in some of the early mines through squeezing, where too much coal had been removed in the advance work. The bottom came up and large amounts of coal were lost; in many cases whole sections of mines were lost.

Still another factor is that the surface is perhaps more valuable than the coal in the ground and if you disrupt this comparatively level sur-

face, you are deranging the tiling and the drains so that valuable farm land is converted into swamps. One of the greatest problems in coal mining that we have in the country is to get around this factor, because we will not be satisfied until we effect a better saving of coal in those districts confronted with these problems. If you could disregard the surface, the pillars could be pulled quite thoroughly and the top dropped. Some years ago, we endeavored to figure that a saving could be effected if a company would defer making the final recovery until it reached the boundary. I do not mean to say wholly with entry work alone, but in which larger pillars were left between the rooms and then the complete recovery of the coal was made in working back from the boundary. Disregarding the surface, it figured an economy, and today even more so. It could be planned broadly so that work could be handled this way. The drainage could be brought to some central place and from there turned into the streams by one agency. Even pumping, if it was handled in a big enough way, might be done, bringing back the coal regularly toward the shafts. That has been done in longwall mining in the northern part of the state, where the first subsidence caused a swamp. In the case of one mine we had to dig deeper channels and even do some pumping but as the face advanced the land corrected itself and has been recovered.

E. N. ZERN,* Pittsburgh, Pa.—Why have the number of entries been limited to three and four? It appears to me that in a mine of this size, bearing in mind the extensive development which will be required, a six- or eight-entry system would in the long run prove more economical. The Consolidation Coal Co. uses an eight-entry system in its Kentucky mines, the object being to keep down the water gage at such time when the mines are well developed and require a large amount of air. Of course, we all realize that the cost of labor, yardage, condition of roof, etc. must be considered as offsets to the savings made in ventilation, but we must not lose sight of the tremendous cost of circulating large volumes of air at high water gages.

CHAIRMAN SCHOLZ.—One of the limiting features, in Illinois, of entries is the cost, and that means the yardage. A four-entry system costs $4 a foot. An eight-entry system, therefore, costs twice as much. Another factor is the cost of maintaining entries, because in most mines in Franklin County we have to timber. We can better afford to timber a smaller number of entries and increase the horsepower cost of the ventilation system than to maintain entries with timber, which must be renewed every few years, or with steel, which costs something like $15 a set.

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* Editor, Keystone Cons. Pub. Co.

I want to say in connection with the adoption of the entry-driving machines, that we have two machines which under-cut, break down, and load the coal in one operation. There is not much economy connected with the operation of these machines at the present time; the economy will come as these entries advance. They will ultimately drive 3½ mi. to the boundary. We can, after 18 months' operation, notice a great deal of difference in the condition of the ribs that were cut by these machines as against those cut and shot down by explosives. The explosives will sooner or later cause cracks to reach the roof and the coal will commence to fall and require timbering. The entries driven by the machines are as solid today as the day they were put in.

Mr. Sperr.—What height do you drive your entries?

Chairman Scholz.—From 6½ to 7 ft. (1.97 to 2.1 m.).

J. W. Paul, Pittsburgh, Pa.—Referring to the part of the paper discussing the safety work of the Old Ben Corporation on p. 831, I would say that I have had the opportunity of seeing the Jones dust barriers in operation in the mine and on the surface. I am also advised by the management that they feel sure that by the operation of these devices they have prevented the spread of two explosions in two mines.

The purpose of the barrier is to throw, in the face of an advancing explosion wave, a large body of finely ground material so as to absorb the heat of the flame of the explosion. The dust barrier, as made, suddenly precipitates its contents on to the floor and may be operated in advance of the flame of the explosion by the shock wave which usually precedes an explosion. The barrier should be so constructed that it would precipitate gradually its contents during several seconds of time so that when the trap is thrown by the shock wave that precedes the explosion, there will be a time interval before all of the dust may be precipitated to the floor. At the same time, a certain amount of the dust should be left in the box so that when the violence of the explosion reaches the box, additional dust will be thrown into suspension. That can be easily accomplished by introducing shelves 4 to 6 in. wide in the interior of the box. These will retard the flow of the rock dust and at the same time retain a certain amount of the dust, which will be in a position to act when the flame and violence of the explosion would arrive. I believe it would be an improvement if the boards were made less than 10 to 12 in., probably 8 in. wide. That would give a larger clearance between the road and overhead for walking in case of accidental throwing of the trap.

Graham Bright,* Pittsburgh, Pa.—On p. 816, the author says that when a skip holds a two-mine-car load, the hoisting speed is cut in half. In some of our mines the hoisting distance is not very great and we find that with the given output you can reduce the speed considerably more than one-half. Assuming that actual raising time is about the same as the caging time, you can take one-third the rope speed and get the same output by hoisting twice the quantity of material per trip. Of course, with greater depths you do not get such a big difference, so the rope speed will be between one-half and one-third. Another large advantage gained is in the capacity of the equipment. If direct current is used, the capacity of the equipment includes the motor-generator set. In some cases, a hoist requires a flywheel set in order to keep down the peak load, due to the small capacity of the power system. This is true where we have high-speed cycles using single-car operation. By using skips, rope speed is cut down considerably and the peak load during actual operation is cut down to a much larger extent, so in some cases it permits the use of alternating-current motors to drive the hoist, where with a single-car operation it could not be used due to heavy peaks. The operation by skips often effects a large saving in the first cost of apparatus, and also in actual amount of power required.

* Westinghouse Elec. & Mfg. Co.

# Gas-producer Practice at Western Zinc Plants

BY G. S. BROOKS,* E. M., AND C. C. NITCHIE,† A. B., DEPUE, ILL.

(Chicago Meeting, September, 1919)

WITH the gradual depletion of the natural-gas pools of the Kansas district, together with the uncertainty of further cheap fuel developments, some of the western zinc companies turned to the coal fields, attracted by their permanency and the success of the Matthiessen & Hegeler Co. and the Illinois Zinc Co. Being then confronted with the problem of using a relatively high-priced fuel, although the ultimate supply was very great, the fuel economies of the kilns and furnaces became of far more importance, and the more modern equipments, as a result, embodied distinct gains in this phase of the business. In addition, there have been some pronounced advances in the actual making of gas from these coals.

Increased efficiency in gas making has contributed its share to lowering costs and those plants located in the higher priced fuel districts have probably been obliged to study these economies somewhat more closely. Naturally, many beneficial changes have resulted in both practice and apparatus in these sections. On certain western coals, for instance, the actual saving in fuel consumption has been 50 per cent. By this is meant that, where about 1.75 tons were burned to smelt 1 ton of oxidized ore, 0.7 to 0.9 ton is now required. While comparisons by W. R. Ingalls[1] of fuel economies in smelting in this country and Germany before the war may, in general, be correct, they do not appear to do justice to the more progressive American practice. It is doubtful whether German plants were substantially bettering these fuel ratios. Data furnished by Bender and Frams do not afford records averaging less than 0.9 ton coal per ton ore on materials from 40 per cent. to 50 per cent. zinc.

## FUELS

As producer fuel, Illinois, Indiana, and Kansas coals offer a wide range of material, although varying much in their amenability to gasification. The proper fuel for each plant can only be determined after com-

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[1] Some Points in the Economics of Zinc Metallurgy. *Eng. & Min. Jnl.* (Oct. 2, 1915) 551.

parative testing, the results of which include not only the weight and price of coal consumed per unit of production, but also the cost of gasification, including such items as labor, steam, etc. Naturally, the first cost of the coal at the plant is a great deal a matter of location, because for standard sizes the mine prices do not vary materially. In zinc works, where muffle roasting kilns and retort furnaces are the chief consumers of gas, clean gas is not considered important, so that the finer sizes of coal, dusty though they are, can be used.

*Size of Coal.*—As a general thing, coarse mine-run coal does not make the best producer material. The larger pieces work down through the distillation bed and tend to make unequal blow distributions on both the hand poked and mechanical types.

Egg and nut, being nearly free of fines, make the most easily handled material. They are frequently low in ash and the uniformity of their interstitial channels makes even blowing an easy matter. In the hand-poked producer, these sizes make easier work in "bed loosening" and "side slicing."

With the lower priced screenings, even with their more troublesome clinker and poor blowing properties, the best gains have been made in modern zinc practice. Washing makes some of the poorer grades of slack very satisfactory producer material.

*Ash.*—Particularly with slack coal, not only should the total ash be carefully examined, but also its fusibility; in fact, this latter quality is often the one factor that will exclude a coal as a producer fuel. There is not much accurate information on the melting point or composition of western coal ash. For instance, the ash from a Northern Illinois slack ran as follows: $SiO_2$, 40.8 per cent.; $Fe_2O_3$, 20.7 per cent.; $Al_2O_3$, 20.6 per cent.; CaO, 6.5 per cent.; MgO, 0.9 per cent.; undetermined, 10.5 per cent.

This ash was made from raw screenings containing $H_2O$, 14.5 per cent.; volatile matter, 26.8 per cent.; fixed carbon, 42.0 per cent.; ash, 16.7 per cent.; sulfur, 3.7 per cent.

As an example of poor material, there are found in Saline County, Ill., coals giving an ash with as high a lime content as 26 per cent.; these naturally make a very easily softened ash, so much so in fact that there has been trouble in keeping locomotive grates clean. This coal would be out of the question for gas making. A typical mine run, thick vein coal gave an ash of: $SiO_2$, 34.4 per cent.; $Fe_2O_3$, 30.8 per cent.; $Al_2O_3$, 29.4 per cent.; CaO, 4.3 per cent.; MgO, 0.6 per cent.; undetermined, 0.5 per cent.

This original coal ran 24.4 per cent. ash and 7.1 per cent. sulfur in the coal yet it did not form as hard a clinker as the first slack ash given. Of course the difficulties of clinkering are chiefly controlled by the following: (1) Rate of combustion, and therefore actual combustion zone temperatures; (2) size of coal, *i.e.*, contact factor; (3) steam for disin-

TABLE 1.—*Analyses of Various Grades of Coal*

| | $H_2O$ as Received | Proximate Analysis of Dry Coal: Volatile Matter, Per Cent. | Proximate Analysis of Dry Coal: Fixed Carbon, Per Cent. | Proximate Analysis of Dry Coal: Ash, Per Cent. | Sulfur (Dry), Per Cent. | B t u per Pound (Dry) | Melting Point of Ash, Degrees | Softening Interval, Degrees | $SiO_2$, Per Cent. | $Fe_2O_3$, Per Cent | $Al_2O_3$, Per Cent. | $TiO_2$, Per Cent. | CaO, Per Cent. | MgO, Per Cent | $SO_3$, Per Cent. |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Harrisburg, Ill., screenings.. | 9 1 | 34 0 | 53 6 | 12 4 | 3 1 | 12,621 | 1,192 | 119 | 38 5 | 23 1 | 21 6 | 0 9 | 8.7 | 1 3 | 6 8 |
| Marion, Ill., mine runs. .. | 10 0 | 34 1 | 52 4 | 13 5 | 2 9 | 12,512 | 1,218 | 132 | 48 5 | 16 5 | 28.3 | 0 4 | 4.9 | 1 7 | 5 6 |
| Herrin, Ill., mine run....... | 10.3 | 34.8 | 52.0 | 13 2 | 3.5 | 12,592 | 1,078 | 63 | 42.9 | 25.7 | 21.7 | 0.6 | 3.6 | 0 9 | 4 8 |
| Herrin, Ill., nut ......... | 9 8 | 29.9 | 59 1 | 11 0 | 1.5 | 12,560 | | | | | | | | | |
| Benton, Ill, mine run...... | 7.8 | 38.9 | 47 3 | 13.8 | 4.3 | 12,350 | 1294 | 8 | 38 6 | 26 3 | 19 4 | 0 5 | 6.0 | 0.9 | 7.5 |
| Indiana 4th Vein, Midland, Ind.................. | 19 1 | 34 2 | 55 9 | 9 9 | 1.30 | 12,915 | | | | | | | | | |
| Indiana 7th Vein, Shelburn | 17 6 | 31 5 | 49 1 | 19 4 | 1.5 | 11,633 | | | | | | | | | |
| LaSalle, mine run ...... | 13.2 | 41 9 | 45 3 | 12 8 | 4.8 | 12,475 | | | | | | | | | |
| Peoria, Ill..... ........ | 18 6 | 34 5 | 46.3 | 19.2 | 4 3 | 11,509 | | | | | | | | | |
| Tower Hill, Ill., screenings.. | 10 6 | 35 9 | 50 1 | 14 0 | 2 5 | 11,390 | | | | | | | | | |
| Terre Haute, Ind., slack.... | 19.5 | 33.3 | 53.7 | 13.0 | 2 0 | 11,710 | | | | | | | | | |
| Toluca, Ill., egg .......... | 6.0 | 31.9 | 56 4 | 11 7 | 6 2 | 12,615 | | | | | | | | | |
| Cardiff, Ill., mine run .... | 8.0 | 33.5 | 53.7 | 12 8 | 4.0 | 12,027 | | | | | | | | | |
| Standard, Ill., mine run.... | 10.4 | 36 5 | 47.3 | 16 2 | 6 8 | 11,369 | | | | | | | | | |

tegration; (4) agitation of material in combustion zone effected by hand poking or mechanical means; (5) removal of ash from combustion zone to ash zone promptly and completely.

*Hydrocarbons.*—As a producer fuel, Illinois coals have the well-known advantage of all bituminous coals, an abundance of hydrocarbons. For high-temperature work, this is important in the subsequent efficient transfer of heat from the flames to furnace walls or retorts. It gives to the burning gas what Siemens called its "light and heat radiating capacity," and depends on the presence of free carbon particles, the result of hydrocarbon dissociation. These very small particles are heated to the flame temperature and, remaining incandescent, radiate heat and light in all directions.

Table 1 gives representative analyses of various grades of coal that have been used at Depue during the past 10 or 12 years.

## Types of Producers and Their Operations

*Hand-poked, Dry Bottoms.*—Of this type, the earliest was the Matthiessen & Hegeler generator; its modifications have now come into somewhat extended use at La Salle, Springfield, Danville, Argentine, Hillsboro, and Burgettstown. Essentially, the producer comprises a stationary brick shaft with feed openings in the top for coal and a gradually narrowing throat, or bosh, at the bottom, below which is the ashpit. At the point of minimum opening at the throat there is usually a water-cooled frame; this prevents the destruction of the lining of this opening and breaks the clinkers. The blow box is incorporated in the ash bottom, which is wide enough to allow the ash to seal the column, running out to the angle of repose but not quite reaching the box side. Along these sides, the air circulates and distributes itself through the bed. A fan delivers the air by a main trunk line, using damper cut-offs for the adjustment of the individual producers. There are from one to four units to a spelter furnace generator equipment.

This type of producer is ordinarily used on spelter furnaces with waste heat boilers as heat recuperators, or, when on kilns, is usually operated in connection with air heaters. There is necessarily a large amount of clinkering with this type, as no steam is ordinarily introduced as a "mover" for the primary air. Fig. 1 shows the general arrangement of an installation formerly used at Depue. A positive blower used as an exhauster between the furnace and producers has been in successful operation at La Salle for a good many years. This of course does away with either a fan or a steam blower. The breaking down of these producers often becomes a very serious matter and requires considerable experience and skill unless the coal be uniform and its clinkering inconsiderable.

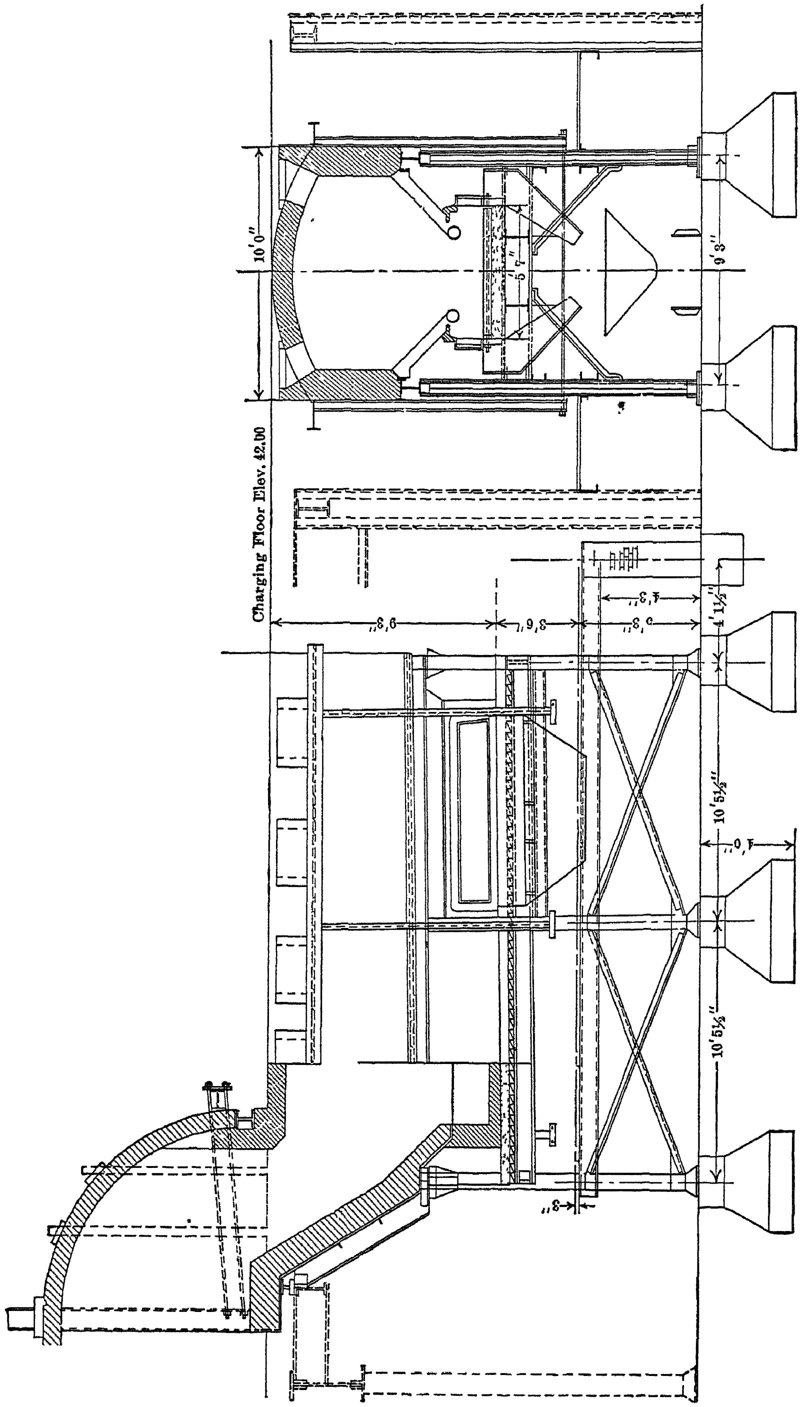



FIG. 1.—GAS PRODUCER FOR SPELTER FURNACE, NASSAU WORKS, MINERAL POINT ZINC CO.

TABLE 2.—*Producer Equipment at Western Zinc Plants*

| Location | Company | Type | Use | Number |
|---|---|---|---|---|
| Depue | Mineral Pt. Zinc | Duff | Spelter | 6 |
| Depue | Mineral Pt. Zinc | Duff[a] | Kilns | 14 |
| Depue | Mineral Pt. Zinc | Chapman | Spelter | 2 |
| Depue | Mineral Pt. Zinc | Hughes | Spelter | 16 |
| Peru | Illinois Zinc Co. | Swindell | Spelter | 7 |
| Peru | Illinois Zinc Co. | Chapman | Spelter | 4 |
| Peru | Illinois Zinc Co. | Swindell | Kilns | 3 |
| La Salle | Matthiessen & H. | M. & H. | Spelter | 7 |
| La Salle | Matthiessen & H. | M. & H. | Kilns | 5 |
| Danville | Hegeler Zinc Co. | M. & H. | Kilns | 3 |
| Danville | Hegeler Zinc Co. | M. & H. | Spelter | 5 |
| Hillsboro | Lanyon | M. & H. | Spelter | 3 |
| Hillsboro | Lanyon | M. & H. | Kiln | 1 |
| Taylor Springs | A. Z. L. & S. Co. | M. & H. | Kiln & spelter | 2–4 |
| Taylor Springs | A. Z. L & S. Co. | Wood | Spelter | 4 |
| East St. Louis | A. Z. L. & S Co. | Hughes | Spelter | 12 |
| East St. Louis | A. Z. L. & S. Co. | Chapman | Kiln | 2 |
| Burgettstown | A. Metal Co. | M. & H. | Kiln & spelter | 2–4 |
| Burgettstown | A Metal Co. | Wood | Spelter | 5 |
| Tiltonville | Prime Western | Duff | Kilns | 4 |
| Mineral Point | Mineral Pt. Zinc | Swindell | Kilns | 2 |
| Pueblo | U. S. Zinc Co | Duff | Kiln & spelter | 2 |
| Pueblo | U. S. Zinc Co. | Morgan | Spelter | 1 |
| Argentine | National Zinc Co. | M. & H. | Kiln | 2 |
| Springfield | National Zinc Co. | M. & H. | Spelter | 7 |
| Grasselli | Grasselli C. Co. | M. & H. | Kiln | 2 |
| Cleveland | Grasselli C. Co. | M. & H. | Kiln | 2 |
| Canton | Grasselli C. Co. | M. & H. | Kiln | 1 |
| Cincinnati | Grasselli C. Co. | M. & H. | Kiln | 1 |
| New Castle | Grasselli C. Co. | M. & H. | Kiln | 1 |
| Terre Haute | Grasselli C. Co. | Wood | Spelter | 5 |

[a] One modified Duff with Chapman Agitator.

The Illinois Zinc Co. Swindell producers are built in the same manner as the Hegeler down to the combustion zone. With these producers, grate bars of square iron section hold the fuel bed from the ashpit. When the morning work is done, a second set of grate bars is inserted through the walls through the top of the ash bed just below the combustion belt. When these are in place the lower set is pulled, letting the ash from 24 hr. operation into the pit. When the cleaning is done, the lower set is replaced and the upper temporary bars pulled, allowing the contents of the producer to settle. This operation permits very little disturbance of the different fuel beds and is an effective system of cleaning fires. It has also great simplicity of both apparatus and operation. These producers are blown with jet blowers of the Schuette & Koerting type. In both,

hand poking through the top is used to keep the beds open and burning uniformly.

Fig. 2 shows the general grouping of these producer units. Three blocks of twelve units are used on each spelter furnace, and two blocks of eight units for each M. &. H kiln. At times when the ash does not give large clinkers, the morning work is done by simply moving the permanent bars aside and working the material down into the pits with a bar. The sliding dump boxes are inclined to be smoky when a producer is under pressure and fill the house with gas at the dumping times.

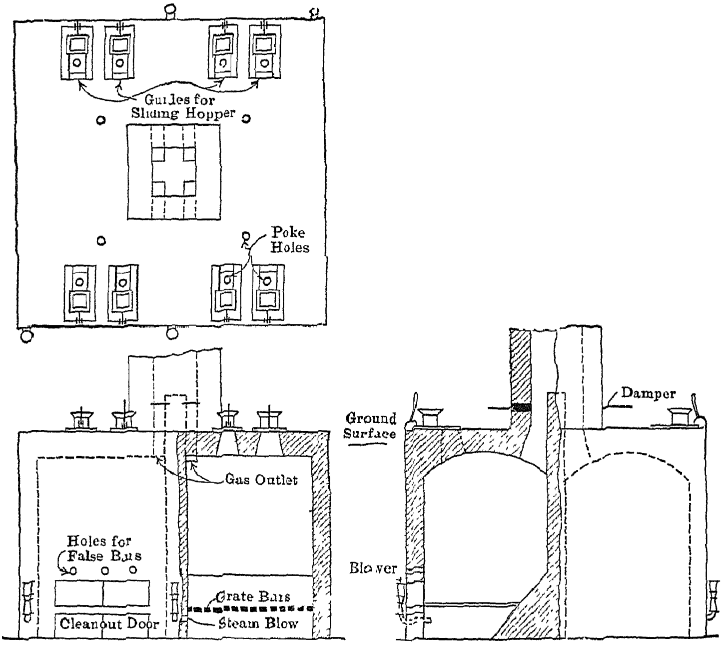



FIG. 2.—SWINDELL PRODUCER IN OPERATION AT PERU, ILL., WORKS OF ILLINOIS ZINC CO. ONE BATTERY, OR UNIT, CONSISTS OF FOUR FIREPLACES FEEDING INTO A COMMON FLUE, OR STACK. TWO BATTERIES USED TO ONE SEVEN-HEARTH KILN, AND THREE BATTERIES USED TO ONE 830-RETORT SPELTER FURNACE.

*Hand-poked, Wet Bottoms.*—In this class are the well-known Duff producers. At Depue there are six of the Duffs on the spelter furnace gas equipment and fourteen in the roasting work. An 800-retort furnace of the Siemens regenerative type requires from three to four producers, The producers are of the usual design and are blown with Schuette & Koerting steam jet blowers, see Fig. 3. It has been found beneficial at Depue to increase the size of openings in the grates over that usually prescribed by the manufacturer. Especially in working up practice on

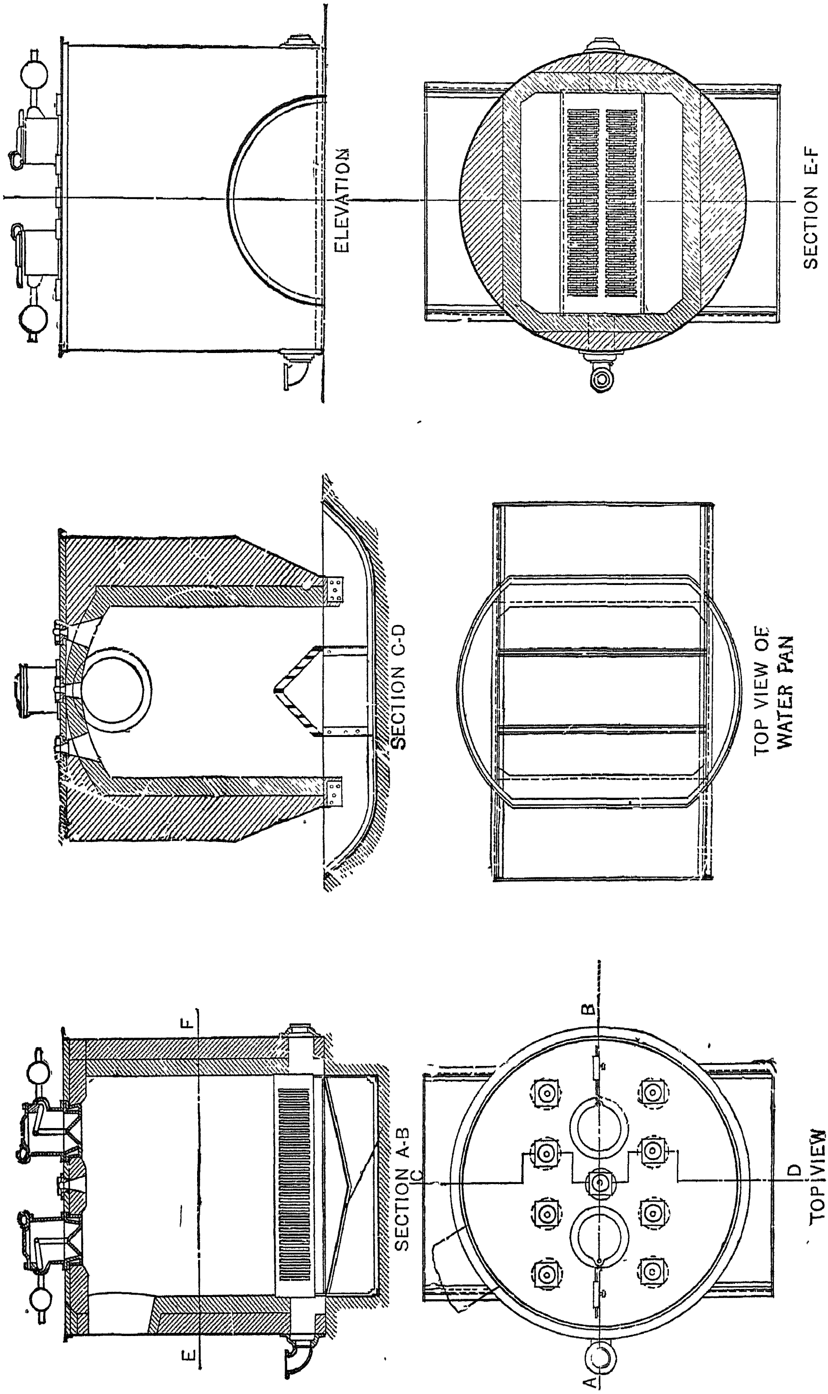

ELEVATION
SECTION E-F
SECTION C-D
TOP VIEW OF WATER PAN
SECTION A-B
TOP VIEW
A
B
C
D
E
F

FIG. 3.

the finer coals, this aids in the blow distribution. Against this, of course, is the more frequent cleaning required below the grates.

Under normal working, the rate of gasification varies from 8 to 10 lb. of coal per square foot of area per hour. It is well to note the fact that, on a spelter furnace, this average rate may be considerably lower than the actual rate required just before the charge is "touched off." These figures are not the limit of the capacity, but somewhere near the proper capacity for a minimum of labor coupled with a good quality of product.

The cycle of operation begins with the removal of ashes from the water-sealed scows at 5.30 A. M. This operation is done by the regular producer man, one to each producer. Care is required in removing the right amount of ash, so that the combustion zone does not drop too low in the next step. As a rough indication of how much ash to remove, a good producer man will run his hand up the steel shell and locate fairly closely the top of the ash bed and the bottom of the combustion zone. Should the clinker be very hard, water is introduced by pipes into the side of the shell; this practice, although somewhat questionable, does make a brittle clinker. Its action on the brick lining, as can be imagined, is not particularly good. The tendency is to take out a little less than the make of ash for, say, three or four days, then as the accumulation begins to be noticeable in the operation, to pull down very low on the fifth day.

Ordinarily the "making of the fire" is done at the same time the kiln or furnace is off gas, which occurs once at least in the shift. The steam being cut off, the producer man opens up all top poke holes preparatory to beginning work. It is advisable to have the fuel bed of green coal pretty well burned off at this time, so that the top is clear of gas. Beginning at one corner, a "first" slice hole is made through the bed, which has been left arched over the cavity made by taking out ash. This is directly above one of the openings between the wall and the cast-iron grate. Working out from this by undercutting the edge of the hole, the ash is gradually poked down upon the grate, allowing the fire bed to follow from above but not to mix. Both sides of the producer are worked in this fashion. The loss of carbon in the ash during this work depends on the care of the operator. Naturally some coke is bound to be tumbled into the ash in this undercutting, but it need not be serious with close attention to the work.

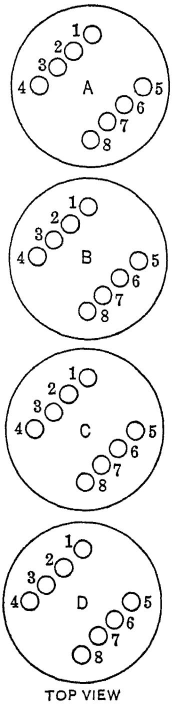
Another possibility that must be guarded against is the mixing of "float" clinker with the coke bed. When this occurs, the floater, no longer having the advantage of a high carbon content, is gradually softened in the combustion zone and builds up with the fine material around it, resulting frequently in a very troublesome slab of clinker.

When the fuel bed has been leveled off, the steam blow is turned on once more and a thin bed of coal is fed upon the made fire. The producer

is now in shape for the next shift. When possible, dumping through the bell hopper should be done at regular intervals. The gas will be of greater uniformity, a condition important to the best metallurgical results.

SCHEDULE

| Producer | Hole | Time of start | Poking period in minutes |
|---|---|---|---|
| C | 2 | 10.56 A.M. | 2 |
| C | 3 | 10:58 | 2 |
| C | 6 | 11:00 | 2 |
| C | 7 | 11:02 | 2 |
| A | 2 | 11:10 | 1 |
| A | 3 | 11.11 | 1 |
| A | 1 | 11:12 | 2 |
| B | 3 | 11:23½ | 4 |
| B | 2 | 11:28 | 4 |
| Observer out 2 hours | | | |
| D | 3 | 1 23 P.M. | 3 |
| D | 7 | 1:23 | 2½ |
| D | 6 | 1.26 | 1 |
| D | 2 | 1:26 | 1 |
| C | 6 | 1:40 | 1 |
| B | 7 | 1:40 | 3 |
| C | 7 | 1:41 | 2 |
| B | 6 | 1:43 | 2 |
| C | 2 | 1:44 | 1½ |
| C | 3 | 1:45½ | 2 |
| A | 7 | 2:22½ | 2½ |
| A | 6 | 2:26 | 3 |
| A | 2 | 2:36 | 1 |
| C | 2 | 2:54 | 2 |
| C | 3 | 2:56 | 1½ |
| C | 6 | 2:58 | 2 |
| C | 7 | 3:00 | 1½ |
| B | 3 | 3:08 | 3 |
| B | 2 | 3:12 | 1½ |




TOP VIEW
PRODUCER BATTERY
SHOWING POKE HOLES

FIG. 4.—POKING SCHEDULE, DUFF PRODUCERS, NO. 1 FURNACE.

The agitation of the fuel bed by hand poking does at least three things: It closes the channels, spreads the piles of freshly dumped coal under the bells out to the edges, and breaks up the sticking of material in the bed. The experienced producer man systematically pokes one hole on each side of the top at intervals. In the case of a six-hole top and 20-min. intervals, he will get over his bed once every hour. The most effective barring is done by beginning with a vertical motion of the poker,

until the area commanded by the holes has been thoroughly broken well down to the ash. This straight up-and-down breaking is simply hand-churn drilling. The poker is then run down into the bed and given a sideways stirring movement; beginning near the surface, this should be continued well into the bed. At the surface, this movement spreads the coal, but farther down closes up the holes and makes the distribution of the blow more uniform over the cross-section of the producer. Here, too, is an opportunity for a mixing of ash with burning coal, with its troublesome results. The practical management of the producer is not a little in knowing how seldom to do this poking, for it can be overdone. Fig. 4 shows the actual spacing of this work by an experienced producer man, working from the appearance of the fires.

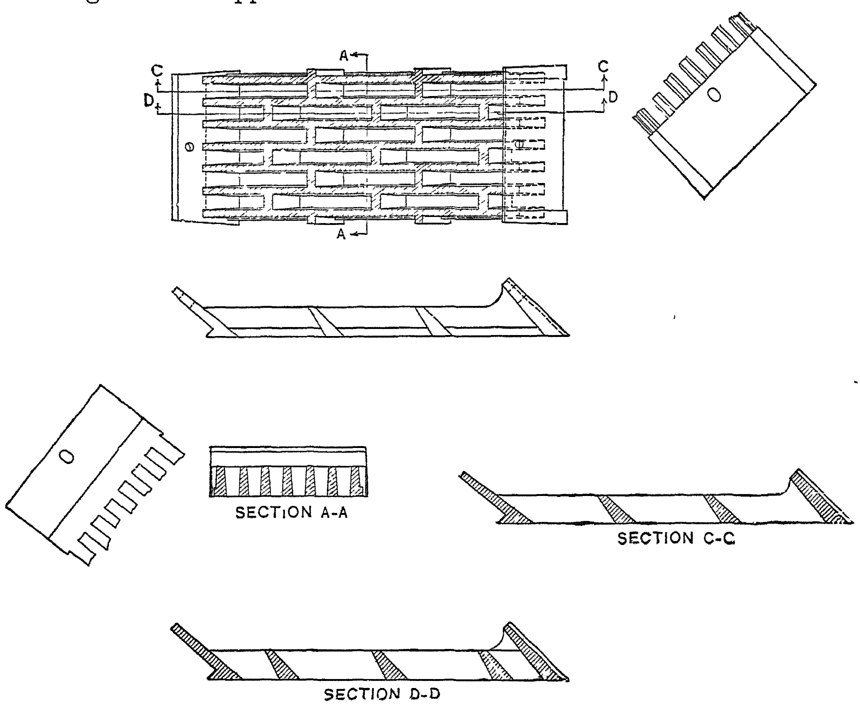



FIG. 5.—SPELTER KILNS, DUFF GAS PRODUCER, DETAILS OF GRATES

The ratio of air to steam can be roughly adjusted at the blower, but for anything like a uniform coal, it is best not to allow producer men very much latitude with these adjustments. A short time will teach a green man that he can lessen his hard muscular work by a large quantity of steam. Steam, of course, is objectionable for metallurgical reasons and because of excessive steam consumption. Especially on very fine slack, the box blow under the grates must be cleaned often, and this is best made a routine affair. In cleaning or slicing the tops of the grate

at monthly intervals, the following procedure is carried out: Shovel out ashes, cut chute through to ridge, the top hanging from clinkering, widen chute to ridge, slice grate face, do the same on other side of gable, thus finishing the producer. The shape of the slots in the grate is so arranged as not to blind easily. The one finally adopted at Depue is shown in Fig. 5, as is the latest cast-iron support.

## Modified Duff With Chapman Agitator

The Chapman floating agitator, as shown by Fig. 6, has a horizontal rabble bar with projecting fingers that plow the fuel bed. This rabble extends nearly the diameter of a circular producer. It is attached to the end of a central shaft, which is revolved by a small motor, acting through an eccentric and ratchet to a worm gear. The speed of the central shaft is about 5 r.p h. The central shaft has a spiral driving lug that permits a 24-in. (60.9 cm.) lift from the lowest position of the rabble. Twelve loose weights, of about 70 lb. (31 kg.) each, may be placed on top of the central shaft, if required, to hold down the rabble arm. All parts extending inside the producer are water-cooled. The cold water enters the top of the vertical shaft through a swing joint and is carried by an inside pipe to the fingers, where it is discharged and forced back up between the outer and inner walls. After cooling the rabble arm, the water is discharged from two overflow pipes on the central shaft into a sort of catch basin over the feed hopper. A spillway then drains the water into the hollow chamber surrounding the feed bell, where it also cools the tripod bearing. The final overflow is from the above chamber just below the floor level. The charging door, feed hopper, and feed bell are on the same principle as the old Duff and are operated by hand. The makers guarantee this machine to eliminate hand poking, and to seal off blowholes, giving a greater efficiency and capacity.

*Installation at Roast Kilns at Depue.*—The producer originally was a 7 by 9 ft. (2.1 by 2.7 m.) rectangle inside a round steel shell 12 ft. 3 in. (3.7 m.) in diameter, as shown on Fig. 3. This illustration shows two blowers, but all producers are operated with one blower. The old bricks were torn out and the shell lined with a firebrick wall about 12 in. (30 cm.) thick, as shown in Fig. 7. The grates are of the "*A*" type with 1 in. (2.5 cm.) slots, see Fig. 5, and extend lengthwise of the original rectangle. Blank grates were added to increase the length to 10 ft. 3 in. (3.1 m.). The grates were blanked off 16 in. (40.6 cm.) on each end, leaving seven open grate castings on each side. The following is the present practice used in operating the mechanical-top producer.

*Labor.*—Originally, there were two producer men on the first shift. With the softening of the clinker and the adoption of better methods,

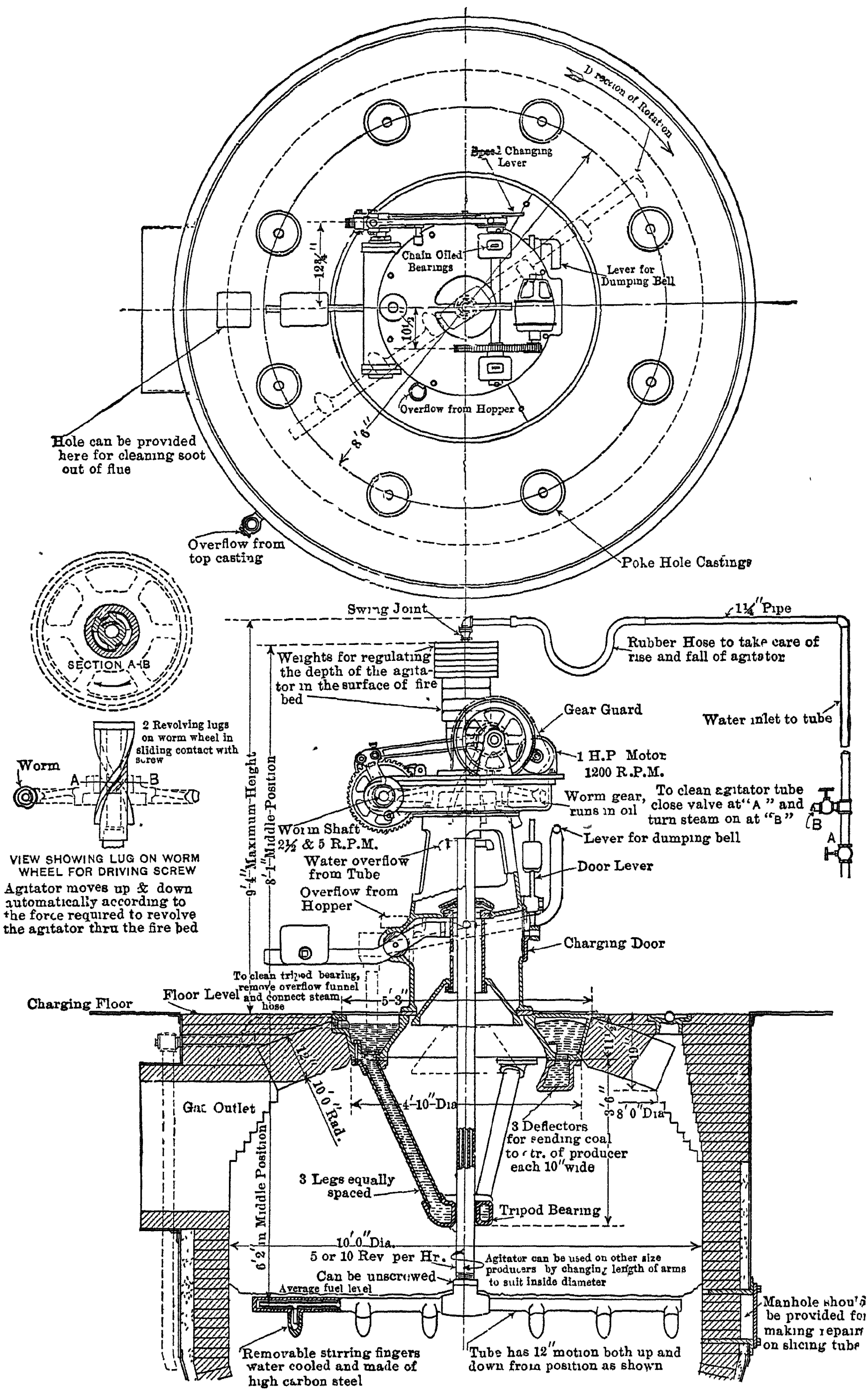



FIG. 6.—FLOATING AGITATOR AND FEED FOR STATIONARY GAS PRODUCER.

the labor of "breaking down" decreased, until it was possible to carry but one man on the first shift. There are three 8-hr. shifts of one man

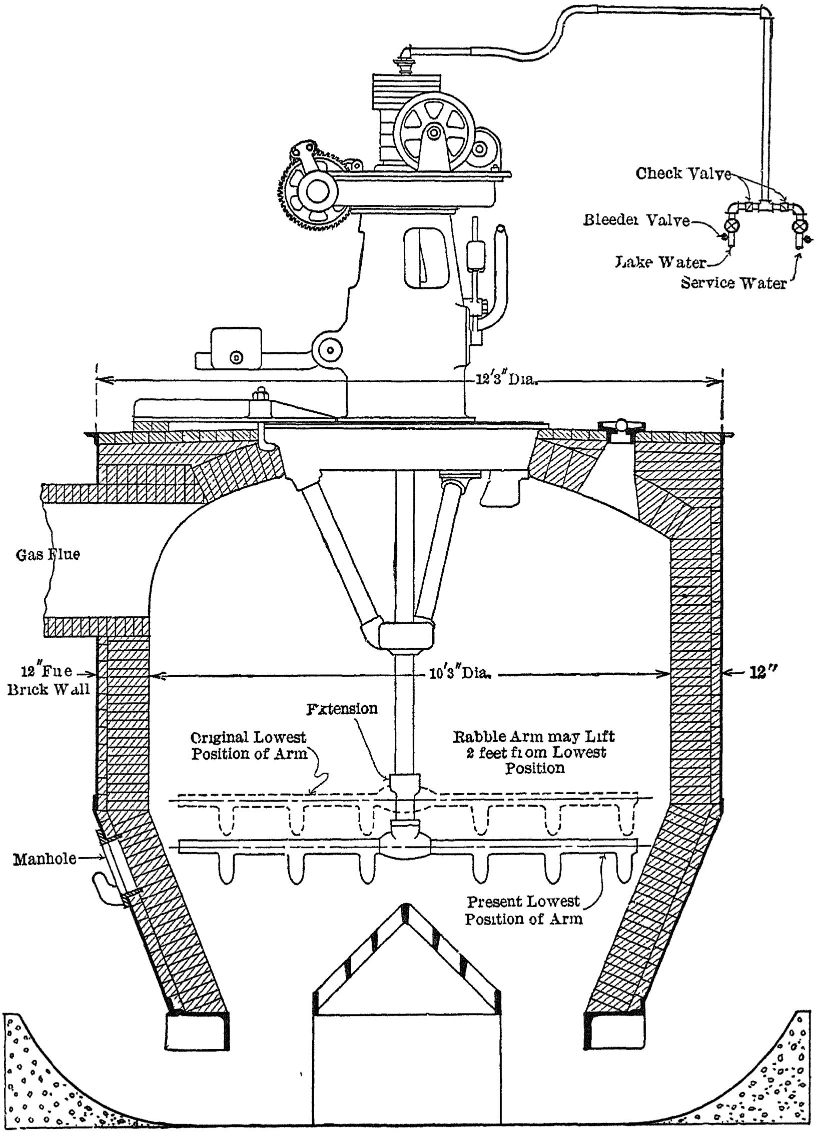



FIG. 7.—CHAPMAN FLOATING AGITATOR INSTALLED ON REMODELED DUFF GAS PRODUCER AT ROAST KILNS

each, 7 A. M. to 3 P. M., 3 P. M. to 11 P. M., and 11 P. M. to 7 A. M. There is of course more work for the man on the first shift, but as the shifts rotate every two weeks, it balances in the long run.

*Removal of Ashes.*—The height of the rabble arm determines the amount of ashes to be removed. The removal of one wheelbarrow (about 380 lb. (172 kg.) dry weight) of ashes will allow the rabble arm (if lifted) to settle a little more than 1 in. (2.5 cm.); that is, provided the ashes are taken out equally from both sides of the producer. The aim is to have the rabble arm lifted about 2 in. after it has leveled off the bed from "breaking down" in the morning. For example, if the arm were lifted 10 in. (from lowest position) at 7 A. M., three wheelbarrows of ashes would be removed from each side. After breaking down the fire and allowing the rabble arm about 1 hr. to level off the bed, the arm would be lifted about 2 in. Lately, it was found that after removing ashes from both sides and not breaking down, the rabble arm settled itself. Accordingly, in order to lessen the work of breaking down in the morning, ashes are removed either during the afternoon or night or both, as the height of the arm warrants. In nearly every case the rabble arm has settled in from 1 to 2 hr. Ashes are always removed before breaking down the fire, and it is important that about the same amount be removed from each side.

Taking out ashes usually occupies about ½ hr. in the morning. The ashes are loaded from the producer into two-wheeled carts holding (when fully loaded) about 550 lb. (249 kg.) wet weight. The wheelbarrows are dumped into the skip of an electric hoist, which the men operate as they use. The skip holds two wheelbarrows of ashes, and carries them from the producer cellar to a chute over a standard gage railway car at ground level.

*Breaking Down.*—Breaking down is done in the morning after taking out ashes. The steam is shut off and the dampers of the Swindell valve at the kiln are set to draw from the producer directly up the stack. Bars 1 in. (2.5 cm.) in diameter by 13 ft. 6 in. (4.1 m.) with a large hand ring and a blunt end have been found the most satisfactory for the main work.

Fig. 8 illustrates the following description of operations, which, while not always the same, are generally as given. Through hole *1*, the bar is put diagonally across the grates and a hole about 18 in. (4.5 cm.) in diameter is punched through the bed as close to the grates as possible. This is extended as a slot parallel to the grates from the center to the corner. The operation is then repeated through hole *2*, care being taken to punch all clinker down and mix in as little coke as possible. Through holes *3* and *4* a corresponding slot is cut on the opposite side of the grates. Shorter and lighter bars, ¾ in. by 12 ft. 6 in., are then used through holes *5* and *6* to break down the corners where the grates meet the wall.

Usually at this stage the arm is started rotating, and, if the bed does not settle evenly, bars are put through holes *7* and *8*. Any large pieces

of clinker that may remain on top are picked up by the stirring fingers; these are broken up with light bars as the arm revolves. Breaking down usually requires about 40 min. for one producer man unaided.

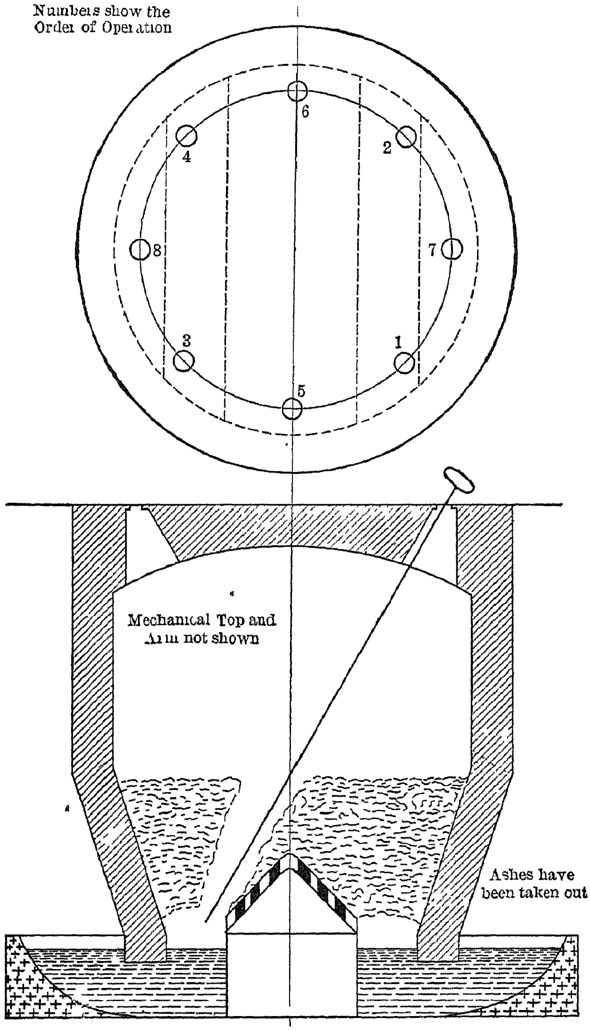



FIG. 8.—METHOD OF BREAKING DOWN MECHANICAL TOP PRODUCER.

*Firing.*—After the bed is broken down the dampers in the kiln are set properly, the cups placed in the poke holes, and the steam turned on. Two full hoppers of coal (about 400 lb.) are dumped when starting. Then in about 10 min. two more full hoppers are dumped. From then on the producer is fired every 15 min., about 200 lb. to a charge. The

producer man watches the fire and, if required, dumps an extra charge in between the regular periods.

*Water Cooling.*—The water-cooling system has been very satisfactory. One of the stirring fingers was unscrewed after the producer had been in operation 2 mo. and it showed practically no deposit. The volume of water used is about 9 gal. (34 l.) per min. The temperature of the water from the last overflow is usually 100° to 120° F. (38° to 49° C.) with the water entering the producer at 40° to 50° F. (4 to 10° C.).

*Maintenance.*—The producer has not been in operation long enough to determine its wearing qualities. The only parts showing wear are the cast-iron dogs on the ratchet drive.

*Cleaning Flues.*—The gas from the producer enters a 27-in. (68.5 cm.), inside diameter, brick-lined flue about 12 ft. (3.6 m.) long. It drops from this to an underground brick flue 2 ft. wide, 3½ ft. high, and 196 ft. long, to a 27-in., inside diameter, riser leading to the damper box. The first flue has an end door through which a scraper and air blast are used to push the soot and dust back into the producer. The underground flue has clean-out openings, about 16 ft. (4.8 m.) apart, through which the deposit is removed with long iron-handled shovels. The air blast, which is a long ¾-in. (19 mm.) iron pipe reduced to ⅜ in. (9.5 mm.) connected to a ¾-in. hose, is used for a final clean-up in the underground flue.

All producer flues, corresponding to the first flue mentioned, are cleaned every 2 wk. The accumulation from the mechanical producer in this flue is more than twice that from one Duff. This underground flue requires cleaning every 2 to 4 wk., while the others run from 3 to 6 mo. This condition is largely due to the low temperature at which the producer is operated; also to the length and cross-section of the underground flue. This one is about 33 ft. (10 m.) longer than the others and has the same cross-section.

## Comparative Test

A test was made for 5 days, comparing the mechanical top producer with two Duff producers. The two Duffs supply gas to No. 6 kiln through an underground flue 163 ft. (49 m.) long, about 33 ft. shorter than that for the mechanical on No. 5 kiln. The coal used at both places comes from the same car. An effort was made to keep both kilns at the same temperature, and that normal. The following methods were used on the test:

*Coal Fired.*—Two platform scales were used, with a large ash can holding 200 lb. (90 kg.) of coal on each. All coal used on the mechanical and on each of the two Duffs was weighed and a grab sample taken from each can to be split down every day. These samples were run in the main laboratory.

*Ashes Removed.*—The two-wheeled carts used for ashes hold, when fully loaded, about 550 lb. of wet ashes. The quantity of ashes removed each morning was estimated from the wheelbarrows. The ashes were sampled by putting every tenth shovel full in a separate pile and splitting it down. These samples were run for moisture and fixed carbon.

*Gas.*—One sample was taken from the mechanical and a "half and half" from the two Duffs every hour, and run in the kiln laboratory for CO and $CO_2$ by the Hempel method.

*Steam Pressure.*—There is one gage for the two Duffs and one on the mechanical, these were read every 3 hours.

*Height of Rabble Arm.*—The number of inches the arm was raised above the lowest position was measured every 3 hours.

*Summary of Results.*—Below is a summary of the average results from the 5-day test:

| | Mechanical | Two Duffs |
|---|---|---|
| Coal (as fired) burned daily, pounds | 19,120 | 22,880 |
| Dry ash removed daily, pounds | 2,560 | 4,500 |
| Fixed carbon in ash, per cent | 15.3 | 32 3 |
| Dry ash to dry coal, per cent | 14.6 | 21.5 |
| CO in gas, per cent | 19.2 | 18.7 |
| $CO_2$ in gas, per cent | 6 1 | 5 4 |

Table 3 shows the results obtained each day. The coal used during the test came from two cars, both run of mine, the first from Marion, Ill., and the second from Benton, Ill.

Coal Analyses

| | Ash, Per Cent. | Sulfur, Per Cent. | Volatile Matter, Per Cent. | Fixed Carbon, Per Cent | $H_2O$, Per Cent. | B t u |
|---|---|---|---|---|---|---|
| 1st day—Marion run of mine | 10 3 | 2.6 | 35 0 | 54 7 | 8 7 | 13,021 |
| 2d day—Marion run of mine | 11 7 | 3.0 | 34 4 | 53.9 | 8 5 | 12,617 |
| 3d day—Benton run of mine | 8 9 | 2 4 | 34.4 | 56 7 | 6 6 | 12,896 |
| 4th day—Benton run of mine | 10 5 | 2 9 | 34.8 | 54 7 | 8 8 | 13,547 |
| 5th day—Benton run of mine | 11.8 | 3 3 | 38.8 | 49 4 | 10 4 | 12,586 |

The bed area of the mechanical producer is 82 sq. ft. (7.6 sq. m.). The coal burned is 234 lb. (1142 kg. per sq. m.) per sq. ft. Deducting 1 hr. the producer is shut down, the coal burned is 10.2 lb. per sq. ft. per hr. The bed area of each Duff is 63 sq. ft. or 126 sq. ft. for two Duffs. The coal burned is 181.5 lb. per sq. ft. Deducting 2 hr. the producers are shut down, the coal burned is 8.25 lb. per sq. ft. per hour.

TABLE 3.—*Results of 5-day Test, Comparing Mechanical Top to Two Duff Producers*

| | 1st Day | 2d Day | 3d Day | 4th Day | 5th Day | Average |
|---|---|---|---|---|---|---|
| Coal burned in No. 5 Duff, pounds | 11,800 | 12,200 | 11,800 | 10,800 | 10,200 | 11,360 |
| Coal burned in No. 6 Duff, pounds | 11,600 | 12,400 | 12,600 | 11,000 | 10,000 | 11,520 |
| Coal burned in two Duffs, pounds | 23,400 | 24,600 | 24,400 | 21,800 | 20,200 | 22,880 |
| Coal burned in mechanical, pounds | 19,600 | 21,400 | 19,800 | 18,800 | 16,000[a] | 19,120 |
| Wet ash from mechanical, pounds | 4,240 | 4,620 | 4,240 | 3,140 | 2,750 | 3,800 |
| $H_2O$ in ash, per cent | 30.0 | 33.0 | 33.0 | 38.0 | 30.0 | 32.8 |
| Dry ash from mechanical, pounds | 2,970 | 3,100 | 2,840 | 1,950 | 1,920 | 2,560 |
| Wet ash from No. 5 Duff, pounds | 4,680 | 3,170 | 4,850 | 5,700[b] | | 3,680 |
| Wet ash from No. 6 Duff, pounds | 2,670 | 4,090 | 3,600 | 6,100[b] | | 3,290 |
| Wet ash from two Duffs, pounds | 7,350 | 7,260 | 8,450 | 11,800[b] | | 6,970 |
| $H_2O$ in ash, per cent | 35.0 | 36.0 | 38.0 | 34.0[b] | | 35.4 |
| Dry ash from two Duffs, pounds | 4,780 | 4,680 | 5,250 | 7,790[b] | | 4,500 |
| Fixed carbon in ash from two Duffs, per cent | 29.5 | 27.2 | 33.2 | 35.8[b] | | 32.3 |
| Fixed carbon in ash from mechanical, per cent | 14.2 | 11.9 | 8.0 | 25.1 | 17.5 | 15.3 |
| Dry ash to dry coal, mechanical, per cent | 16.5 | 15.8 | 15.4 | 11.4 | 13.4 | 14.6 |
| Dry ash to dry coal, two Duffs, per cent | 22.4 | 20.8 | 23.0 | 20.5[b] | | 21.5 |
| Ash in coal by analyses, per cent | 10.3 | 11.7 | 8.9 | 10.5 | 11.8 | 10.6 |
| $H_2O$ in coal by analyses, per cent | 8.7 | 8.5 | 6.6 | 8.8 | 10.4 | 8.6 |
| Average CO in gas for mechanical, per cent | 18.7 | 20.3 | 20.2 | 19.1 | 17.8 | 19.2 |
| Average $CO_2$ in gas for mechanical, per cent | 6.0 | 5.2 | 5.8 | 6.6 | 7.1 | 6.1 |
| Average CO in gas for two Duffs, per cent | 17.2 | 20.8 | 18.8 | 18.9 | 18.0 | 18.7 |
| Average $CO_2$ in gas for two Duffs, per cent | 5.4 | 5.1 | 5.4 | 5.8 | 5.1 | 5.4 |
| Steam pressure for mechanical, pounds | 40 | 41 | 40 | 42 | 41 | 41 |
| Steam pressure for two Duffs, pounds | 13 | 13 | 13 | 13 | 13 | 13 |

[a] Mechanical was shut down until 11 A. M. to clean flues; this also accounts for poor gas on fifth day.

[b] Ash hoist out of order on fourth day.

## Depth of Ash and Fire Bed

The depth of bed varies with the kind of coal fired, lump or fine, and with the amount of ashes removed and the time they are taken out. The height the arm has lifted above the lowest position gages the depth of the combined ash fire and fuel bed. The center of the arm at the lowest position is 14 in. above the top of the grates. The center of the arm is about 4 in. below the top of the fuel bed in average running. This gives 18 in. to be added to the amount the arm is lifted to obtain the depth of combined bed above the top of the grates.

The depth of the fire bed was obtained by putting a ¾-in. (19-mm.) pipe 13 ft. (3.9 m.) long through a poke hole and forcing it through the fire bed. The pipe was left in until heated a bright red at the fire zone, and the red part measured after the pipe was withdrawn. The results obtained can only be approximate.

| | Arm Lifted, Inches | Depth of Total Bed, Inches | Depth of Fire Zone, Inches |
|---|---|---|---|
| Before breaking fire | 10 | 28 | 12 |
| 9 A. M., ash bed not leveled | 4 | 22 | 5 |
| 12 noon | 3 | 21 | 6 |
| 3 P M | 6 | 24 | 8 |
| 6 P M | 9 | 27 | 10 |
| About 1000 lb. wet ashes removed 8 P. M. | | | |
| 9 P. M | 9½ | 27½ | 10 |
| 12 midnight | 11 | 29 | 11 |
| 3 A. M. | 12 | 30 | 11 |
| About 1200 lb. wet ashes removed 3.30 A. M. | | | |

## Comparative Labor Cost

The costs given are based on present wages. The cost on the producers with mechanical tops is taken from the one that was installed. Fig. 9 shows the arrangement of producers.

Before Mechanical Top was Installed:

| | |
|---|---|
| 1st shift, 12 men at $3 96 | $ 47.52 |
| 2 head men at $4.04 | 8.08 |
| 2d shift, 5 men at $3.96 | 19.80 |
| 2 head men at $4.04 | 8 08 |
| 3d shift, 5 men at $3.96 | 19 80 |
| 2 head men at $4 04 | 8.08 |
| Total, 28 men | $111.36 per day |

With One Mechanical Top:

1 man at $3.96 less than above on 1st shift (otherwise the same) . . . . . . . . . . $107.40 per day

Proposed Layout, Five Additional Tops:

| | |
|---|---|
| 1st shift, 6 men at $4.25 | $ 25 50 |
| 1 man at $4.04 | 4 04 |
| 1 man at $3.96 | 3 96 |
| 2 men at $3.52 | 7.04 |
| 2d shift, 3 men at $4.25 | 12.75 |
| 1 man at $4.04 | 4 04 |
| 3d shift, 3 men at $4 25 | 12.75 |
| 1 man at $4.04 | 4 04 |
| Total, 18 men | $74.12 per day |

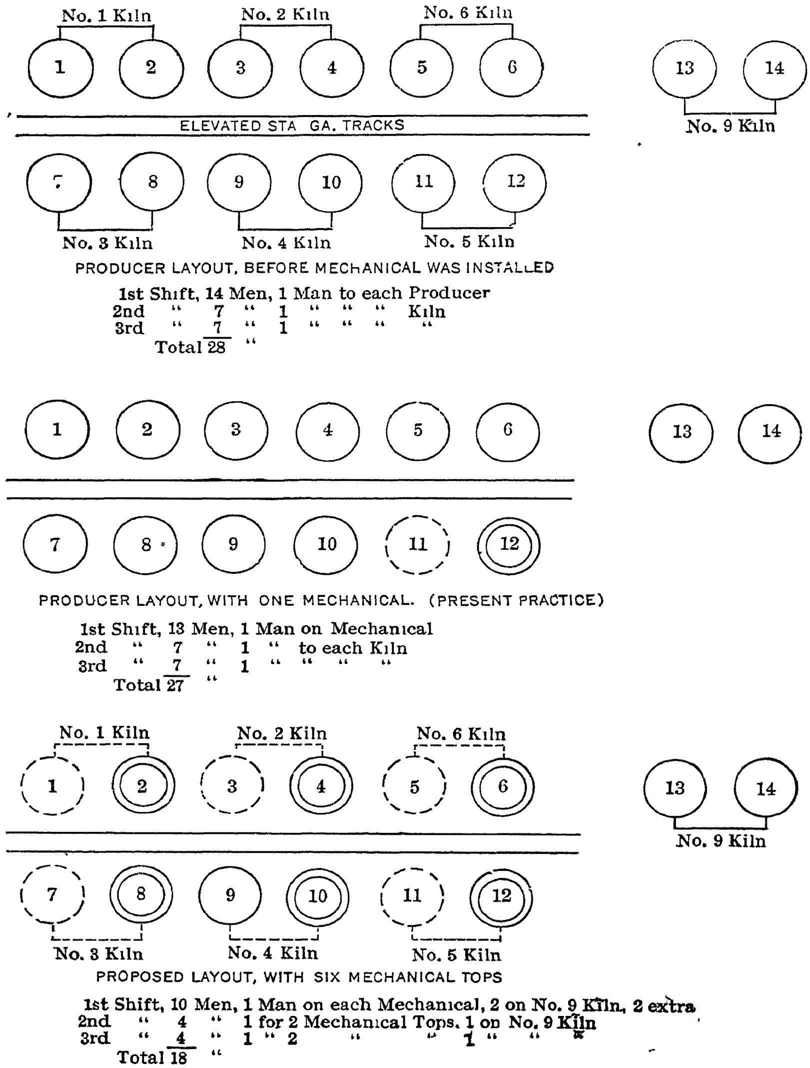



Fig. 9.

*Swindell Producer, Water Bottom.*—This producer consists of a water-seal bottom, thus differing from the Illinois Zinc Co. producer, and has built into its sides a bosh below which are set the inclined cast-iron grates. The blow is introduced at the bottom of the grate, which continues at the inclination of the narrowing sides. The feed hoppers are of the sliding box type. This producer is used at Mineral Point, Wis. Fig. 10 shows the general dimensions and arrangement. At this plant they are gasifying 5 or 6 tons per day, which gives 10 lb. per sq. ft. (48.8 kg. per sq. m.) of area per hour, or practically the same rates as at

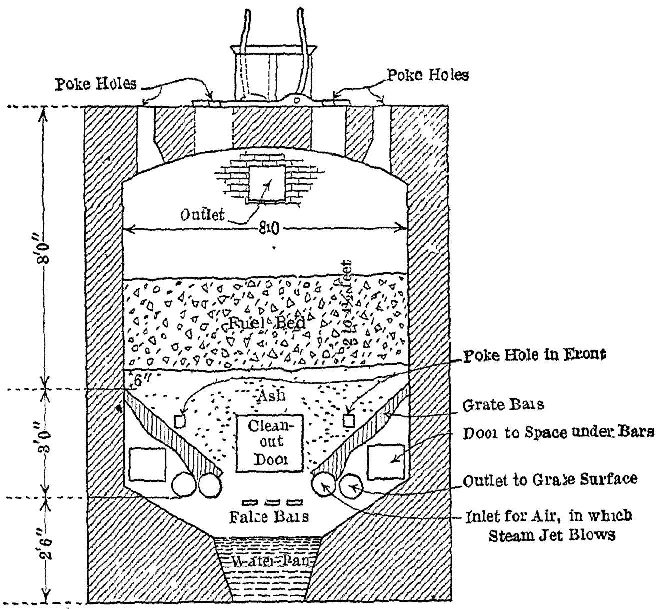



FIG. 10—SWINDELL PRODUCER.

Depue. Screenings through 1¼-in. (31.75-mm.) mesh are used as fuel, these coming from Granville and Cherry, Ill. In addition, Southern Illinois run-of-mine coal is being used. The experience with washed "nut" showed unusually good results, but the cost made this material prohibitive.

The producers are cleaned down once in 24 hr., at which time the false bars are driven in through the ash bed, and the work is carried on much the same as at Peru. The average thickness of ash below the blow is at first 3 to 3½ ft. (0.9 to 1.0 m.). It is intended to leave 6 in. (15 cm.) of ash over the top of the grates. This, of course, increases as the shift progresses. The fire bed itself varies from 2 to 4½ ft., and its poking is left to the judgment of the operator, rather than to regularly timed periods. The gas made runs about 5.4 per cent. $CO_2$ to 23.0 per cent. CO after the producer has settled down to regular work. The

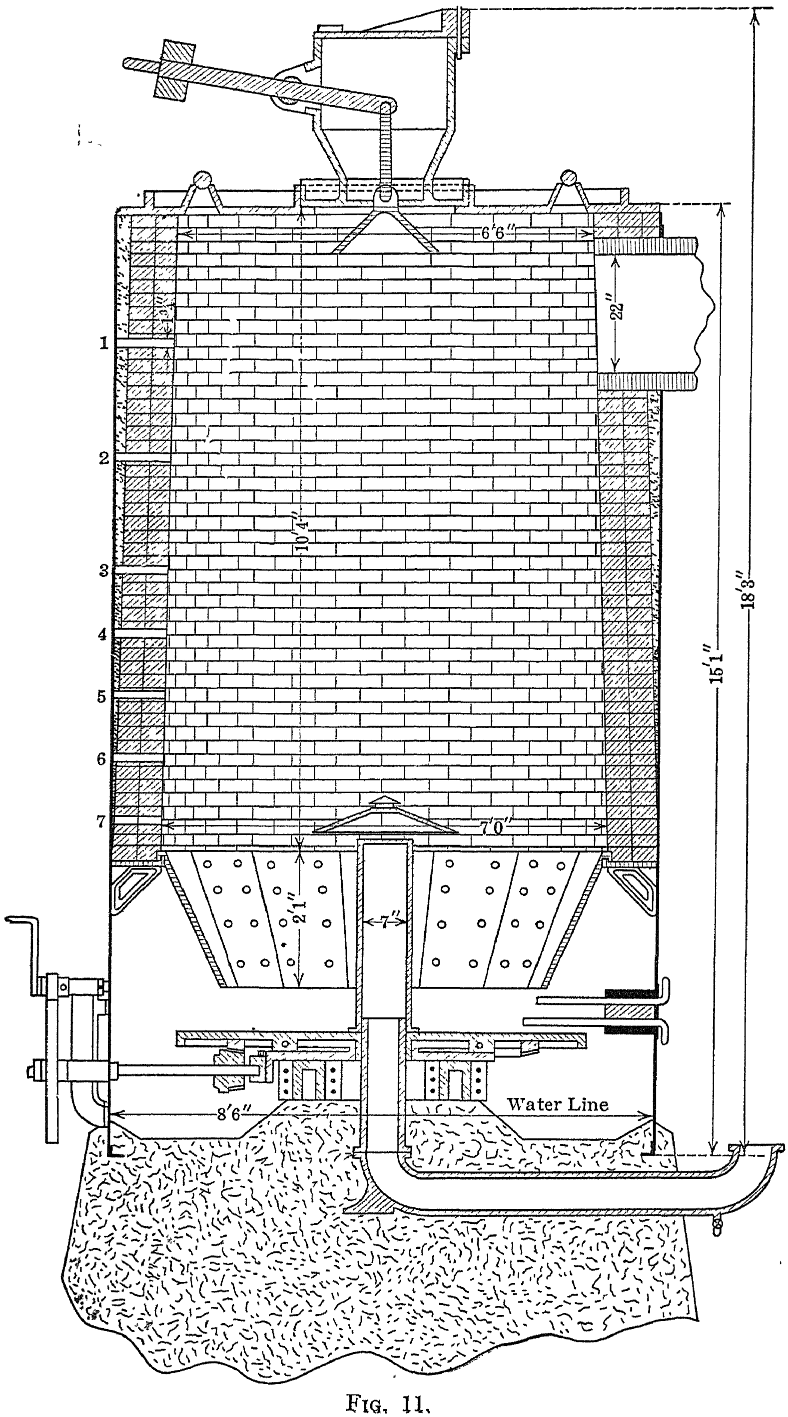



FIG. 11.

blowing apparatus consists of a ½-in. pipe drawn to a ¼-in. opening and projected into a 6-in. cast-iron pipe, which in turn carries the blow into the internal pipe.

## Dry-bottom Mechanical Producer

*Hughes.*—The Hughes dry bottom, Fig. 32, was the first mechanical producer to be successfully adapted to the western slacks. An early type of the revolving producer, which was not a success on Illinois coals, was a trial of two Taylor dry bottoms on one block of 800 retorts at the Illinois Zinc Co.'s plant about 1892. Fig. 11 shows this producer.

At the Depue works, where the Hughes was first tried on this material in 1912, some time was required to develop operating methods, which permitted this producer to handle the varying grades of fuel. This type is now seen at both Depue and Granby. The operation becomes at once one in which maintenance is nearly as important as labor. Not that proper handling is less important; but the total man-hours per ton burned is much less and the upkeep of the apparatus greater. With this machine, six modifications have been made to meet the conditions and fuels at Depue.

1. The cast-steel spiders that support the central blow formerly caused some trouble from failure. This occurred only when handling the lower grade coals, which were high in ash. It was believed at first that proper cleaning down, *i.e.*, not drawing down the ash bed so low, would stop this trouble. However, even with the most careful watching, conditions of great variation in the quantity of ash have developed and, occasionally, at such times the best of producer men would draw down hot clinker and coke and soften these castings. A sudden change in this ash content is very apt to change the thickness of the combustion zone and the depth of the ash above the hood. Usually, at such times the producer is "off feed" somewhat and the clinker may be hard to break down evenly. Then there is a tendency for some of the coke to "run," in other words, to rill down with ash where it does not belong. Any producer man is likely to pull his ashes either too little or too much; in the latter case the castings suffered.

In studying this, it was clear that uniform, better grades of coal would probably cause little annoyance at this point. It was, however, the aim at Depue to tend more and more toward the less desirable coals and endeavor to build up a producer practice that could handle lower cost materials. For this reason, a steam-cooled box girder section was worked out by the Mineral Point organization and the Wellman-Seaver-Morgan Co.; this is shown in Fig. 12. At the cleaning-out period, about 5 lb. pressure is kept on the blow, which insures a circulation of steam

and air up through the central blow column, which in turn aids in cooling the spiders. In 3½ yr. we have had no trouble from these arms.

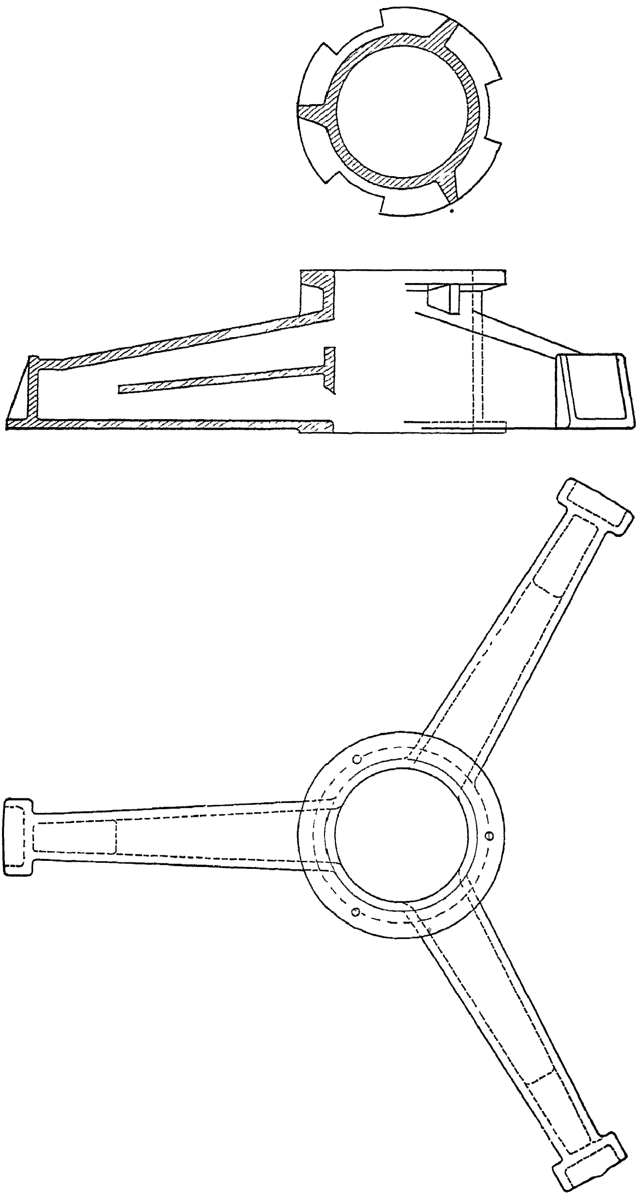


FIG. 12.—DETAILS OF AIR-COOLED SPIDER, HUGHES GAS PRODUCER.

2. At the bottom of the brick lining is hung a cast-steel apron that extends to the ash tray and is supported upon the main frame by a projecting flange. In the case of warping, this apron ring, which is made in segments,

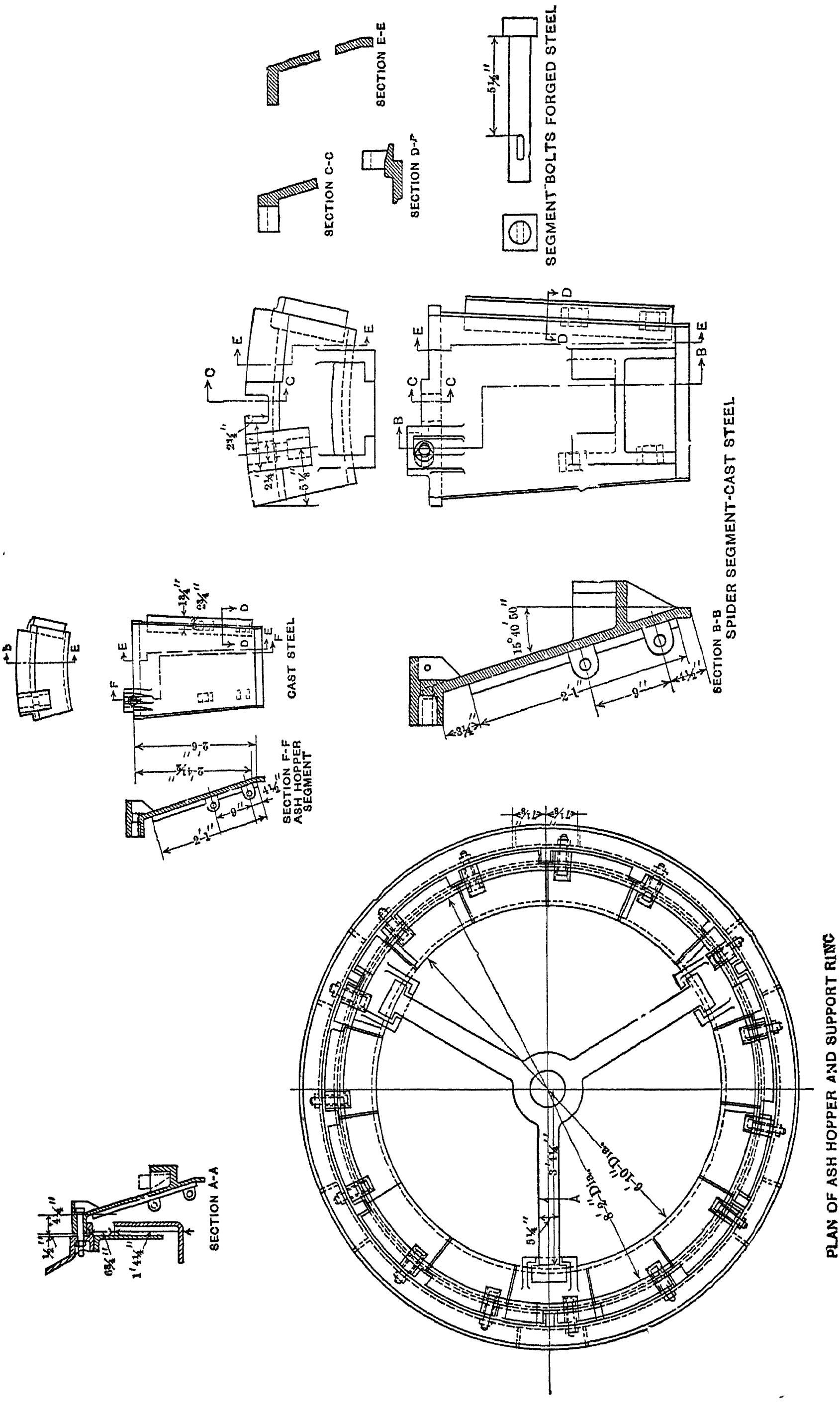



FIG. 13.—DETAILS OF SEGMENTS FOR ASH HOPPER, HUGHES GAS PRODUCER.

has so distorted its upper circle as to allow the entire ring to drop upon the ash pan. Of course the hood must follow. In order to make this more sturdy, a change has been made in the fastening by adding steel clips

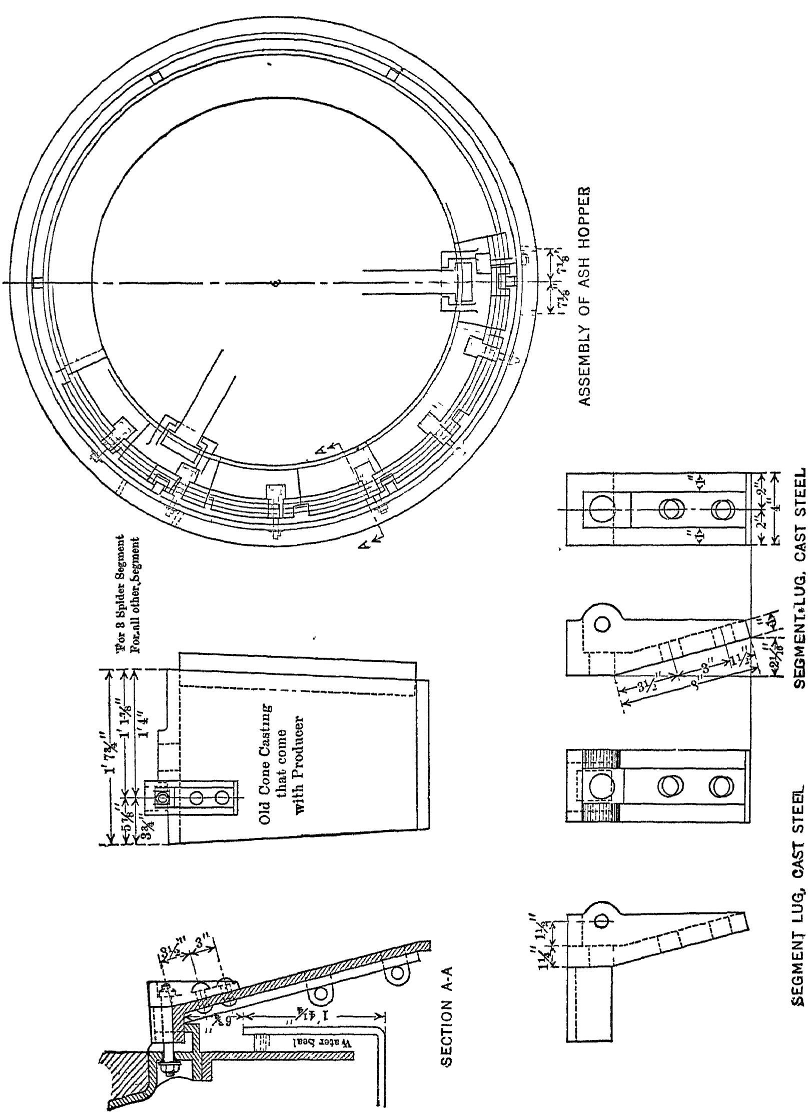



FIG. 14.—DETAILS OF SUPPORT CASTINGS FOR ASH-HOPPER SEGMENTS THAT COME WITH HUGHES GAS PRODUCER.

and bolts at the top, as shown in Fig. 13. The original design is shown in Fig. 14, with clips to hold the old-style plates in place.

3. In mending poker ends, which have a life of from 5 to 6 mo., the following method has been adopted: Castings of steel are made of vary-

ing lengths, as shown in Fig. 15, to take care of the different amounts worn off. The worn poker is machined to a square end and the casting held with a Thermit weld. Of course these pokers wear more on the side that approaches the lining at the end of the stroke; at this point the abrasion is much greater from wall clinker.

4. The blow nipple, which is usually installed at 5⁄16 in. (7.9 mm.), has been changed to 5⁄8 in. (15.8 mm.); the steam consumption can now be held low by operating at a lower line pressure, and at times of emergency there is less sluggishness in driving a producer up to a higher rate of gasification.

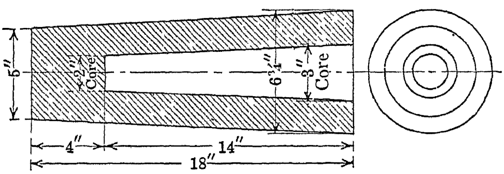



Fig. 15—Cast steel poker tip.

5. All overflows from the top carrying the cooling water are protected by vertical skimmers, much like those used on jig-screen discharges. This is to prevent the tar and soot accumulations from choking the pipes.

6. A flat-plate top modification was found necessary for the blower.

With coals that are high in tars, it is well to carry an extra poker, cross-head, and supporting trunion so that when it becomes necessary to change pokers there need be little delay from sticking in taking the head to pieces.

The Wellman-Seaver-Morgan Co. is now putting out a poker with a tip of high-carbon steel that is screwed into the end of the casting. It is the aim to twist the poker in the trunnion every 3 mo., by which the life of an end may be lengthened. Of course it may be rather serious when one of these pokers begins to leak during a shift. At such an emergency, if the water is cut off and the poker pulled at once, hand poking, if promptly and properly done, will carry the shift along until a new poker can be substituted. Sometimes, by cutting the flow to the poker to a minimum, so that the leakage into the fire bed is negligible, the poker will hold up until the end of the shift. It is convenient, in handling these repairs rapidly, to incorporate in the building frame a rail and crawl over the center line of the poker trunnions. From this depends a chain block, from which the castings are lowered into place.

The labor of "breaking down" a Hughes is shown by a typical motion study on Fig. 16. Fig. 17 is a diagram of the producer operating conditions, showing the various fluctuations as handled on one Illinois slack. Figs. 18 to 21 show the work of a battery of one Hughes and one Duff making gas for a 760-retort furnace.

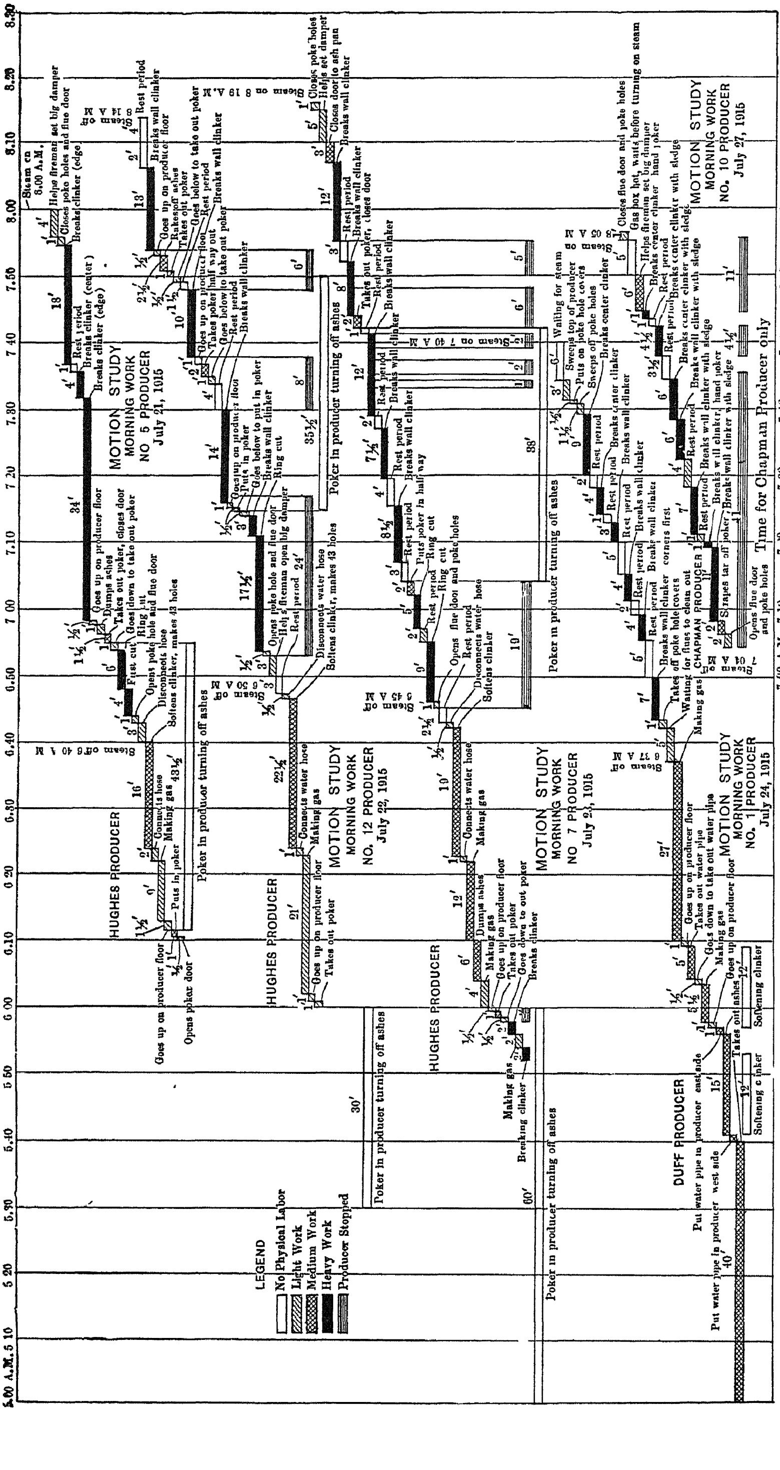



FIG. 16.—MOTION DIAGRAMS OF MORNING WORK ON PRODUCERS.

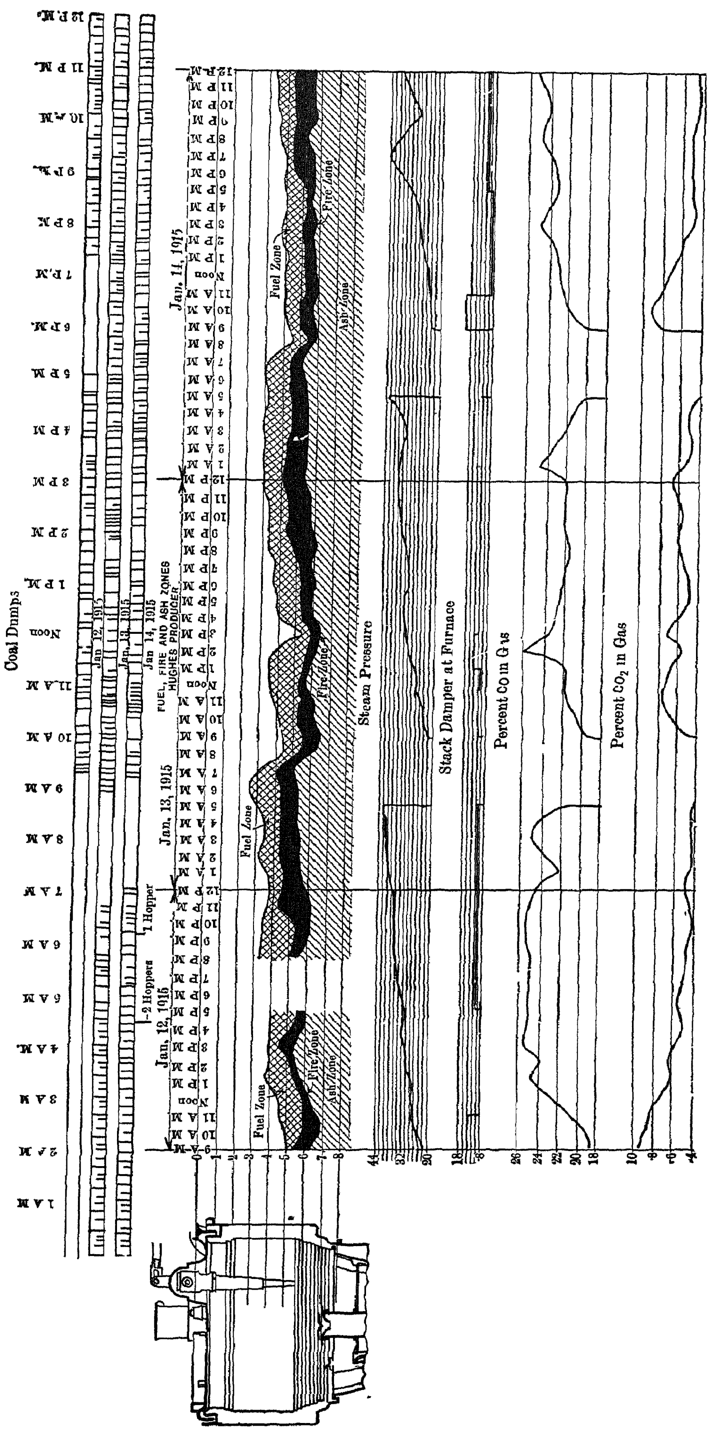



FIG. 17.—TEST ON HUGHES PRODUCER

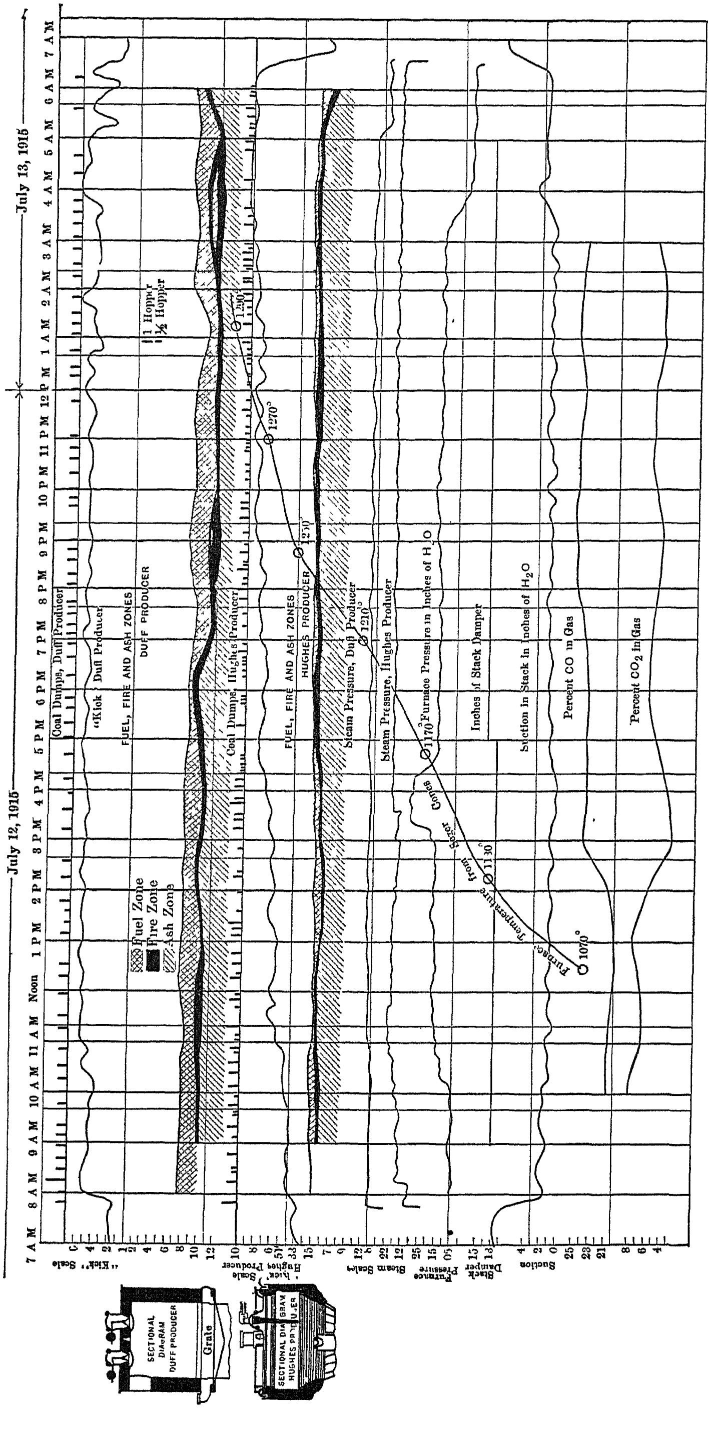



FIG. 18.—PRODUCER OPERATION, NO 5 BATTERY.

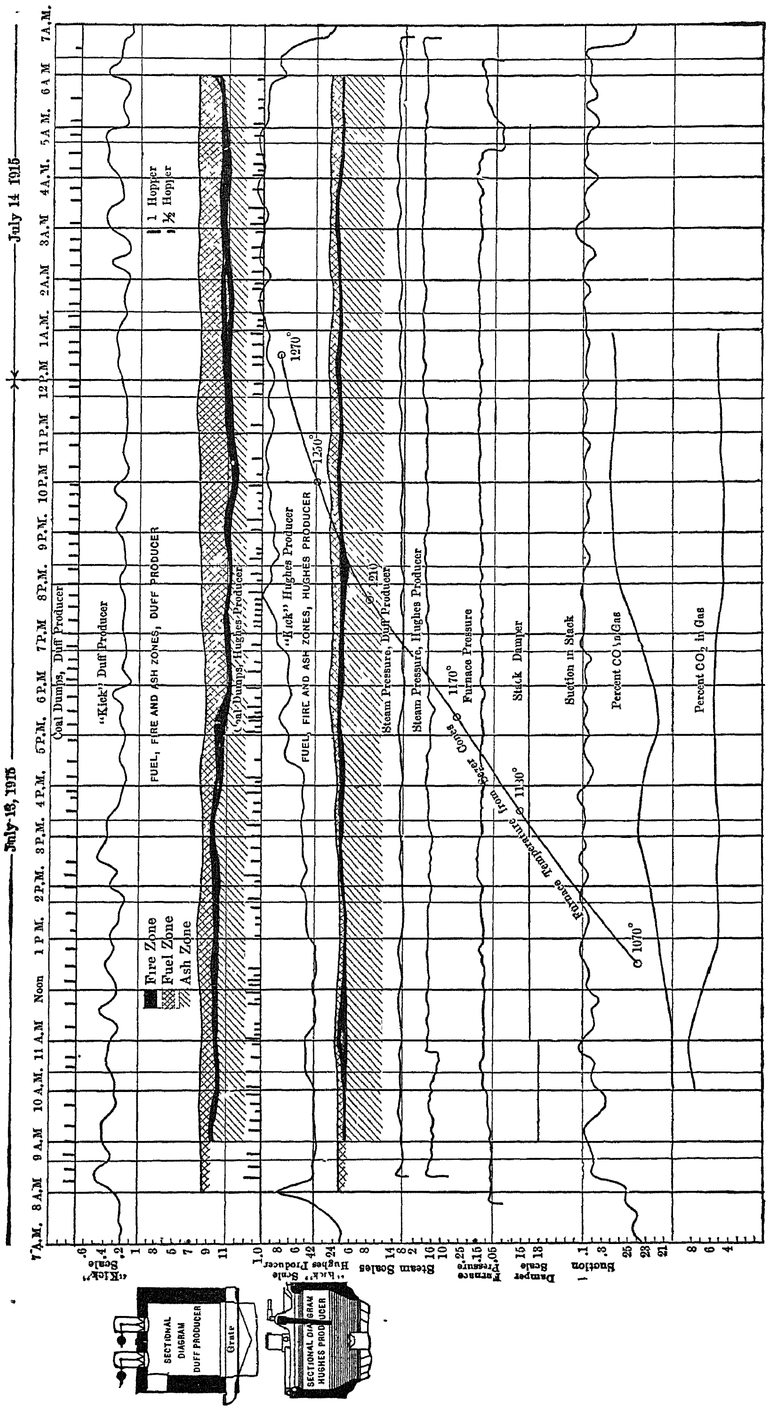



FIG. 19.—PRODUCER OPERATION, No. 5 BATTERY.

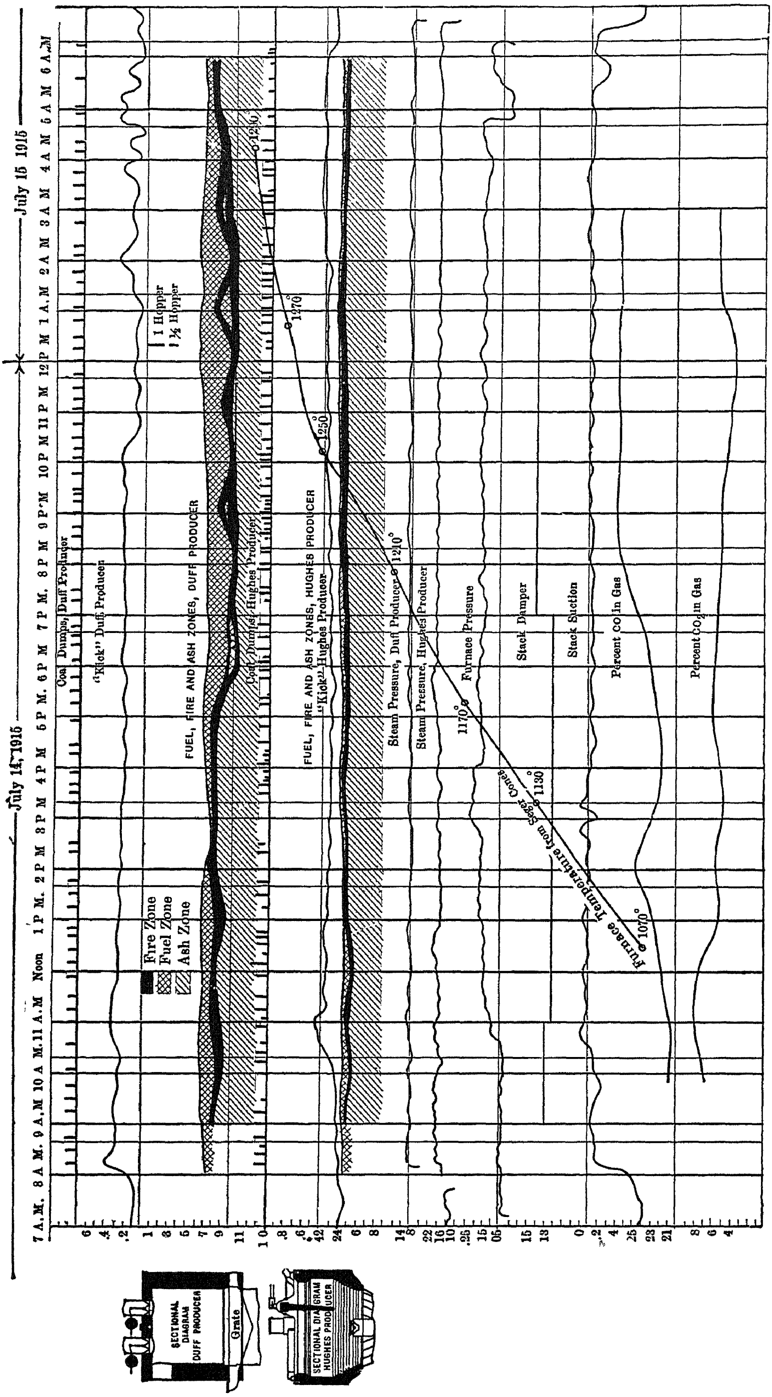



FIG. 20.—PRODUCER OPERATION, No. 5 BATTERY.

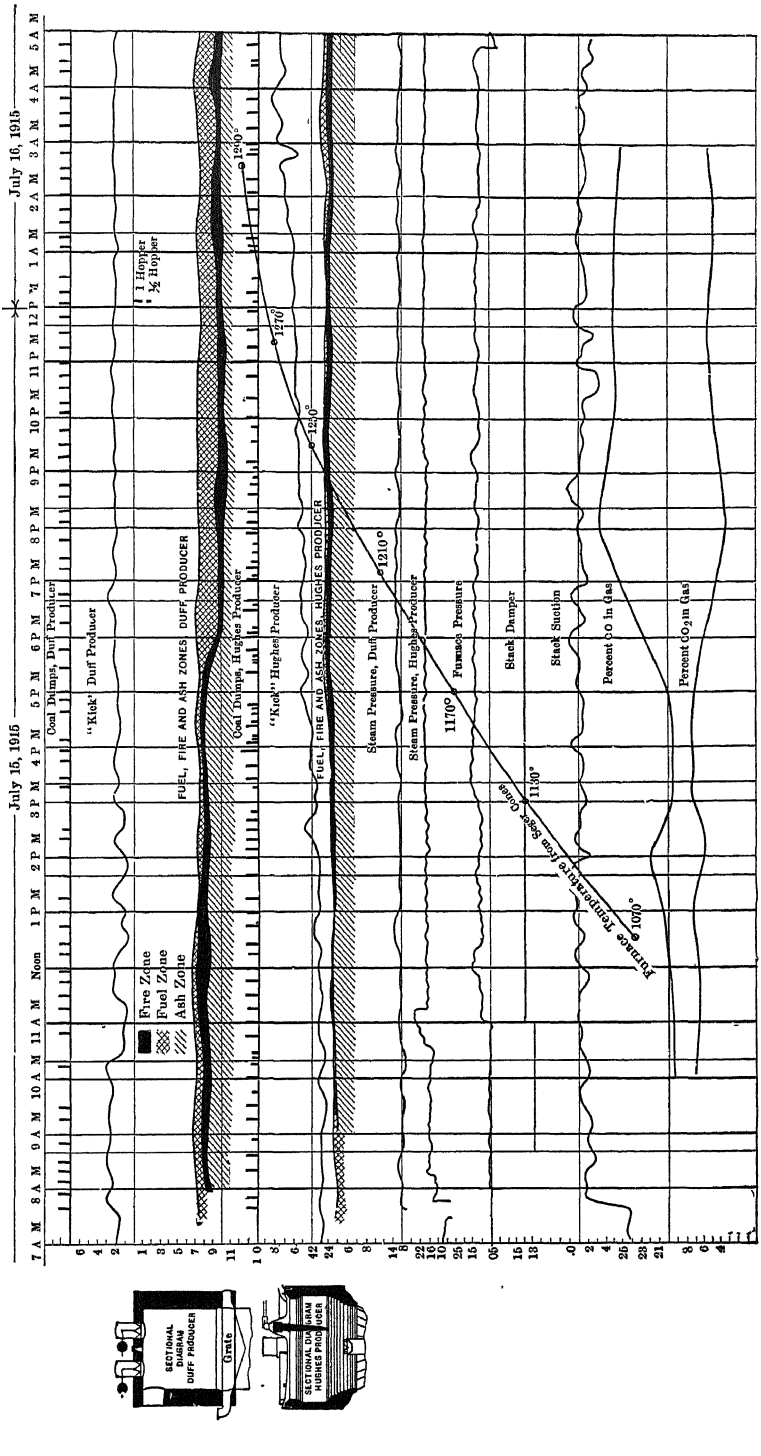



FIG. 21.—PRODUCER OPERATION, NO. 5 BATTERY.

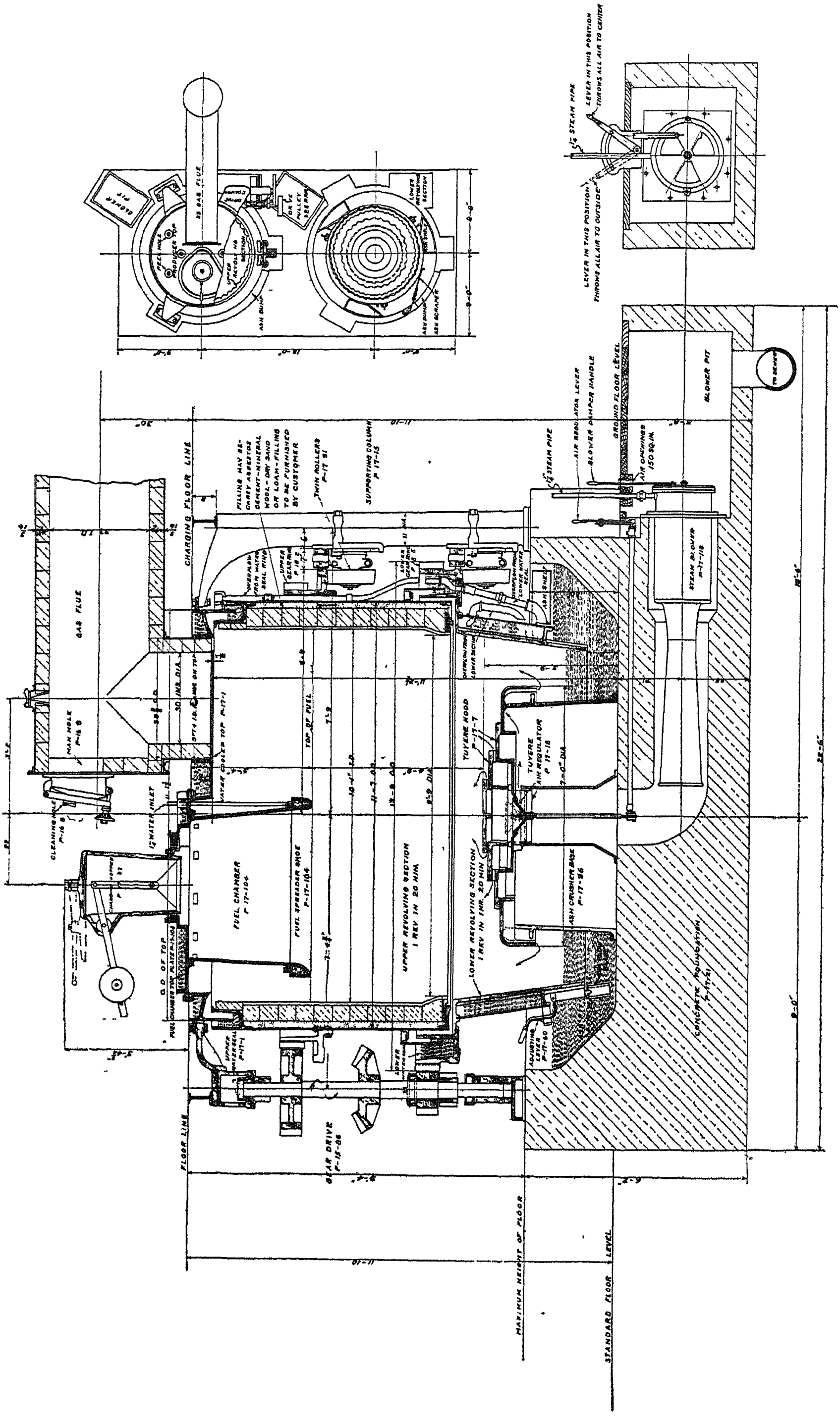



FIG. 22.—CHAPMAN 10-FT. ROTARY GAS PRODUCER.

## Wet-bottom Mechanical Producer

*Chapman.*—The construction of the Chapman producer is shown in Fig. 22. The two-speed shell gives a twisting to the fuel bed, while the stationary tuyere at the bottom functions also as a crusher of clinkers. These are subsequently raised by plows from the water scow with plows attached to the revolving shell. A magazine feed at the top serves also as a fuel-bed leveler, which latter operation is aided in some cases by a slicing loop hanging from the producer top into the fire bed. These producers are used at both Depue and E. St. Louis, and four have been installed at Peru. The maintenance of these machines is the only important point not yet fully determined. Should the abrasion incident to keeping a fuel bed open include both the central hood or side plates instead of an easily replaced poker, there will be some question as to its fitness under long service on western coals.

The chief difference in the handling of this producer and the Hughes is mainly in: Continuous discharging of ash, regulation of beds, and feeding.

Although the producer is stopped for some time at least once at the beginning of each shift, there is seldom much done in normal clearing out of ash or accumulated clinker from the preceding shift. In fact, the producer remains in a quiescent state and the beds are accordingly not disturbed. Should the routine work on the valves or flues require longer than usual, it is well to allow the producer man to put on a small amount of blow with air "full on" in order to maintain a proper bed temperature for efficient work when the furnace starts. The principal means for controlling the proper ratio of bed depths are: Rate of ash withdrawal by changing angle of the plows, rate of feed, and distribution of blow on tuyere.

Hand poking is seldom required with a proper location of the ash and fire beds in relation to the twisting zone. The first adjustment requires some patience to work out, but with watching can be handled within fairly close limits.

A change added by the Chapman Co. is the Shelby steel water-tube drag, which is counter-weighted and hung into the top. It functions both as an agitator of the green-coal bed and a fuel leveler or distributor. With the tar and soot of a cool top, this is apt to give trouble from sticking at the higher level. With some slacks, the ash-bed resistance is so great from packing and clinkering that it is impossible to blow; in which case it is out of the question to carry the top of the fuel bed up to the magazine level. At such times, the combustion zone is pulled well down over the hood and the feed does not function as a spreader or magazine.

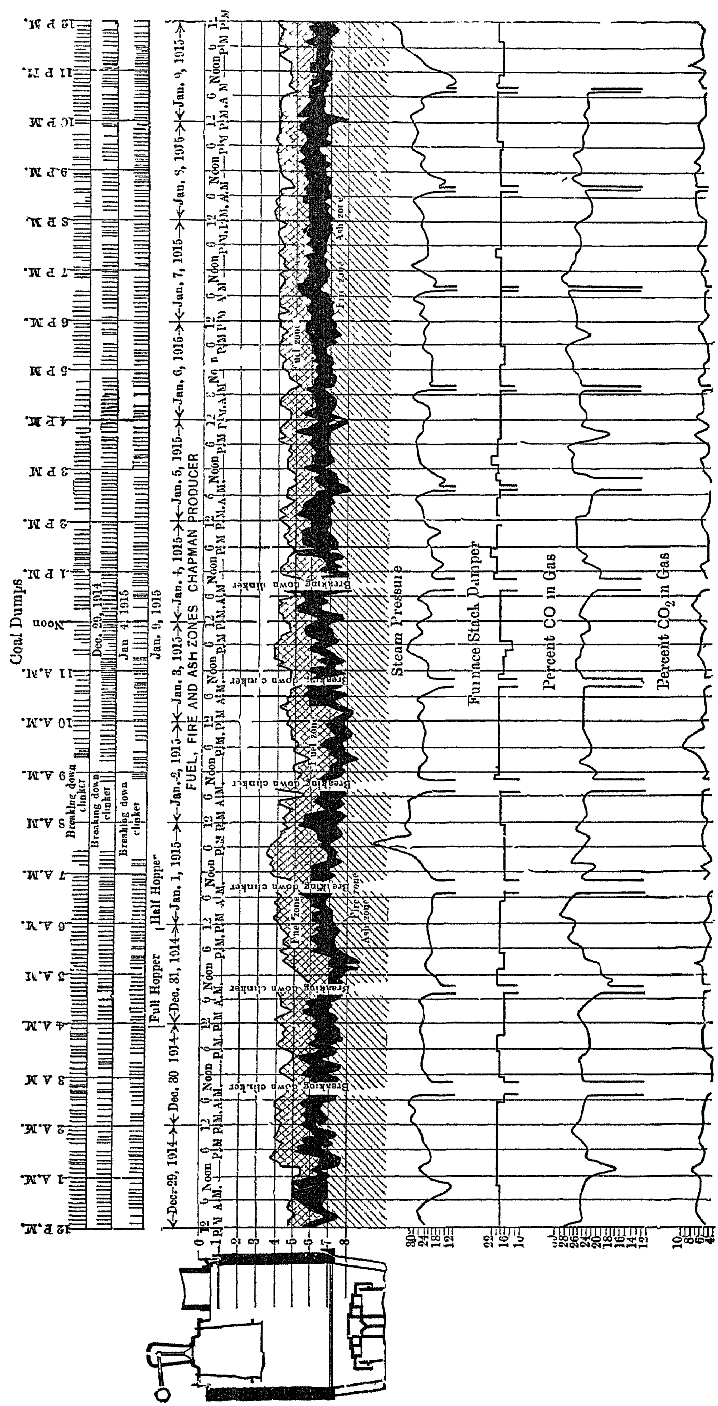



FIG. 23.—TEST ON CHAPMAN PRODUCER.

For a period of 41 days on one producer no real breaking down of the producer was required, and this on Illinois slack. Fig. 23 shows a detailed study of this operation. The double feed hopper, which is a later development, is a real improvement over the original magazine arrangement.

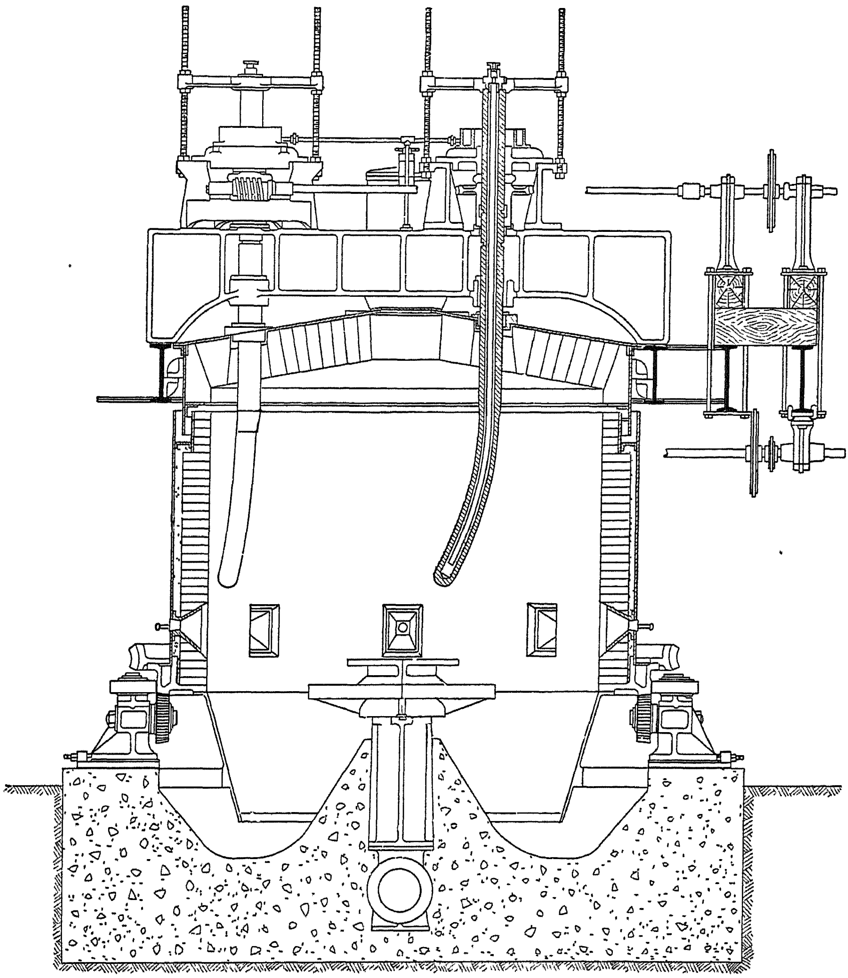


FIG. 24 —MECHANICALLY STIRRED PRODUCER—SECTION THROUGH STIRRER BARS.

*Wood Producer.*—The general make-up of the Wood producer is shown in Fig. 24, embodying a stationary top, a revolving shell, an ash pan and water-seal bottom. Through the top of the producer are curved water-cooled pokers, which are raised and lowered through the bed. The ash is discharged in much the same way as on the Chapman. This producer has not the appearance of being very heavily armored against

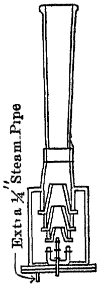
wear. It is said to gasify as high as 30 tons of coal per 24 hr. The Wood producers are in service at Meadowbrook, Burgettstown, Terre Haute, and Taylor Springs. From all reports they have shown a greatly increased efficiency in comparison with the Hegeler generators, which they have generally replaced, requiring 37 to 40 tons of coal per block per day as against 55 with the older type. Satisfactory operation is much easier than on the Hegelers and the maintenance moderate. The life of the poker is about 6 months.

## Blowers

The three appliances used to furnish air for combustion in the producer are fans, steam jets, and turbo-blowers. Of the steam blowers, the Schuette & Koerting of the general Eynon-Evans type is most often seen on Duff installations, while the Chapman and Hughes have their own proportioning of a series of concentric nozzles. Simple steam jets on cast-iron pipe are seen, but these are not of easy or complete regulation, as are the other types. At the best, any of the blowers are not very easy to regulate. With the great variation of the moisture content in the air pumped into a producer, there is no very effective way of keeping the proportioning of water and air correct. Blowers that can quickly meet the exigencies of the work and at the same time maintain the proper ratio are not simple to build. For example, this ratio will vary greatly with a high or low pressure under the beds. A close control can be had only when the blowing apparatus can force through a required quantity of air with a definite percentage of water, regulated in turn by some type of humidifier. This should be possible in spite of a change in bed resistance. Although fair results can be obtained with these crude devices, it is likely that they will be improved, both in respect to economy in horsepower used per cubic foot of air blown, as well as in control of air and water ratios.

At some German installations, a small quantity of producer gas is by-passed to be burned under a steam vaporizer; the vapor is then mixed with the desired quantity of air and blown into the producer. This system would appear at a slight disadvantage compared with using waste steam and a positive blower as proposed in this country. On steam-blown producers, the usual consumption of pounds of steam per pound of coal burned is about 0.4 to 0.6; at Depue 0.5 is a fair average. Certain blower manufacturers are now claiming, however, less steam than this.

Fig. 25 gives some of the data of the three most generally used blowers; some of this is works' information and some furnished by the manufacturer. Figs. 26 and 27 show the construction of those used on the Hughes and Duffs. A point not provided against by the manufacturer is the possibility of small pieces of clinker falling into the blower and plugging




HUGHES BLOWER

STEAM CONSUMPTION—HUGHES BLOWER

| Steam pressure | Size of nozzle | Lb of steam per hour |
|---|---|---|
| *f*25 | ⅜ | 397 5 |
| *f*15 | ⅜ | 240 0 |

*f* Extra ¼" Steam Pipe—One turn of Globe Valve.

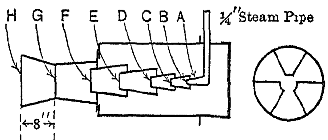
Note: Back Pressure = ¼" of water.

STEAM BLOWER COMPARISON

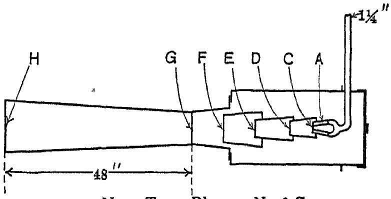
Area of Nozzles

| Section | A | B | C | D |
|---|---|---|---|---|
| Old type... .... | 0.150 | 0 307 | 1 623 | 5 412 |
| New type..... .. | 0 150 | | 1 623 | 5 412 |

| Section | E | F | G | H |
|---|---|---|---|---|
| Old type | 13 36 | 25.96 | 51 85 | 132 73 |
| New type . . | 13 36 | 25.96 | 51.85 | 132.73 |




Old Type Blower-No.6 C



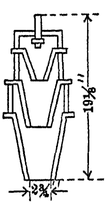

New Type Blower No.6 C

CHAPMAN BLOWERS

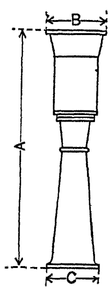
| Steam pressure pounds-gage | Steam temperature Degrees F | Pounds of steam Used per hour | Counter pressure, in of water: Old type, Orifice = 48.708 sq in | Old type, Orifice = 59 532 sq in | Old type, Orifice = 70 356 sq in | New type, Orifice = 48.708 sq in | New type, Orifice = 59 532 sq in | New type, Orifice = 70 356 sq in | Boiler h p, A S.M E. Standard |
|---|---|---|---|---|---|---|---|---|---|
| 20 | 257 | 279 | 1.1 | 0.8 | 0 7 | 1 7 | 1.4 | 1.3 | 93 |
| 40 | 262 | 435 | 2 4 | 1.8 | 1 5 | 3 2 | 2.6 | 2 3 | 145 |
| 60 | 301 | 588 | 3 6 | 2 8 | 2 3 | 4 7 | 3.9 | 3 3 | 136 |
| 80 | 318 | 739 | 4 6 | 3 8 | 3 1 | 6 3 | 5.1 | 4.3 | 246 |
| 100 | 336 | 889 | 5 8 | 4.7 | 3 8 | 7 6 | 6.2 | 5 4 | 293 |

DIMENSIONS OF DUFF BLOWERS

| Diam of discharge | A | B | C |
|---|---|---|---|
| 3 | 19¼ | 7 | 5½ |
| 4 | 26¼ | 8 | 6½ |
| 5 | 34⅝ | 11 | 8½ |
| 6 | 41⅝ | 12 | 9½ |
| 7 | 46¾ | 14 | 11 |
| 8 | 57¾ | 15 | 12 |
| 10 | 65⅜ | 17½ | 13 |
| 12 | 75⅜ | 19½ | 15 |
| 14 | 86¼ | 21½ | 17½ |
| 18 | 111 | 23 | 23 |




SCHUTTE & KOERTING BLOWER




DUFF BLOWER

FIG. 25.—BLOWER DATA, GAS PRODUCERS.

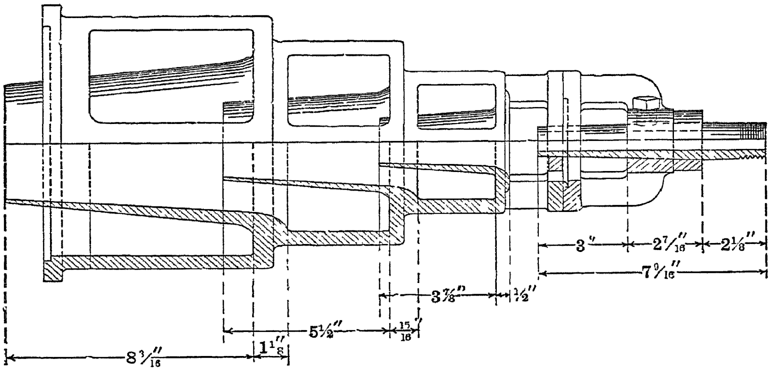



Fig. 26.—Details of steam blower, Hughes producer.

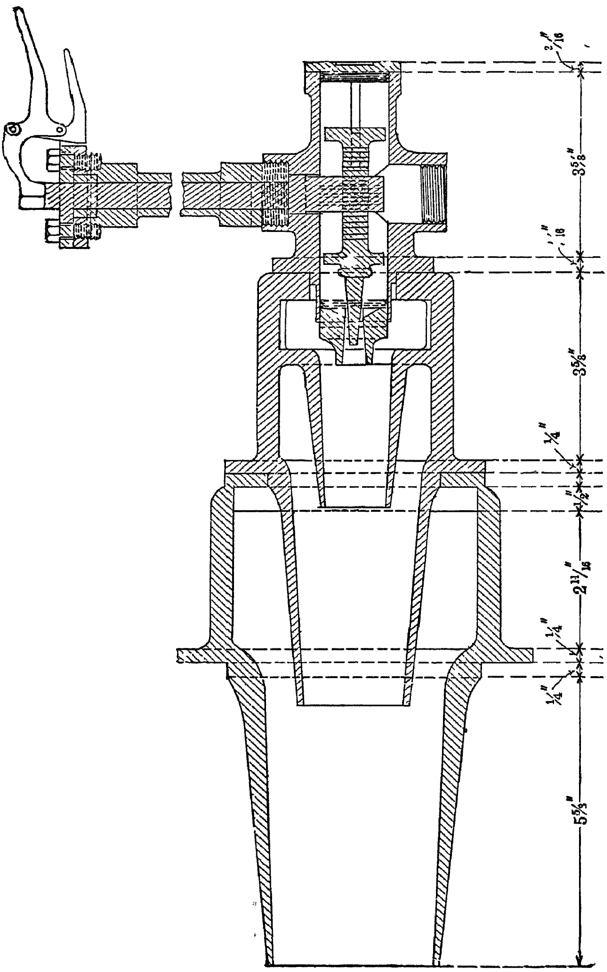



Fig. 27.—Blower regulator, Duff producer.

the nipple. The ordinary blower, after a few months of service, is not easily taken down for cleaning, as it is always rusted. A piping arrangement should be put in that can be quickly and easily removed and the nipple cleaned.

Nassau Form No 15

**MINERAL POINT ZINC CO.**

NASSAU PLANT

**SPELTER DEPARTMENT**

DAILY REPORT OF GAS ANALYSIS FOR Feb. 1, 1915.

| FURNACE NO. 1 | | | | | FURNACE NO. 2 | | | | | FURNACE NO. 3 | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| TIME | $CO_2$ | CO | | | TIME | $CO_2$ | CO | | | TIME | $CO_2$ | CO | | |
| 10 | 6.0 | 27.2 | | | 10 | 5.0 | 24.0 | | | 10 | 7.0 | 21.0 | | |
| 11 | 4.6 | 24.4 | | | 11 | 6.0 | 21.0 | | | 11 | 7.2 | 22.0 | | |
| 1 | 3.6 | 26.4 | | | 1 | 3.4 | 27.8 | | | 1 | 3.6 | 27.0 | | |
| 2 | 6.0 | 24.0 | | | 2 | 5.8 | 21.8 | | | 2 | 3.0 | 27.0 | | |
| 3 | 4.6 | 25.6 | | | 3 | 4.0 | 24.8 | | | 3 | 5.0 | 24.0 | | |
| 5 | | | | | 5 | 5.4 | 26.6 | | | 5 | 5.0 | 26.4 | | |
| 8 | 6.0 | 29.4 | | | 8 | 3.6 | 25.0 | | | 8 | 3.0 | 24.6 | | |
| 10 | 4.4 | 26.4 | | | 10 | 5.0 | 26.0 | | | 10 | 4.6 | 25.4 | | |
| 12 | 4.6 | 26.8 | | | 12 | 4.2 | 24.0 | | | 12 | 4.4 | 25.2 | | |
| 1 | 5.0 | 25.2 | | | 1 | 5.4 | 26.6 | | | 1 | 4.6 | 25.6 | | |
| 3 | 4.6 | 26.6 | | | 3 | 5.0 | 23.8 | | | 3 | 5.0 | 24.0 | | |
| Avg. | 4.9 | 26.2 | | | Avg. | 4.8 | 24.7 | | | Avg. | 4.8 | 24.7 | | |

| FURNACE NO 4 | | | | | FURNACE NO 5 | | | | | FURNACE NO | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| TIME | $CO_2$ | CO | | | TIME | $CO_2$ | CO | | | TIME | $CO_2$ | CO | | |
| 10 | 4.8 | 27.0 | | | 10 | 5.4 | 27.6 | | | | | | | |
| 11 | 6.0 | 26.0 | | | 11 | 7.6 | 23.0 | | | | | | | |
| 1 | 3.0 | 26.2 | | | 1 | 3.6 | 26.6 | | | | | | | |
| 2 | 3.0 | 25.6 | | | 2 | 3.6 | 27.4 | | | | | | | |
| 3 | 2.0 | 23.0 | | | 3 | 4.0 | 27.0 | | | | | | | |
| 5 | 5.0 | 24.2 | | | 5 | 4.6 | 25.4 | | | | | | | |
| 8 | 3.4 | 28.4 | | | 8 | 4.4 | 24.2 | | | | | | | |
| 10 | 4.4 | 27.0 | | | 10 | 4.0 | 25.0 | | | | | | | |
| 12 | 3.4 | 29.4 | | | 12 | 3.6 | 25.6 | | | | | | | |
| 1 | 5.4 | 26.0 | | | 1 | 4.2 | 26.0 | | | | | | | |
| 3 | 3.6 | 26.2 | | | 3 | 4.0 | 25.0 | | | | | | | |
| Avg. | 4.0 | 26.9 | | | Avg. | 4.5 | 25.5 | | | | | | | |

REMARKS

FIG 28.—TYPICAL DAILY SHEET.

## QUALITY OF GAS

With the price of coal low, as it is at most plants, the question of quality of products at the gas plant is not always given the closest at-

tention. There is, however, a growing tendency to keep more complete data on this point. There are certain aspects of the subsequent combustion in the furnaces and kilns that make the quality important, regardless of the question of a little more or less coal burned. With this in mind, some plants sample the gas regularly and even pay bonuses based on analyses. To those who have practised this over a period of several years, the results are seen to have easily justified the expense of such controls. These gains are incontrovertible, both as to ultimate fuel economy and metallurgical practice. In effect, they are valuable from two angles: first, a comprehensive knowledge to those in charge of the average working condition of the gas plants from month to month and, second, a help to those who are actually doing the work on the producer floor.

At Depue, samples are taken at 2-hr. intervals and reported to each individual battery and to the gas-house foreman, while the entire daily results are entered for record. A modified Hempel method is here used, with mechanical-shaking apparatus and such conveniences that one boy can handle all the work on a twelve-furnace plant. That each producer crew may shape up the work extra well just before sampling periods is natural, but, on the other hand, a fit producer will do passable work for at least 1½ hr. more, so a fair average condition of machine and gas is maintained. Fig. 28 shows a typical daily sheet. In addition, a portable Smith calorimeter is used for continuous work; the principle and design of these instruments is well known. On bituminous-coal gas, additional filters are necessary to keep the machine from fouling on tars, dust, etc. Mineral-wool filters are used as roughers and large size filter paper for the final cleaning.

In testing the accuracy by analysis and computation, the most reliable values for the heats of combustion of the different constituents are assumed to be those used by the Bureau of Standards at Washington. These are the means of values found by a number of observers as given in Landolt-Bornstein-Meyerhofer's "Physikalisch-Chemische Tabellen" 3d Edition (1905). They are said to be reliable to about 2 per cent. These have been corrected to give the so-called low values for those constituents containing hydrogen, as the Bureau values assume that the water formed as a product of combustion is condensed to liquid, which is not the case either in the furnace where the gas is burned or in the calorimeter. These revised values were calculated by deducting the latent of heat of vaporization (at 62° F. or 17° C.) of the water formed by combustion. They are: $H_2$, 275 B.t.u. per cubic foot, 62°F., 30 in. mercury; CO, 323; $CH_4$, 910; $C_2H_4$, 1497.

The following shows a typical calibration after cleaning up the instrument. The latter was adjusted after each setting until the reading agreed with the calculated value.

| Setting | $CO_2$, Per Cent. | $C_2H_4$, Per Cent. | $O_2$, Per Cent. | CO, Per Cent | $CH_4$, Per Cent | $H_2$ Per. Cent | B t u (Calculated) | B t u. (Read) |
|---|---|---|---|---|---|---|---|---|
| I | 6 6 | 0 4 | 0.0 | 23 8 | 3.2 | 13.7 | 150 | 162 |
| II | 4 2 | 0 4 | 0.0 | 25 8 | 2.8 | 13.6 | 152 | 162 |
| III | 5 9 | 0 3 | 0 3 | 24.6 | 2.3 | 12.4 | 139 | 147 |
| IV | 6 8 | 0.4 | 0 1 | 20.9 | 3.8 | 12 2 | 142 | 136 |
| V | 7.2 | 0 2 | 0 2 | 21 6 | 3 1 | 13 0 | 137 | 141 |
| VI | 6 2 | 0 4 | 0 0 | 23.4 | 2 6 | 6.8 | 124 | 126 |
| VII | 6 6 | 0 5 | 0 0 | 24 6 | 2 5 | 12.7 | 145 | 145 |

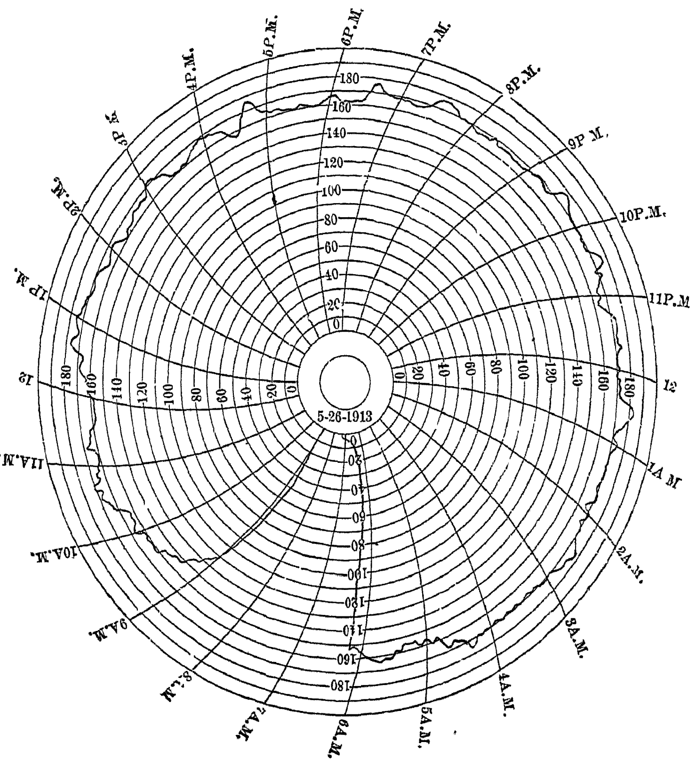



FIG. 29.—TYPICAL CHART.

A typical chart is shown in Fig. 29. For very close watching of the work, and especially for comparative testing of new coals, this calorimeter is valuable. Figs. 30 and 31 show the theoretical relation of the different gas constituents under various conditions of air and steam, assuming pure carbon as the fuel.

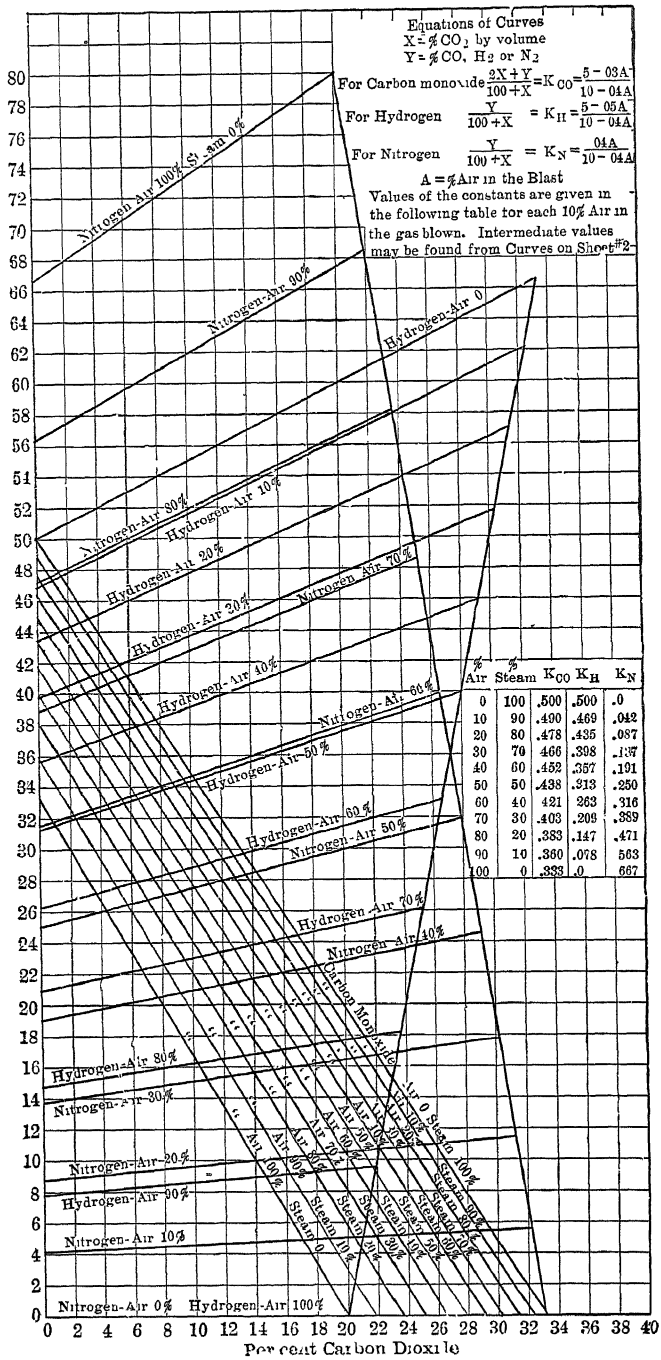



FIG. 30.—COMPOSITION OF PRODUCER GAS FROM PURE CARBON BLOWN WITH VARYING PROPORTIONS OF AIR AND STEAM

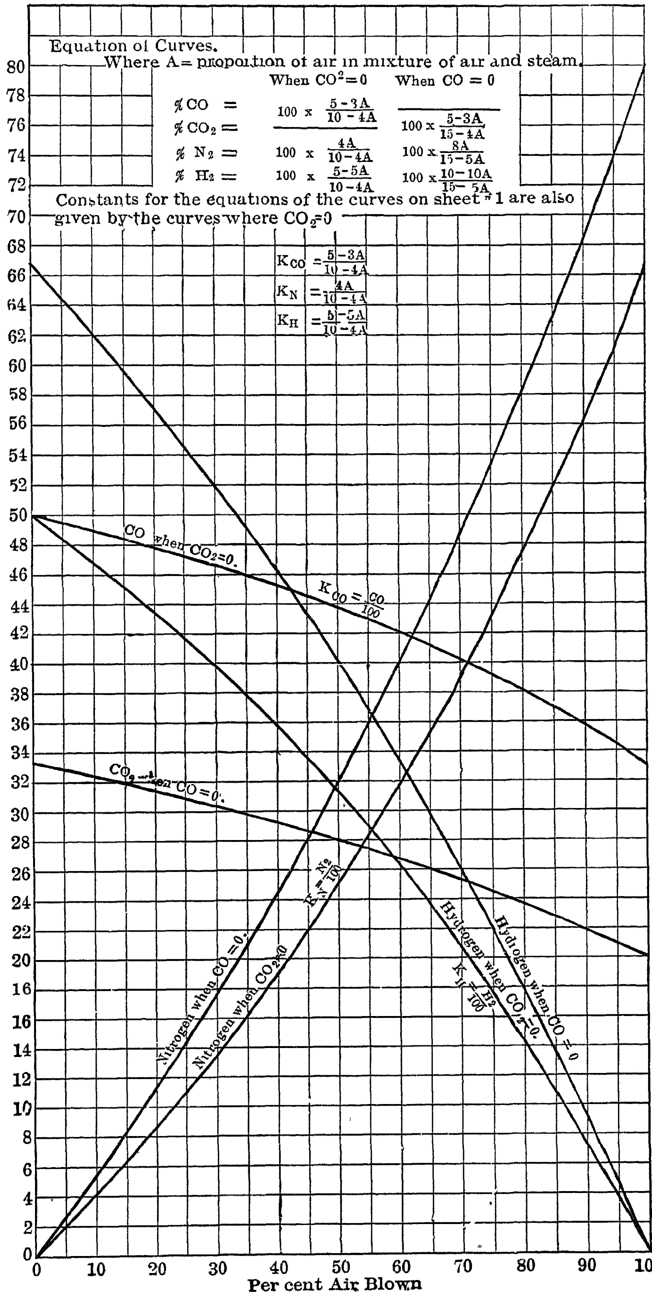



FIG. 31.—MAXIMUM AND MINIMUM PER CENTS OF CONSTITUENTS OF PRODUCER GAS FROM PURE CARBON FUEL BLOWN WITH DIFFERENT PROPORTIONS OF AIR AND STEAM. ALSO CONSTANTS IN EQUATION OF CURVES IN FIG. 30.

## Gas Cleaning

Very little has been done along the line of gas cleaning at any western plant. Where generators are at the immediate ends of furnaces, with

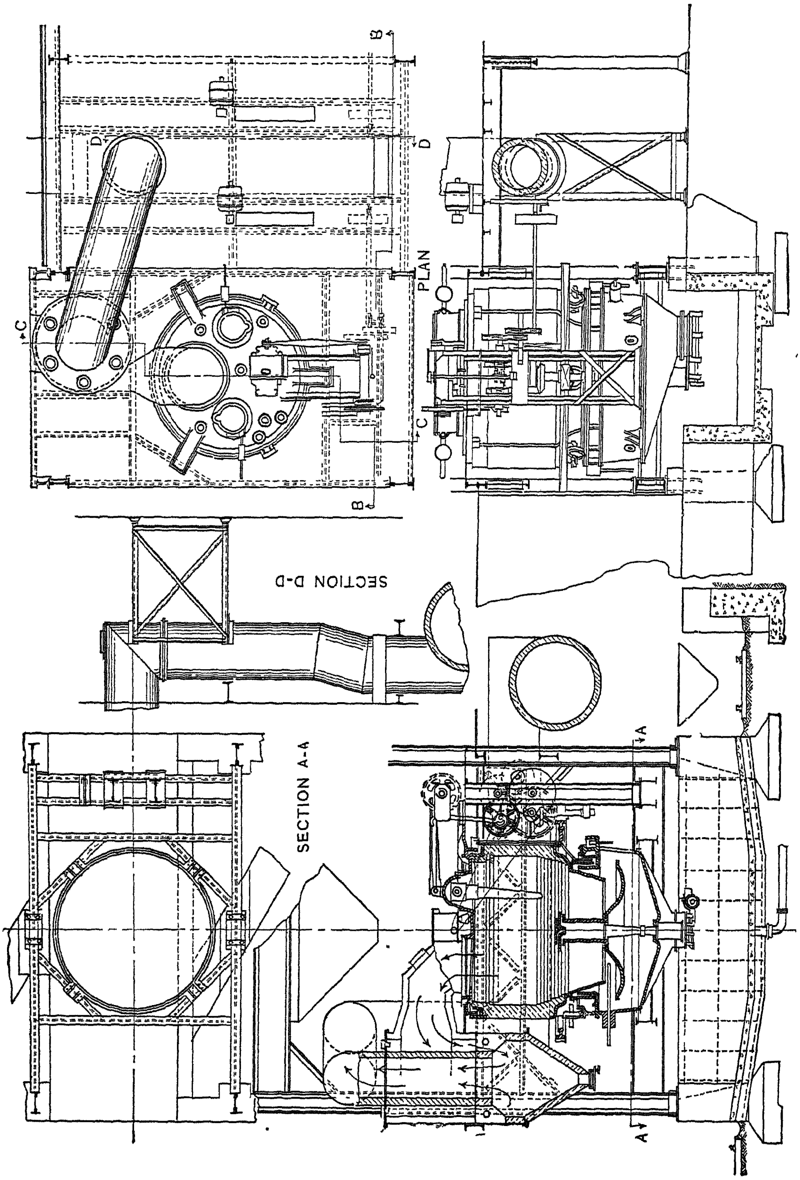



Fig. 32.—General arrangement of Hughes gas producer for No. 2 furnace.

waste-heat boilers for recuperation at the other, there is little trouble from ash, dust, etc. On the other hand, where recuperators are of the checker-work type, with constricted ports delivering gas to the laboratory of the

TABLE 4.—*Results of Test Made on a Hughes Working with One Duff Producer as a Spelter Block Unit*

| | Hughes | | Duff | |
|---|---|---|---|---|
| | Total | Average per Hour | Total | Average per Hour |
| Hours firing producer | 22 33 | | 22 42 | |
| Coal as fired, pounds | 26,460 | 1185 | 16,390 | 731 |
| Dry coal fired, pounds | 22,090 | 989 | 13,686 | 610 |
| Combustible fired, pounds | 19,760 | 885 | 12,243 | 546 |
| Total ash in coal, pounds | 2328 | | 1442 | |
| Carbon in ash, per cent. | 10 | | 10 | |
| Ash removed, pounds | 2587 | | 1603 | |
| Carbon in ash, pounds | 259 | | 160 | |
| Fuel bed area, square feet | 56 75 | | 62 5 | |
| Coal as fired per square foot of fuel bed | 466 | 20 9 | 262 | 11 70 |
| Dry coal fired per square foot of fuel bed | 390 | 17 5 | 219 | 9 77 |
| Combustible fired per square foot of fuel bed | 348 | 15 5 | 196 | 8 74 |
| Average steam pressure, pounds per square inch | 25 6 | | 15 0 | |
| Weight of steam used, pounds | 9088 | 407 | | |
| Coal fired equivalent to steam used | 1300 | 55 8 | | |
| Dry coal fired equivalent to steam used | 1085 | 48 7 | | |
| Combustible fired equivalent to steam used | 970 | 43 4 | | |
| Power used for driving machinery, horsepower | 1 6 | | | |
| Average pressure below fuel bed, inches, water column | 1 6 | | 0 43 | |
| Average temperature of gas, degrees C | 405 | | 610 | |
| Average pressure of gas above atmospheric, inches, water column | 0 1 | | 0 1 | |
| Barometric pressure, inches | 29 6 | | 29 6 | |
| Average temperature of outside air, degrees F | 0 | | 0 | |
| Average temperature cooling water, entering, degrees F | 55 9 | | | |
| Average temperature cooling water, exit, degrees F | 89 8 | | | |
| British thermal units in coal fired | 285,240,000 | 12,774,000 | 176,680,000 | 7,880,600 |
| British thermal units in coal fired and steam equivalent | 299,250,000 | 13,402,000 | | |
| Cubic feet of gas produced, 62° F, 30 in mercury | 1,132,700 | 52,965 | 759,980 | 33,897 |
| Cubic feet of gas per pound coal as fired, in producer | 44 70 | | 46 37 | |
| Cubic feet of gas per pound dry coal fired in producer | 53 54 | | 55 53 | |
| Cubic feet of gas per pound combustible fired in producer | 59 85 | | 62 07 | |
| Cubic feet gas per pound coal as fired, including steam equivalent | 42 62 | | | |
| Cubic feet gas per pound dry coal as fired, including steam equivalent | 51 03 | | | |
| Cubic feet gas per square foot of fuel bed | 20,840 | 933 | 12,160 | 542 |
| Cubic feet gas per pound combustible, including steam equivalent | 57 05 | | | |
| British thermal units from gas | 184,270,000 | 8,252,000 | 109,190,000 | 4,865,900 |
| British thermal units from gas per pound coal as fired in producer | 6964 | | 6656 | |
| British thermal units from gas per pound dry coal as fired in producer | 8342 | | 7971 | |
| British thermal units from gas per pound combustible as fired in producer | 9325 | | 8910 | |

TABLE 4.—(*Continued*)

| | Hughes | | Duff | |
|---|---|---|---|---|
| | Total | Average per Hour | Total | Average per Hour |
| British thermal units from gas per pound coal as fired, including steam equivalent | 6460 | | | |
| British thermal units from gas per pound dry coal as fired, including steam equivalent | 7738 | | | |
| British thermal units from gas per pound combustible as fired, including steam equivalent | 8650 | | | |
| British thermal units from gas per square foot fuel bed | 3,242,000 | 145,410 | 1,745,500 | 77,854 |
| British thermal units from gas, per cent of British thermal units in coal fired | 64 60 | | 61 74 | |
| British thermal units from gas, per cent of British thermal units in coal, including steam equivalent | 61 58 | | | |

AVERAGE GAS ANALYSIS

| | PER CENT | PER CENT. |
|---|---|---|
| $CO_2$ | 4 48 | 5 15 |
| $O_2$ | 0 18 | 0.04 |
| $C_2H_4$ | 0 4 | 0 4 |
| CO | 26 32 | 24 40 |
| $CH_4$ | 3 86 | 3 83 |
| $H_2$ | 10.60 | 8 49 |
| B.t u. per cu. ft., 62° F., 30 in. mercury | 155 8 | 143 6 |

ANALYSIS OF COAL

| | PER CENT | PER CENT |
|---|---|---|
| Moisture | 16 5 | 16 5 |
| Ash | 8 8 | 8 8 |
| Total carbon | 51 2 | 51 2 |
| Sulfur | 3 1 | 3.1 |
| B.t.u. per pound as fired | 10,780 | 10,780 |
| B.t u. per pound dry coal | 12,910 | 12,910 |
| B.t u. per pound combustible | 14,430 | 14,430 |

furnace, and, where long flue lines including reversing valves are included in the heating system, settling dust at convenient points in the system guarantees freedom from plugging at inaccessible bends and makes the clean-up less objectionable. At Depue, two general forms of apparatus have been tried, the first and less efficient is a simple vertical flue, into which the neck of the producer enters as a T. The bottom half is used as a holder for settled dust, while the gas rises in the top part and is run over into the furnace flue. While this removed some of the fine dust, a good deal was carried along the flue and deposited. The diameter

of this settler was the same as the flue, which was a mistake as it neither dropped the gas velocity materially nor gave sufficient impinging of gas on surfaces.

A more successful form is along the lines of a round settler or cyclone. It was not expected that, with the velocities as low as they are known to be, it would throw out much material from centrifugal impact. At the same time, a gently whirling action with the presentation of ample surface was depended on, together with several changes in direction of the flow. This arrangement, shown in Fig. 32, has done very thorough cleaning. For example, a flue line from a mechanical producer that ordinarily required cleaning once a week, when equipped with one of these cleaners, was not cleaned for 6 mo. Each day a few wheelbarrows full of ash dust were run out of the bottom into a narrow-gage car, which was all the flue cleaning necessary. The first cyclones were built with steel risers in the center, but these required replacing after a relatively short service. The center flues are now being built entirely of firebrick, with arches below.

## Efficiencies

The relative fuel efficiency of western producers is difficult to determine. There are too many operations carried on with little or no testing, and perhaps are only required to give general results. A producer must give an abundance of gas, break down easily, and stand up to hard service. As to whether it utilizes the coal to the best possible advantage, that is, gives a minimum consumption, or that its product is the best suited for the work, few are in a position to say. Judging from the appearance of carbon in the ash, there are some serious losses at many plants.

Radiation losses are probably higher on hand-poked producers than on mechanically stirred. This results from the greater depth of combustion zone and hotter tops resulting from blowholes. Of course with M. & H. producers, the loss by radiation from hot gas is much less than with the other types, and the sensible heat is utilized in heating. The recovery of the sensible heat in the gas is limited with recuperative heating systems, for as they arrive at the checkers at a higher temperature, so will the stack gases show a corresponding rise. One disadvantage of a hot top in heat recovery is the breaking up of the hydrocarbons, which should more properly be burned in the laboratory of the furnace. Testing on mechanical producers at Depue indicates a fuel efficiency of 65 per cent. This is upon the basis of British thermal units in coal charged to British thermal units in gas produced.

It is certain, however, that the mechanical machines are the most efficient. An even production of gas and better control of both depths of producer beds and their temperatures is possible. Temperature

charts of the producer beds have been made at Depue. On the Hughes, for instance, these beds show a curving of zones opposite in shape to those worked out on the Taylor at Norfolk by R. H. Fernald of the U. S. Geological Survey.

## Developments

The Morgan producer, Fig. 33, shows evidence of great care in details of construction. If the representations made by the manufacturers are

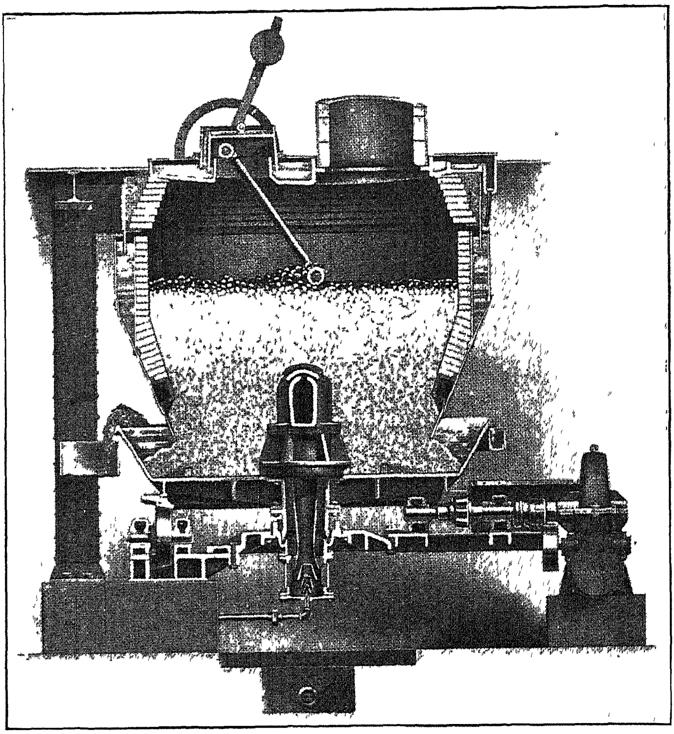


Fig. 33.—Morgan producer gas machine.

dependable, their operating performance on the first installation is excellent. How well the agitation of the bed can be maintained when badly clinkering coals are used is a point not as yet well proved. The manufacturer's statement that a "gas-making fire must not be disturbed" is at least open to question on some western slacks. Water jacketing of the body has been introduced in American practice.

Note should be made of the tendency of producer manufacturers abroad to build producers capable of handling very low-grade fuels carrying over 30 per cent. ash. It was reported, by the Bureau of Mines who went over the situation in 1914–15, that, although these lower grade fuels could be worked, many plants preferred to pay a higher

price for a fair grade of material. Undoubtedly we may look to those countries where good coals are becoming yearly more scarce, for substantial producer development on badly clinkering fuels. It will likely be some time before western plants are obliged to consider any such high-ash coals. However, when some of the beds on the northern edge

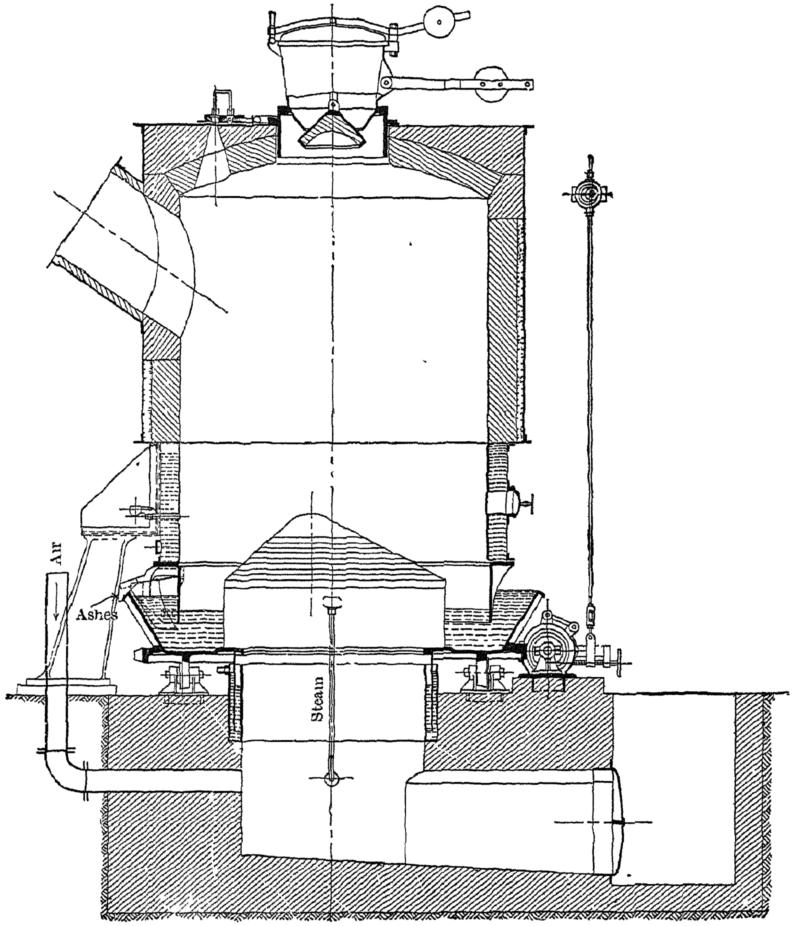



FIG. 34.—SECTION OF PINTSCH REVOLVING ECCENTRIC-GRATE GAS PRODUCER.

of the Illinois basin, for instance, are used, either preliminary washing or a change in producer practice will be necessary.

The two changes in producer construction that have done the most to allow these clinkering coals to be handled are: Water jackets in the walls around the hot zone and revolving eccentric grates.

In the first feature, lower fuel efficiencies may result from a high transfer of heat at the jacket surface. However, this may well off-set

the disadvantage of a very wet blow, which would be required to make the clinker workable. Furthermore, high efficiencies are not probable on this class of material.

The second change, in respect to the agitation of the bed, is to bring the grinding action into play well toward the bottom of the producer. It should be more positive than the shearing zone of the Chapman and

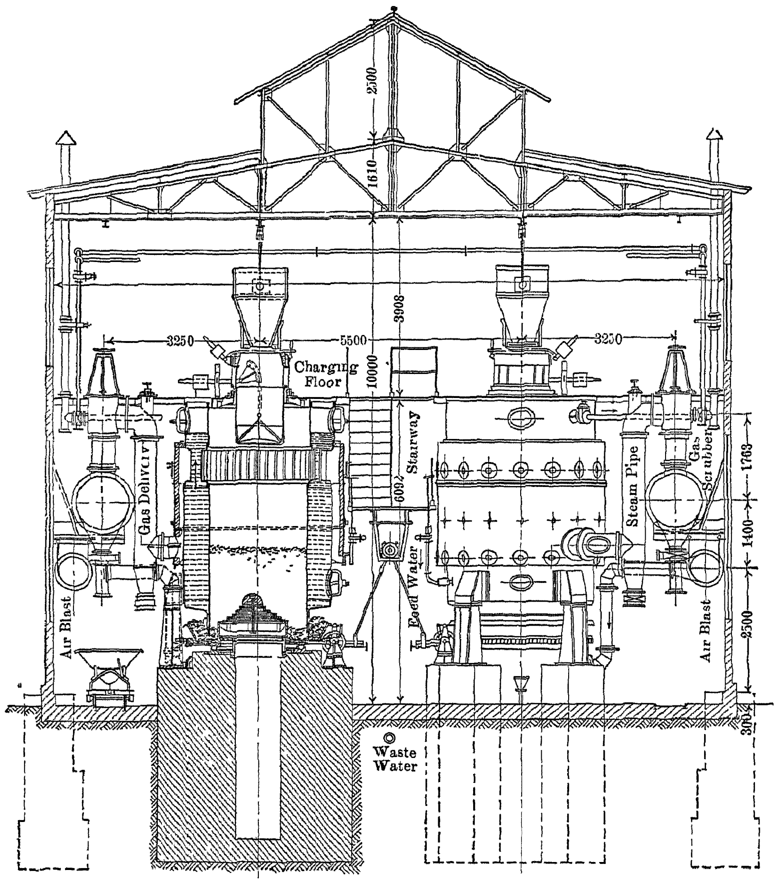



FIG. 35.—CROSS-SECTION SHOWING ESSENTIAL FEATURES OF REVOLVING ECCENTRIC GRATE GAS PRODUCER

appears to serve a relatively greater depth of bed. The Rehmann, in fact, varies the depth of the grates to meet this change of thickness of the clinker. Developments along the lines of slagging producers seem to be only moderate. A few places in France and Germany have overcome, to some degree, the two drawbacks of high maintenance of linings and trouble from freezing, but only after much study.

## APPENDIX*

Since the preparation of this paper, a new type of Hughes mechanical producer has been placed on the market. It embodies a number of changes from the original design while retaining the revolving shell and the oscillating poker for stirring the fuel bed. The dry bottom has been abandoned in favor of a rotary water-sealed pan with ash plows for the continuous removal of the accumulated ashes. Our opinion, without an actual test of the machine, is that this may not be a very distinct improvement since we have found, with other producers of this general type, that the wear and tear on this type of ash remover greatly increases the maintenance expense. The hollow steam-cooled spider is quite similar to the one developed at Depue; it appears to have been designed with great care and should be a very satisfactory change from the old design. The automatic feeder appears to be an improvement, but it probably will not be able to handle the larger sizes of coal, if we can judge by our experience with such devices. However, if the coal is crushed before feeding, the results should be satisfactory.

## DISCUSSION

K. Marsh.—Would accurate information on the temperature of the gas produced be of any assistance in operating the producer?

C. C. Nitchie.—Only in this respect, we attempt to run with a cold top in order to conserve as much hydrocarbon contents as possible. Of course there is considerable tar collection, but we find the gas with the higher hydrocarbons giving the best results. The coal bed is hardly more than a very dull red.

K. Marsh.—Do you have any facilities for taking that temperature?

C. C. Nitchie.—Not as a regular thing.

K. Marsh.—Do you think it would aid in operating a producer if you did?

C. C. Nitchie.—It is hard to say. We have never tried anything of that sort.

H. A. Grine, Langeloth, Pa.—We are using a pyrometer on the outgoing gases. We have the Hegeler type of furnaces where it is desirable to have the gases coming into the producer end of the furnace at a temperature of 1500° or 1600° and get smelting temperature in the first sections of the furnace. It is undesirable to have temperatures above 1600°, because the hydrocarbons are broken down and we get a gas running higher in carbon monoxide.

* Received May 1, 1920.

C. C. NITCHIE.—In that connection, I might say that high temperature is desirable in the case of a producer such as Mr. Grine mentioned, where the gas passes directly from the outlet of the producer to the furnace and where waste heat boilers are relied upon for heat regeneration. We use regenerative furnaces of the Siemens type and high gas temperature is of no advantage; it would raise the temperature of the checkerwork and thus of the stack gases. We figure that the temperature of the incoming gases is practically lost.

W W. KEEFER.—In cooling down the producer beyond a certain point, do you reduce the CO content?

C. C. NITCHIE.—The analyses we give are fairly typical and average around 25 per cent. CO.

W. W. KEEFER.—Is there any difference in the depth of the fuel bed with different sizes of coal? Not long ago I read that if very fine fuel is used in producers, it would be desirable to increase the depth of the fuel bed.

G. S. BROOKS.—As to varying the depth of bed with different coal sizes, we have found, when handling the finer sizes of coal, that it is difficult to control the distribution of blow in the producer with too heavy a bed, so that while we have to run with a rather narrow bed on fine coals, a thicker bed is possible with coarse coal. The difficulty is that the resistance to blow increases very rapidly on fine coals with a heavy bed.

R. M. CHAPMAN, Hammond, Ind.—I understand that automatic devices for regulating gas producer operation depend very largely on analyzing the gas. I am inclined to think that better results can be obtained by controlling a number of the fluctuating elements that enter into the making of producer gas. In one producer, the best gas we could get was 155 B.t.u. With the automatic feed, in a producer, in the same house, attended by the same man and with the same coal conditions, we easily got 175 B.t.u. gas. The only difference was that the coal was fed in continuously. By feeding the coal continuously, we did not have so much fluctuation in temperature and the $CH_4$ in the gas was correspondingly higher. The mixture of steam and air also varies a good deal at times, depending on the depth of the firebed and the thickness of the firebed and on whether or not the firebed is sticky and cold on top. If it is, the steam will not suck in as much air and the proportion gets all wrong. Recently people have been using automatic devices for constantly keeping the ratio of air and steam uniform. Then, there is the automatic control of the pressure of the gas leaving the producer. At times the various furnaces draw very hard on the producers; in many cases it is left entirely to the gas-producer operators to judge whether or

not their pressure is going down. There is also a great fluctuation at the time the ashes are removed. If the ashes could be removed continuously, in many cases a good gas could be had the whole 24 hr. instead of 23 or 23½ hr. Do you think these regulating devices would add to the excellent results you are already getting?

G. S. Brooks.—In attempting to answer some of Mr. Chapman's points, I will say that the automatic or regular feeding of the producer is a point that we have studied since 1912. We have not been able to evolve and have not seen a feeder of the flexibility necessary for our requirements of coarse to fine coal, and everything that we have developed to date has been out of the question because of maintenance costs. A uniformly fed producer unquestionably will give better results and lower fuel ratios in smelting than anything that can be obtained from a producer fed by hand.

The control of the steam blow is another problem that has engaged our attention. The positive or fan blowers with exhaust steam are along the right lines. We tried one in an Eastern plant some years back without much success because of the high maintenance costs on that machine. There is no question but what a well proportioned ratio of steam to air is a very important thing for good producer practice. Getting more gas by increasing the blow is very questionable practice. Some of the automatic regulators are based on this idea.

As to continuous operation at the roast kilns, it would be very much more satisfactory if producers could operate continuously without the cleaning out. We have yet to be shown a producer mechanism that will discharge its ashes continuously and still give reasonable maintenance charges with our hard fine ash of low-grade Illinois coal. Our fine ash from the lower grade Illinois coals is very abrasive and hard material to grind. We would like nothing better than to find a producer that would discharge continuously and still run a number of years without being overhauled.

H. A. Grine.—We have two gas producers using crushed lump coal from 4 in. down to ¾ in. and have no trouble with the automatic feed, and get much more uniform gas than the producers operating with hand dumps. Instead of Koerting steam blowers, we are using the open type fan and controlling the clinkering conditions and temperature conditions by the introduction of steam with the air.

# Testing of Coals for Byproduct Coking and Gas Manufacture

BY HORACE C. PORTER,* PH. D., PHILADELPHIA, PA.

(Chicago Meeting, September, 1919)

MOST of the bituminous and semibituminous coals of this country will coke, and all of them yield, on carbonizing, more or less marketable gas and byproducts. We need, however, a finer distinction as between various grades of coking coals and between coals of different degrees of byproduct and gas-making possibilities. There is required by the consumer or the coal producer a dependable test to show whether a certain coal or mixture of coals will make coke of good commercial value for blast-furnace, foundry, water-gas, or domestic use; whether any difficulties in operation may arise from properties inherent in the coal; and whether gas and byproducts may be expected of good quality and in profitable quantity. For determining coke quality, the most dependable test is an actual run in an oven, but for byproducts a coal may better and more accurately be tested in the laboratory or in a miniature oven or retort.

As to what are the most desirable qualities in blast-furnace coke, the furnace men themselves are not in complete agreement. Some desire a hard, well-burned, "hot" coke, and others get better results from a "greener," less hardened product that burns somewhat more readily in the furnace. But certain qualities of cell structure and porosity are commonly agreed upon as desirable. Information as to what kind of coke a certain coal may be made to yield is valuable in advance, whatever may be the consumer's ideas as to the particular kind most suitable to his needs.

The question arises, Can we predict coking quality with any degree of satisfaction from a small-scale test? It used to be said that coals could not be tested satisfactorily on a laboratory scale either for the coke they would yield or the byproducts. On account of the mystery with which the coking quality of coal has been shrouded and the large determining influences that many believe are exerted by mass, pressure, and the passage of carbonaceous volatile matter through the coking material, it has seemed difficult to reproduce in the laboratory the exact conditions of commercial practice. Great progress, however, has been made in recent years in elucidating the process of carbonization and many of the factors that make for the formation of high-grade coke and the recovery of the best

* Chemical Engineer, The Chemical Service Laboratories, Inc.

quality and yield of byproducts are known. The matter of conditions is capable of careful analysis and it is possible to simulate closely in the laboratory the practical conditions of coke oven or gas retort.

By giving due weight to each of a number of factors derived from the analysis and nature of the coal and its behavior in a carbonizing test, it is possible to form a composite conclusion predicting with approximation the coal's coke-making possibilities. The factors that will be found most helpful are five in number: (1) Origin and geologic history. (2) Proximate composition, particularly the amount of moisture retained by the air-dried coal, amount of volatile matter, and percentage of sulfur (an incidental impurity undesirable in coke or gas, but not affecting coking quality). (3) Calorific (B.t.u.) value of volatile matter, calculated to a "pure" coal, or so-called unit coal, basis. (4) Amount of water of combination and $CO_2$ obtained from coal in a carbonization test. (5) Quality of coke obtained in a so-called "Dutch Basket" test (in an oven) or in a laboratory carbonization test under carefully chosen conditions, having regard particularly to the rate of coking or of the progress of the plastic zone.

Byproduct yields may be determined by a laboratory carbonization test but unless conditions are used that, by careful analysis, prove similar to those of commercial units, the results will require correction by arbitrary factors, a procedure that is not entirely satisfactory.

To analyze, for purposes of comparison and duplication, the conditions in the various commercial carbonization processes, it is well first to tabulate the values of certain common dimensions on an equivalent basis. Since the velocity of gas passage is a basic factor in determining byproducts, we may reduce the other dimensions to a basis of equal gas flow, say 1000 cu. ft. (28.3 cu. m.) per hr., and note their relative values in the different processes. In Fig. 1 are shown the more common carbonizing ovens, or retorts, compared as to relative size and shape, and in Table 1 are given calculated figures showing particularly the heated wall area, the average volume of charge traversed by gas, and the average free space above the charge, all on an equalized basis of gas flow, namely, 1000 cu. ft. of gas per hour.

We see that in these three factors the processes vary considerably and the results of actual practice are relatively such as may be logically predicted from these variations. For example, from the vertical gas retort, tar is obtained in greater quantity, but thinner and of much lower free carbon content, than from a byproduct coke oven or old-type horizontal retort; much less naphthalene, anthracene, and off-take pipe carbon are formed in the verticals and a greater production of ammonia. The horizontals, however, commonly yield gas of higher candlepower. All of these results are largely attributable to the relationship shown by factor *D* of Table 1, the greater contact of volatile products with super-

heating space in horizontal retort and coke oven, the former being nearly three times that of the vertical and the coke oven nearly twice that of the vertical. The coke oven, on the other hand, as shown by $A$ and $C$,

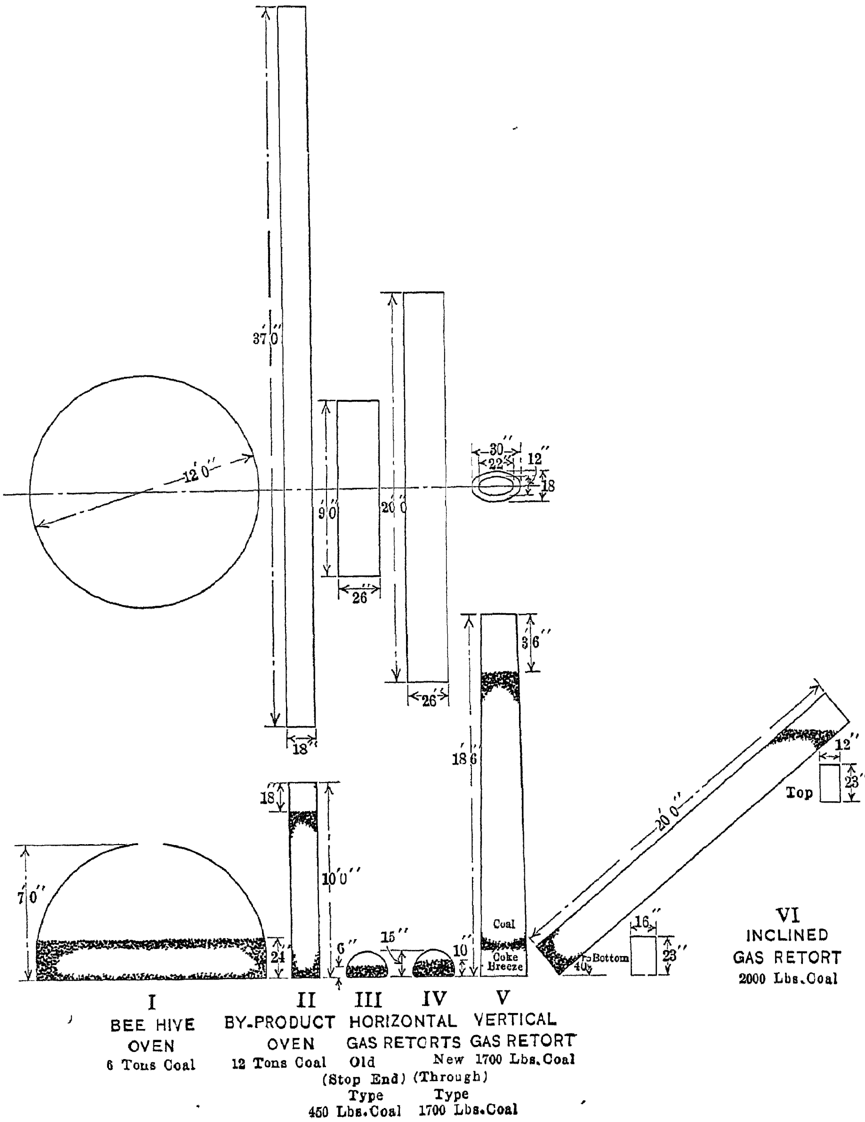



FIG. 1.—RELATIVE SIZE AND SHAPE OF VARIOUS OVENS AND RETORTS FOR CARBONIZING COAL.

has a considerably lower coking velocity than the gas retorts and a greater volume of carbonizing mass per unit flow of gas so that, due principally to these differences, it makes a stronger and better coke

TABLE 1.—*Comparative Dimensions in Various Carbonizing Processes*

| | Beehive Coke Oven | Byproduct Coke Oven | Vertical Gas Retort[a] | Horizontal Gas Retort | |
|---|---|---|---|---|---|
| | | | | Old Type[b] | New Type[c] |
| Coal charge, tons | 6 5 | 12 0 | 0 85 | 0.23 | 0 85 |
| Carbonizing period, hours | 48 | 17 | 11 | 4 | 6 |
| Gas made per hour, thousand cubic feet | ..... | 7 6 | 0 87 | 0 65 | 1 63 |
| A—Coking velocity, inches per hour | 0 50 | 0 53 | 0 68 | 1.10 | 1.0 |
| Velocity of travel of plastic zone | .... | 0 82 | 1.25 | ...... | ..... |
| Heated wall area, square feet | . | 795 0 | 97.1 | 53 1 | 130.0 |
| B—Wall area per thousand cubic feet gas per hour | ..... | 104 6 | 111 6 | 81 7 | 80.0 |
| Volume of charge, cubic feet | 290.0 | 472.0 | 35.5 | 9 4 | 34.5 |
| Depth of charge, feet | 2 2 | 8.5 | 15.5 | 0.5 | 0.82 |
| C—Average cubic feet of charge per thousand cubic feet gas per hour | ...... | 62 05 | 40 8 | 14.5 | 21.2 |
| Open space above charge, cubic feet | ...... | 83.25 | 5.2 | 10.3 | 10.4 |
| Open space, per thousand cubic feet gas per hour | ..... | 10.96 | 5 98 | 15 8 | 6.4 |
| Approximate temperature in free space, degrees centigrade | .... | 950 | 800 | 1150 | 1050 |
| D—Approximate time of gas contact with free space, seconds | .... | 39 | 21 5 | 58 | 23 |

[a] U. G. I. intermittent type, 18½ ft. long, 12 by 22 in. top, 18 by 30 in. bottom, filled within 3 ft. of top.

[b] Old "stop end" type, 15 by 26 in. by 9 ft., charged to 6 in. depth.

[c] Modern "through" type, 15 by 26 in. by 20 ft., charged to 10 in. depth.

for metallurgical furnaces. By the same reasoning, if coke-oven tops are kept cooler or somewhat restricted in volume of space for gas passage above the charge, the ammonia and light oil yields are increased and naphthalene and free carbon reduced.

Careful studies of the carbonization process have been made by Oskar Simmersbach[1] and others as pertaining to coke ovens and by O. B. Evans[2] and others in vertical gas retorts. It is well established thereby that in the coking process a plastic zone forms and progresses through the charge at a rate dependent on the temperature of the wall and the distance of the plastic zone from it. The plastic state is formed at about 350° C. and solidifies at about 440° C. The rate at which this plastic material is transformed to hardened coke probably has considerable influence on the quality of the coke. Furthermore, during the plastic stage, the coal gives up about 10 per cent. of its gas, and this passes chiefly

[1] "Grundlagen der Kokschemie," Berlin, J. Springer, 1914.

[2] *Proc.* Amer. Gas Inst. (1913) 8, 688.

through the uncoked core of the charge, but of the remaining 90 per cent. evolved from the solidified coke, a large proportion passes through the partly coked charge between the plastic zone and the wall, depositing carbon on the cell walls of the coke and undergoing material change itself in the chemical composition of its constituents. Further changes in the gas and byproducts occur in passing through the heated open space above the charge, where such alteration is promoted by the slowing down of gas travel on entering the region of greater cross-section. The large effect of these conditions on coke quality and byproduct yields cannot be doubted, and the necessity for having due regard, in the design of a laboratory test apparatus, for the relative dimensions of the commercial plant as given in Table 1, based on some such equivalent as unit flow of gas, is fully evident.

Owing to the fact, for example, that a small circular cross-section, as of a laboratory tube, on being increased to the much larger section of a commercial retort, enlarges its area as the square of the diameter, the volume of coal through which a unit of gas flows per hour is very much larger in the commercial retort than in such a tube, with equal rates of coking per linear inch. With a higher rate of coking, which is likely to prevail in the laboratory tube, this difference becomes even greater, and it is small wonder that coke quality cannot be judged from coking in a ½-in. (12 7 mm.) tube or in a platinum crucible.

Byproduct yields, as determined in a small tube on a few grams of coal, require correction by arbitrary factors but are of a certain limited value when carefully used by reference to a standard coal of known commercial performance.

Such a test, in a ½-in. glass tube on 20 gm. of coal, is in use in this country in the laboratories of the U. S. Steel Corpn. and of The Koppers Co. It originated in Germany, has been described by J. Schramm in 1913[3] and, with modifications, by the U. S. Steel Corpn. (Chemists Committee) in a pamphlet entitled "Methods for the Commercial Sampling and Analysis of Coal, Coke and Byproducts," 1916. It is claimed to be of value only when calibrated carefully against commercial results on the same coals and when the results are modified by application of suitable factors.

Tests on a larger scale, carbonizing about 1 lb. (0.45 kg.) of coal in an iron retort, are used by the Semet-Solvay Co., Syracuse, N. Y., the Westmoreland Coal Co., Irwin, Pa., the Illinois Steel Co., Gary, Ind., and by the U. S. Bureau of Mines. The results in such tests, particularly on ammonia yields, are found by the Semet-Solvay Co. to be a valuable check on plant operating efficiencies, and the other companies named also use this test as a valuable aid in estimating yields of gas and other

[3] *Journal für Gasbeleuchtung* (1913) **56,** 389.

byproducts from new or unknown coals. The conditions in such a test, compared to those of the smaller test, are more easily adapted to approximating those of commercial processes, so that a better indication may be obtained of coking quality and the quality of the byproducts. Any catalytic action that may be exerted by the iron of the retort in decomposing ammonia or the other byproducts is considered by the above-named users to be too small to play a role of any practical importance. In Fig. 2, reproduced from a paper by Dr. L. C. Jones,[4] chief chemist, of the Semet-Solvay Co., by courtesy of the author, are shown the results of tests by the Semet-Solvay Co., on a good coking coal, at different

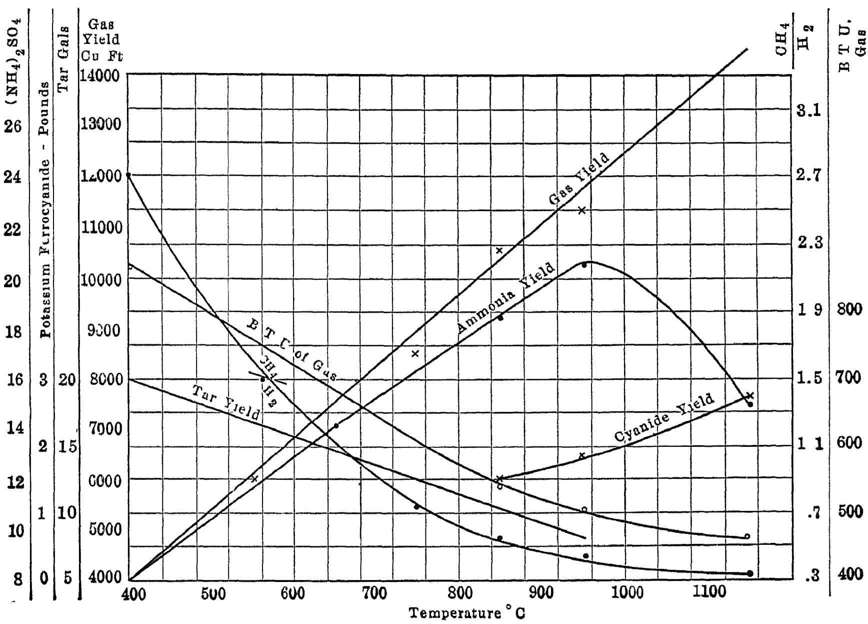



FIG. 2.—INFLUENCE OF TEMPERATURE UPON YIELDS AND CHARACTER OF BYPRODUCTS FROM AN AVERAGE COKING COAL. TEMPERATURES REPRESENT HOTTEST PART OF RETORT AT BEGINNING AND END OF CARBONIZING PERIOD. (SEE FOOT-NOTE NO. 4)

temperatures. The important influence of the maximum temperature of the retort on byproduct yields is readily to be seen.

One of the largest factors influencing byproduct yields is the relative size and temperature of the free space above the coal. This factor is controlled in the laboratory test by the use of a solid plug of suitable size fitting into the open space of the retort and slotted or otherwise shaped so as to permit flow of gas at the required rate.

To judge the quality of coke to be obtained from a coal, the laboratory test alone, for reasons explained in the foregoing, is not adequate. The

[4] Coal and its Byproducts. *Jnl.* Frank. Inst. (1914) **177**, 531.

most satisfactory test for this purpose is a full-scale oven trial. Proper weighing, however, of the five factors enumerated gives a very good indication of the coking possibilities of any coal.

The geologic conditions under which the coal deposit lies are indicative, more or less, of its coking quality. In general, the older coals, the more remote from the peats and lignites, are likely to show the better coking characteristics. Geologic age, under conditions such as have effected deoxidation (reducing the percentage of oxygen in the coal substance), in general seems to parallel coking quality, until the semianthracites are reached.

From the proximate analysis, valuable information is obtained. The amount of water in the coal, after air drying, is a perfectly reliable indication of its degree of deoxidation and of its probable tendency to coke. A percentage larger than 5 ordinarily indicates that the coal substance has not been altered sufficiently by deoxidation to make a high-grade coke. The percentage of sulfur and ash are important as showing objectionable impurities that will remain in the coke. The volatile matter in a good coking coal is ordinarily not less than 15 per cent. nor more than 38 per cent. on the dry basis.

As has been suggested by S. W. Parr[5] and by L. C. Jones[6] the "cokability" of a coal is strongly indicated by the relative calorific value of its volatile matter. Many coals that appear to be rich in volatile matter, as far as quantity is concerned, yield relatively large amounts of $CO_2$ and water in their volatile products, which have no heating value and denote a low order of "cokability." The calorific value of the volatile matter is computed from the calorific value of the coal itself and the percentage of fixed carbon (which has known calorific value) both being on the basis of "pure" coal—free of moisture, sulfur, and corrected ash. Fig. 3 is reproduced from Dr. Jones' paper[7] and shows a comparison of the calorific values of the volatile matter for a number of typical coals. The author says, "By deducting the calorific value due to fixed carbon from the total calorific value and dividing by the percentage of volatile matter, all on the pure coal basis, we get a series of coals of increasing cementing or coking tendencies. * * * This variation in calorific value has been found to represent results in actual coking practice."

The percentage of oxygen in a coal, or the ratio of oxygen to hydrogen, has been proposed as an index of its coking quality, but this factor adds very little to that discussed in the preceding paragraph, since the calorific value of the volatile matter varies approximately in inverse ratio to the percentage of oxygen in the dry coal.

From a laboratory carbonizing test, if made on a sufficiently large

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[5] *Bull.* 20, Illinois State Geological Survey (1915).

[6] *Jnl.* Frank. Inst. (1914) **177, 511.** [7] *Op. cit.*, 529.

scale, we have an opportunity to observe the quality of coke made and obtain an indication of the commercial possibilities. But in addition, this test shows the production $CO_2$, CO, and water of constitution by the carbonization of the coal; and when these products exceed a certain amount, the coal may ordinarily be set down as unsuited for production of good commercial coke, unless, in other ways, it shows very favorable indications. The total heat units contained in the gas per pound of coal, coupled with the yield of tar and light oils in proportion to the percentage of volatile matter, give valuable indications also of the suitability of the coal for coking purposes.

A "Dutch Basket" test, in which a small quantity—say 10 to 50 lb. (4 to 22 kg.) of the coal to be tested is placed in a wire basket, or in

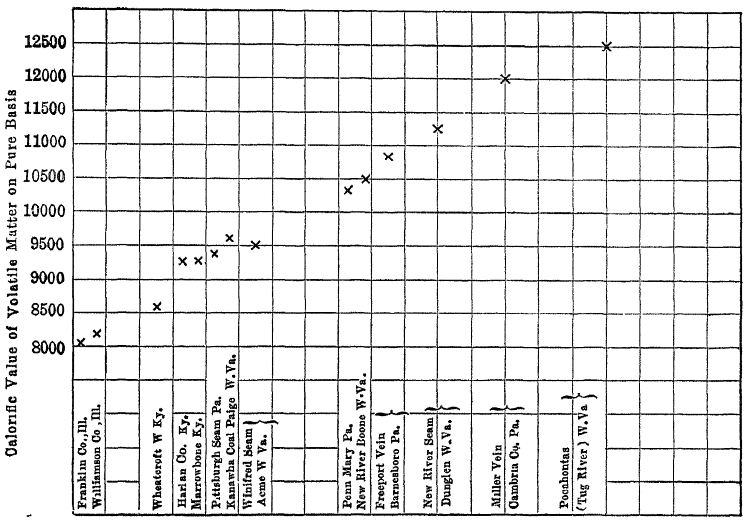


FIG. 3.—CALORIFIC VALUES OF VOLATILE MATTER FOR TYPICAL COALS OF UNITED STATES. CALCULATED ON THE "PURE" COAL BASIS. (SEE FOOT-NOTE NO. 4.)

a wooden box or keg, and lowered into the charge of a coke oven or retort, gives a valuable indication of the quality of coke to be expected, although the results are influenced by the passage of volatile products from the surrounding coal in the main charge.

Summarizing the main points in the foregoing discussion, it is evident from a careful analysis of the conditions and dimensions pertaining to the commercial carbonizing processes, that the rate of coking, the relative volume of charge per unit of gas flow, and the relative amount and temperature of the space above the charge per unit of gas flow vary in large degree and that these variations influence materially both coke quality and byproducts. The nature of the coal, of course, is also a large factor

in the results obtained. It is possible, by a careful control of conditions, to judge both coking quality and byproduct yields by a laboratory carbonizing test, giving weight in this judgment to the proximate analysis of the coal, the nature of its volatile matter, and its geologic history. Such a test should be made comparative by choosing a standard coal having a known commercial performance.

With a view to the control of conditions in accordance with these principles and obtaining thereby concordant test results representing as nearly as practicable those of commercial practice, The Chemical Service Laboratories, Inc., are carrying out a series of careful experiments in an apparatus carbonizing about 4 or 5 lb. (1.8 or 2.3 kg.) of coal. It has been impracticable, on account of the short time available for preparation, to include in this preliminary publication a report of experimental results thus obtained. Such a report, however, will, it is hoped, be ready for inclusion in the final issue of the paper.

## DISCUSSION

H. C. Porter.—The percentage of fixed carbon is determined by the proximate analysis and, having the B.t.u. value of the coal and the known B.t.u. value of the fixed carbon (assuming it to be the same as pure carbon), the calorific value of the volatile matter can be easily calculated. This has been brought out by Professor Parr of the University of Illinois.

F. W. Sperr,* Jr., Pittsburgh, Pa.—A correct understanding of the peculiar factors involved in coal distillation and a knowledge of practical results obtained under different operating conditions are essential to prevent misleading interpretations of laboratory results. I have noted a few misconceptions relating to this subject that are very popular. The first misconception, which is very common, is that the chemist only has to analyze a sample of coal to find out how much ammonia, tar, benzene, naphthalene, etc. it will produce when treated in a byproduct coke oven or a gas retort. This is a fallacy that probably all of you recognize, but it is so common that it ought to be mentioned. Of course coal does not contain ammonia or tar or any of the other byproducts for which it is famous, it only contains the elements that under suitable conditions will unite to form these different substances. The amounts of the byproducts produced are dependent on conditions. They are more dependent on the conditions of the treatment than they are even on the nature of the coal. Ordinary methods of chemical analysis, of course, are not applicable to tests of coal.

The second misconception is that, in order to determine the yields

* Chief Chemist, The Koppers Co.

of byproducts from coal, we may not resort to analysis but must build a small experimental plant with dimensions proportional to the byproduct coke oven or gas retort and regulate the conditions of heating to correspond with the conditions on a large scale; and that then we will get the same proportional amounts of ammonia and other byproducts. This is far from the truth; a study of Dr. Porter's paper shows why this is so. He rightly brings out the dimension factor, which is very important in determining the yields of byproducts. You cannot reckon in small-scale testing without that dimension factor, without finding your small-scale results very greatly at variance with large-scale results.

A third misconception has not been sufficiently emphasized. This is that although it may be true that a small-scale apparatus gives different results from those obtained in plant practice, these results are always in proportion from one coal to another and may be reduced to a proper basis by a system of factors. Take a simple case: Suppose that with a certain laboratory test, you get 6 gal. of tar from a coal which, in plant practice, yields 9 gal. There you have a factor of 150 per cent. Some experimenters apply that 150 per cent. factor to all coal no matter whether it comes from the Pittsburgh seam in Pennsylvania, or from the Walla-Walla seam in Madagascar. Coals of different origin will not necessarily give yields in proportion to those obtained from the same coals in plant practice. For example, take Pittsburgh coal yielding, say, 25 lb. of ammonium sulfate in a plant, which is a fair average figure. Let us say it will yield 28 lb. in the laboratory when heated in a certain way. There is then a ratio of 25 to 28. Some Indiana and Illinois coals heated in the same apparatus in the same way will give 35 lb. of ammonium sulfate, but in actual plant operation these same coals produce only about 28 lb. Here the ratio is 28 to 35.

The fourth conception is a little more intangible and possibly is of interest only to those concerned with the operation of byproduct coke or gas plants; it is that plant efficiency can be calculated from small-scale laboratory tests. To put the matter in as few words as possible; it is wrong in principle to work out a method for determining plant yields based on actual plant results and then turn around and use that same method to criticize, from the standpoint of efficiency, the very results upon which it was based.

R. D. Hall, New York, N. Y.—It seems to me that some of us might question the statement that, in general, the older coals, that is those that are the more remote from the peats and lignites, are likely to show the better coking characteristics. I think the true bituminous coal will reach pretty nearly the climax from a coking point of view. Beyond that, as the volatile matter declines, the coking quality begins also to decline.

F. W. Sperr, Jr.—It does not seem to me that the coking qualities decline until we get past the semi-bituminous coals. We get wonderfully excellent coke from semi-bituminous coals properly treated.

R. D. Hall.—Would you call Pocahontas coal as good a coking coal as Connellsville?

F. W. Sperr, Jr.—From the standpoint of the beehive oven, it may not be as good as the Connellsville; but Pocahontas coal is a better coking coal than many of the true bituminous coals high in volatile matter. In beehive practice, Connellsville coal represents the apex of coking quality; but in byproduct practice there is no sharply defined apex. We get excellent cokes from coals ranging from 20 per cent. up to 38 per cent. in volatile matter; many of these are in the semi-bituminous class; and if you make true bituminous coals a dividing line, it would seem unfair to coals somewhat lower in volatile matter to say that they exhibit a decline in coking quality.

H. C. Porter.—It is commonly preferred, in the coking industry, to add to a high-volatile coal a certain percentage of coal of the type of Pocahontas so as to improve the quality of the coke. Pocahontas coal coked alone gives a very strong well-developed coke, although in certain respects it may not always be regarded as the very best coke for metallurgical uses. The coking quality accordingly is very well developed in the Pocahontas type of coal.

# Occurrence and Origin of Finely Disseminated Sulfur Compounds in Coal*

BY REINHARDT THIESSEN,† PH D, PITTSBURGH, PA

(Chicago Meeting, September, 1919)

UNDER sulfur in coal, is usually understood that form of sulfur which is combined with iron and known as pyrite. It occurs in the form of balls, lenses, nodules, continuous layers, thin sheets, or flakes, both in horizontal planes and vertical cleavage fissures. But pyrites also occur as very fine microscopic particles, or nodules, disseminated through the compact coal This form has had but very little consideration. Finally, there is sulfur in coal in an amicroscopic form (not visible with an ordinary microscope), probably combined with the organic matter that exists in the coal. This form has had considerable attention from a scientific standpoint, but has probably not been recognized enough on the economic side. One or the other, or both, of these two latter forms may comprise the larger part of the sulfur content of coal, especially after it has been washed or otherwise prepared for use.

## PYRITES IN MICROSCOPIC PARTICLES

All coals that have been examined by the author contained a varying amount of sulfur in very small globules, or particles, of pyrite. These particles are seen in a thin section as roughly rounded opaque dots, Figs. 1 to 10. When isolated, they are generally shown to be approximately spherical in shape, with a rough outer surface. They vary in diameter from a few microns to a hundred microns, the majority measuring from 25 to 40 microns; relatively few exceed the latter diameter. They are, therefore, very small objects. Their size is best appreciated by comparing the illustrations, in which they are shown at a magnification of 100 diameters, with some known area like the period used in the ordinary printed page. The period is about 0.6 mm. in diameter, and when magnified 100 times, will cover a circular area of about 2½ in. in diameter, or all but the corners of the illustrations. The illustrations, therefore, represent an area about as large as an ordinary period, and the pyrite globules form but a small part of the total area. Very frequently a number of these particles are joined together in horizontal rows; oc-

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† Research Chemist, U. S Bureau of Mines.

casionally, a number are grouped together into small lenticular masses or into irregular groupings. Occasionally a number have coalesced into one mass.

During the process of preparing the coal sections, many of the globules break into innumerable small cubical fragments, which usually form a streak on either side of the globule. These are so small that it requires a

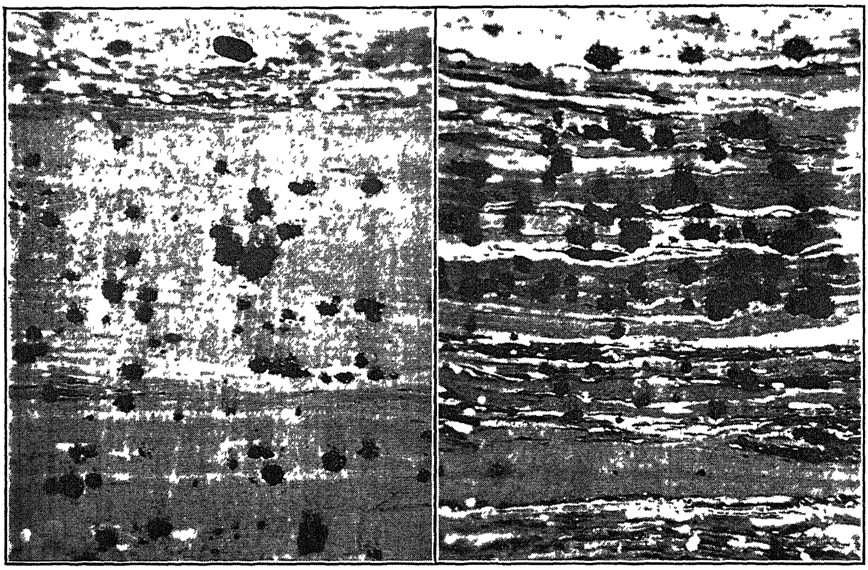

FIG 1. FIG. 2

FIG. 1.—THIN CROSS-SECTION OF COAL FROM VANDALIA MINE NO 82, NEAR TERRE HAUTE, IND., NO. 5 BED, SHOWING A THIN LAYER OF ANTHRAXYLOUS COAL WITH NUMEROUS MICROSCOPIC PYRITE GLOBULES. PYRITE GLOBULES ARE SHOWN BLACK AS IRREGULAR ROUGHLY ROUNDED AREAS. MANY HAVE BEEN PARTLY BROKEN AND FRAGMENTS, CONSISTING OF MINUTE CUBES, HAVE BEEN DRAGGED TO SOME DISTANCE OVER THE SECTION. ANTHRAXYLON IS THAT PART OF COAL DERIVED FROM PARTS OF LOGS, STEMS, BRANCHES, OR ROOTS × 100.

FIG. 2.—THIN CROSS-SECTION OF COAL FROM VANDALIA MINE NO. 82, NEAR TERRE HAUTE, IND., FROM NO. 5 BED, SHOWING LAYER OF DULL COAL CONTAINING NUMEROUS THIN STRIPS OF ANTHRAXYLON EMBEDDED IN AN ATTRITUS OR DEBRIS. BLACK, ROUGHLY ROUND AREAS REPRESENT MICROSCOPIC PYRITE GRAINS; WHITE IRREGULAR STRIPS REPRESENT CUTICLES; AND SHORT LINEAR PATCHES REPRESENT SPORES. TENDENCY OF PYRITE GLOBULES IS TO FORM ROWS ALONG THIN STRIPS OF ANTHRAXYLON. ATTRITUS IS THAT PART OF COAL DERIVED FROM ALL SORTS OF MACERATED PLANT PARTS AND PLANT PRODUCTS. × 100.

very high magnification to distinguish one from the other. This is shown in almost all the illustrations, but particularly in Figs. 1 and 2.

The amount of pyrite in this form varies considerably in different beds from which coals have been examined and also in different samples from the same bed, or even in different parts of the same section. A section without these pyrite particles is rarely obtained; so far, no regularity has been discovered. Some of the coals examined contained, on the

whole, relatively large amounts, others but very small amounts. There is more finely disseminated pyrite in the coal from the Vandalia mine No. 82, of the No. 5 bed, in Indiana, than in any of the other coals studied. Figs. 1 and 2 show common appearances. All sections of this coal contain scores of pyrite particles, and in some laminæ they are so numerous that it is impossible to cut sections. This finding is in harmony with the analysis of the coal from this bed, which shows from 2.46 to 4.21 per cent. of sulfur.

FIG 3

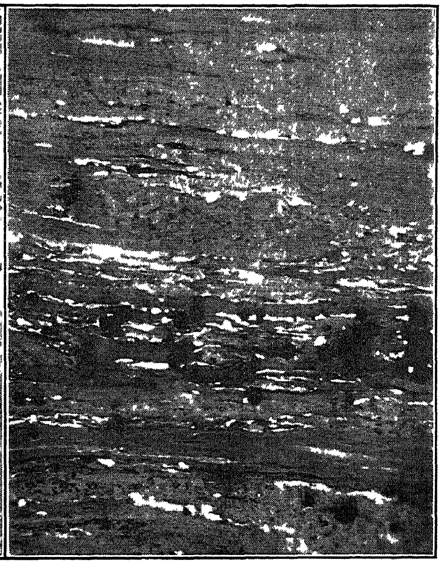

FIG. 4.

FIG. 3.—THIN CROSS-SECTION OF COAL FROM LA SALLE, ILL., SHOWING A LAYER OF ANTHRAXYLOUS COAL, INCLUDING A NUMBER OF PYRITE GLOBULES. × 100.

FIG. 4.—THIN CROSS-SECTION OF COAL FROM SESSER, ILL., NO. 6 BED, SHOWING PYRITE GLOBULES IN DULL COAL, WHICH HERE IS COMPOSED OF THIN STRIPS OF ANTHRAXYLON AND ATTRITUS; THE LATTER INCLUDES SPORES, SHOWN WHITE. × 100.

The Illinois coals from beds No. 6 and No. 2, containing from 0.50 to 6.79 per cent. sulfur, by analysis, as far as they have been examined, do not contain nearly as much pyrite in this form. Nevertheless, there is hardly a section made in which none is observed. As in the Vandalia coal, it is here very irregularly distributed. Similarly, in some sections, the globules are so numerous that it is difficult to cut a satisfactory section from it. Normal appearances of some of the Illinois coals are shown in Figs. 3 to 5. Different coals from the different beds of Illinois have not been compared in this respect.

The coal from the Pittsburgh seam, analyzing from 0.78 to 2.83 per cent. sulfur, apparently contains less pyrite in this form than the Illinois

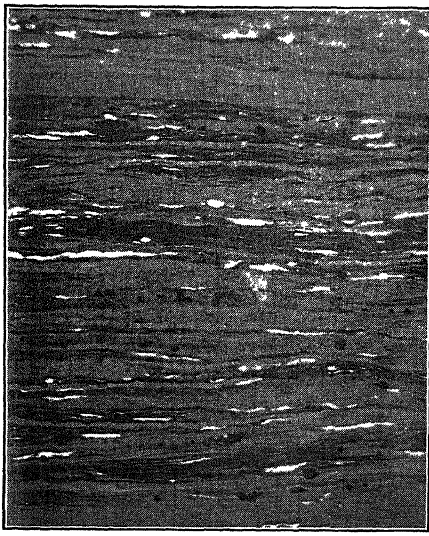

FIG. 5.

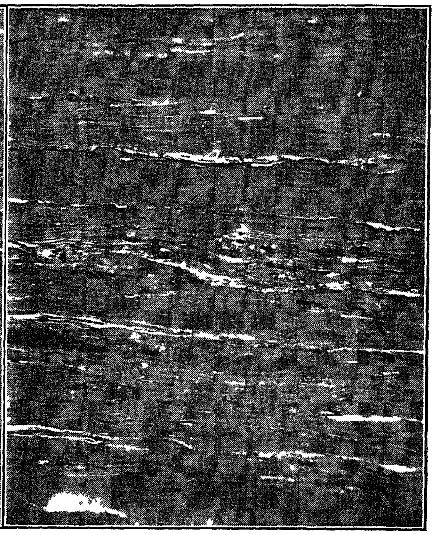

FIG. 6.

FIG. 5.—THIN CROSS-SECTION OF COAL FROM SHELBYVILLE, SHELBY CO., ILL. PYRITE GLOBULES ARE DISTRIBUTED THROUGH WHOLE SECTION; HERE AND THERE, SEVERAL HAVE JOINED AND OTHERS HAVE COALESCED. GLOBULES ARE SOMEWHAT SMALLER THAN IN OTHER SECTIONS SHOWN. × 100.

FIG. 6.—THIN CROSS-SECTION OF COAL FROM SIPSEY MINE, OF BLACK CREEK BED, ALA. SOME OF PYRITE GLOBULES HAVE COALESCED INTO LENTICULAR MASSES, SMALLER GLOBULES ARE DISTRIBUTED THROUGH WHOLE SECTION. × 100.

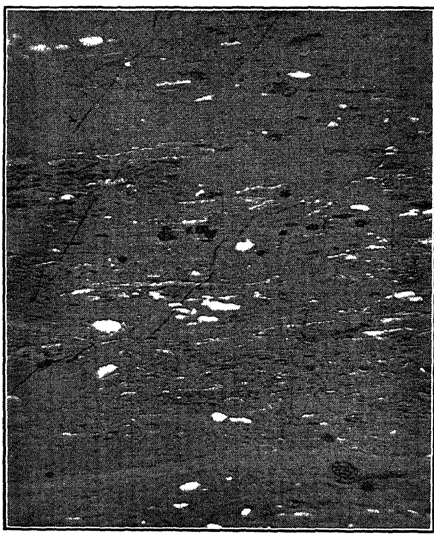

FIG. 7.

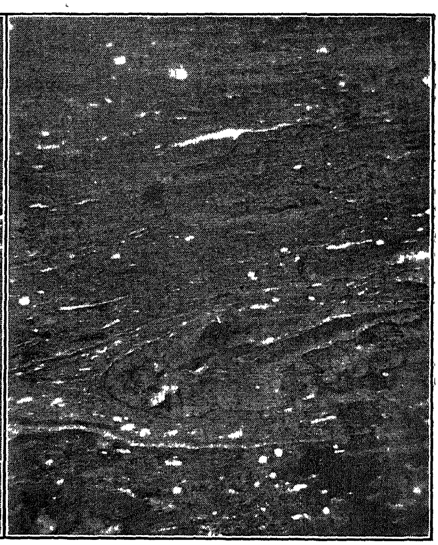

FIG. 8.

FIG. 7.—THIN CROSS-SECTION OF SUB-BITUMINOUS COAL FROM STONE CANYON, CONTRA COSTA CO., CALIF. COAL SHOWN CONSISTS OF RATHER FINELY MACERATED WOODY MATTER, INCLUDING RESINOUS PARTICLES AND CUTICLES, BESIDES PYRITE GLOBULES, SHOWN IN BLACK. × 100.

FIG. 8.—THIN CROSS-SECTION OF LIGNITE FROM MONTANA. SECTION SHOWN CONSISTS OF MACERATED WOODY MATTER AND OTHER PLANT DEBRIS, INCLUDING SOME SPORE AND CUTICULAR MATTER. ONLY TWO PYRITE GLOBULES ARE SHOWN. × 100.

coals. It is of general occurrence, however, and, on the whole, an appreciable amount of the sulfur is present in this form of pyrite. A sample of coal examined from the Sipsey mine in the Black Creek Bed, Ala., analyzing from 0.83 to 1.27 per cent. of sulfur, shows similar contents, Fig. 6.

All the sub-bituminous coals and lignites examined from ground sections revealed globules of similar form and appearance. Fig. 7, of a sub-bituminous coal from Stone Canyon, Calif. (4.48 to 4.95 per cent. sulfur

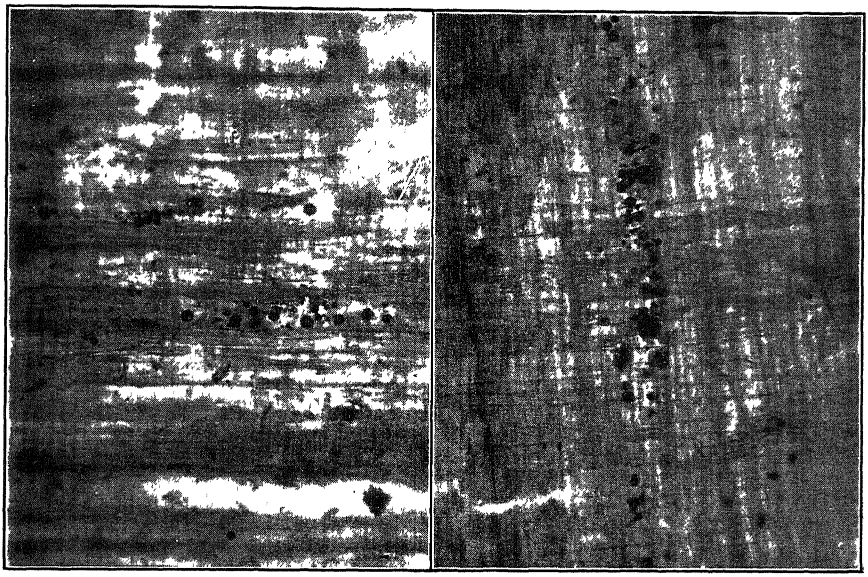

FIG. 9. FIG. 10.

FIG. 9.—THIN SECTION OF WOODY PEAT, TAKEN FROM PEAT BOG NEAR HAYTON, WIS. BLACK DOTS REPRESENT PYRITE GLOBULES LODGED IN WOOD FIBERS; SEVERAL HAVE BEEN BROKEN INTO MINUTE CUBICAL FRAGMENTS. × 100.

FIG. 10.—THIN SECTION OF WOODY PEAT TAKEN FROM PEAT BOG NEAR HAYTON, WIS., SHOWING CONSIDERABLE NUMBER OF PYRITE GLOBULES LODGED IN WOOD FIBERS. IT WILL BE NOTICED THAT THEY ARE STRUNG OUT IN CAVITIES OF WOOD FIBERS. × 100.

by analysis), represents average conditions of the sections examined. Some of the samples of lignites were found to contain relatively large amounts of it. Fig. 8 shows less than the average conditions.

The largest number by far of the pyrite globules are found in the anthraxylon of the coal, that is, that part of the coal derived from the woody parts of plants, although a considerable number are distributed through the attritus, or debris. The pyrite globules are readily distinguishable from the resinous inclusions of the anthraxylon, in that the pyrite globules have a yellowish glistening appearance and are opaque, while the resinous inclusions are dull and are at the most slightly translucent. They are also distinguished from resinous globules by

their outline, which is usually rough, or ragged, while the outline of the latter is usually smooth.

## Organic Sulfur

A certain amount of sulfur has been found to be present in coal in an amicroscopic form. Although, in certain samples, no sulfur can be detected ordinarily by the microscope, microchemical and qualitative chemical tests reveal sulfur. When such samples are burned, in some cases, their ashes show sulfur by microchemical tests Also, recent observations and analyses of a large number of samples from different coals have shown that, in a number of cases, more sulfur is found than can be accounted for if the sulfur were combined only with the minerals found in coal. This form of sulfur is probably that recognized as organic sulfur. Little or nothing is known of organic sulfur and it is not even positively known that sulfur exists in this form. But there are a number of observations on record that lead one to assume that it is present as such.

Prof. T. G. Wormley,[1] of the Ohio State Geological Survey, was apparently the first to call attention to the fact that many coals that contain but little iron have a large percentage of sulfur, a larger amount of sulfur than could be accounted for if the sulfur were combined only with the iron found in the coal. His experiments go to prove that a large part of the sulfur found in coals exists as some organic compound, the exact nature of which he was not able to determine. A few years later a number of analyses made by Andrew S. M'Creath[2] for iron and sulfur show that the sulfur in most cases is largely in excess of the amount required to convert the iron into iron pyrite. In only two instances did all the sulfur seem to exist as bisulfide of iron. Kimball[3] a few years later reviewed the whole field of sulfur in coal, and concludes that some of the sulfur may be combined with the organic matter of coal, the same as it is supposed to be combined with rubber in vulcanized rubber. Drown,[4] in the effort to develop a better method for the determination of sulfur in coal than was in vogue at that time, incidentally observed, among other interesting results of his analyses, figures that led him to believe that sulfur must exist in coal as organic sulfur.

Little work has been done since on the organic sulfur in coal until recently, when the work was resumed in the coal laboratory of the Pittsburgh Station of the Bureau of Mines, as already referred to. At about the same time, the study was resumed in the Engineering Experiment

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[1] Some Theoretical and Practical Conclusions on Coal. Geol. Survey of Ohio (1873) **1**, 360–364.

[2] Second Geol Survey, Pennsylvania, Report of Progress in the Laboratory of the Survey. **M.** (1874–75) 30–32, and Report **MM.** (1876–78) 123–126.

[3] *Trans.* (1880) **8**, 181–204.

[4] *Trans.* (1881) **9**, 656–663

Station, University of Illinois, by Parr and Powell,[5] with results that led to the conclusion that sulfur exists as organic sulfur in coal. It is, therefore, highly probable that sulfur, like nitrogen, is still present in organic compounds similar to those existing in living plants.

## Origin of Sulfur in Coal

The origin of the pyrite in coal is a matter of much speculation. The consensus of opinion appears to be that it must have been deposited by circulating waters or by seepage through the superincumbent rocks. Both sulfur and iron are found in underground waters. If in the form of sulfates originally, the organic matter of the coal is supposed to have reduced them to sulfides. The possibility that the sulfur in coal may have had its origin through the plants that contributed to the coal appears to have received but little consideration, although some admit of the possibility.[6] That this is not improbable will be shown in the following consideration of sulfur in plants.

## Sulfur in Plants

Sulfur is an essential element in almost all proteins and proteins are essential to all living organisms—plants as well as animals. Being an essential element of proteins at once signifies its wide distribution and universal presence wherever there is life. Wherever there are, or were, peat-forming deposits there are living organisms, hence, wherever there are coal beds, whether lignites, sub-bituminous or bituminous coal beds, there was life and, therefore, proteins. The percentage of sulfur present in proteins may be small, but the universal presence of plants in large numbers and other living organisms, hence of proteins, must make the total amount of sulfur thus combined tremendously large. The proteins are, therefore, of great importance in the history and the chemistry of the sulfur found in coal

Proteins are of such complex structure that their chemical investigation is a matter of extreme difficulty. Investigations are rendered even more difficult by the fact that, with a few exceptions, they do not crystallize, and cannot be distilled without decomposition. A number of groups of nitrogenous compounds are classed as proteins, making the number of proteins quite large; a few of the most important and best known are albumin, globulin, edestin, gliadin, and legumin. From a physiological

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[5] A. B Powell and S W. Parr: A Study of the Forms in Which Sulfur Occurs in Coal. *Bull.* 111. Engng. Ex. Sta., Univ. of Illinois.

[6] T. G. Wormley. *Op. cit.*

Editorial: Original Sulfur in Coal. *Coal Age* (1913) **4**, 273–274, The Element Sulfur. *Coal Age* (1913) **4**, 586.

standpoint, the proteins form well-defined groups but from a chemical standpoint it is a difficult matter to define them exactly and to embrace each in well-defined limits. Although they may exhibit great differences in physical and chemical behavior, they do not differ much from one another in their chemical composition and the elements carbon, hydrogen, oxygen, nitrogen, and sulfur, always present, vary only within small limits in the different ones, as shown in the following table: carbon, 50.0 to 55.0 per cent.; hydrogen, 6.5 to 7 3 per cent ; oxygen, 19.0 to 24.0 per cent.; nitrogen, 15.0 to 17.6 per cent.; sulfur, 0.3 to 5.0 per cent. Some also contain from a trace to 0.5 per cent. of phosphorus

Different kinds of proteins from different sources give quite constant results within small limits. Albumin, for example, from a number of seeds and nuts was found to be similar in all cases and its composition was the same carbon, 51.25; hydrogen, 6.88; oxygen, 22 25, nitrogen, 18.69, and sulfur, 0.93. Other examples[7] are:

| | Legumin of Pea, Per Cent | Edestine of Pea, Per Cent. | Vignin of Cow Pea, Per Cent | Globulin of Cow Pea, Per Cent |
|---|---|---|---|---|
| Carbon. | 52 20 | 53 3 | 52 64 | 53 25 |
| Hydrogen | 7 03 | 6 99 | 6 95 | 7 07 |
| Nitrogen | 17 90 | 16 30 | 17 25 | 16 36 |
| Sulfur | 0 39 | 1 06 | 0 50 | 1 11 |
| Oxygen | 22 48 | 22 34 | 22 66 | 22 21 |

Although relatively little is known of the structure of the proteins as a whole, after many researches and investigations the conclusion has been reached that they are built up of a group of amino acids. In other words, the amino acids form the foundation of protein just as bricks form the walls of buildings. Cystein is the one responsible for the sulfur in proteins; its formula is $C_6H_{12}N_2S_2O_4$. By hydrolytic cleavage with mineral acids the sulfur of the protein substance is regularly split off as cystein. The sulfur in this group is regarded as a derivative of hydrogen sulfide, since hydrogen sulfide and some methylmercaptan, $CH_3SH$, are split off in the putrefaction of proteins.[8]

*Non-protein Forms of Sulfur in Plants.*—While most of the sulfur in plants occurs in the proteins, a number of plant families contain other sulfur. In the mustard family, the sulfur is found both as nitrogenous substances, the oils of mustard, combined with glucose and other compounds to form glucosides, and as sulfides; while in the onion family, sulfur occurs only as sulfide.

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[7] Thomas B. Osborn and George F. Campbell. Legumin and Other Proteids of the Pea and Vetch. *Jnl.* Am. Chem Soc (1896) **18,** 583–609, (1897) **19,** 494–500; (1897) **19,** 509–513, (1898) **20,** 348–375.

[8] Olof Hammarsten. "Text-book of Physiological Chemistry." Trans by John A. Mandel. Wiley, 1911.

*Sulfur in Ashes.*—Nine elements are regularly constituents of the ashes of plants—sulfur, chlorine, phosphorus, silica, potassium, sodium, calcium, magnesium, and iron. Here and there a few other elements, like manganese and aluminum, are also present. The largest amounts of sulfur are found in the ashes of plants or plant parts and organs rich in proteins, and in the ashes of the cruciferæ and the onions, rich in oils of garlic, oils of mustard and their glucosides. In the ashes of these plants, as high as 4 to 8 per cent. of sulfur is found, while the ashes of leaves contain usually from 1 to 2 per cent. of sulfur.

The wood of trees is generally low in ash content but sulfur is a constant component in varying amounts. In some wood ashes it is found in considerable amounts; as in Prunus mahaleb or Maheleb cherry, 2.8 per cent. sulfur; Morus alba or white mulberry, 3.9 per cent. sulfur; and Pinus strobus or white pine, 3.7 per cent. sulfur. It should be noted in this connection that the percentage of sulfur in ashes is not a true index of the original content of sulfur in the plant from which they came, since a considerable part of the sulfur escapes during burning.

In a study of the sulfur in plants, it is necessary to make the deductions from the living plants because it is only these whose chemistry can be studied. There is, however, every reason to believe that the chemistry of the plants of any geological period, whether of the Paleozoic, the Tertiary, the Cretaceous, or the Recent is the same. Paleobotany teaches that the plants of all ages, as far as can be learned from their anatomy, morphology, and structure had the same kinds of organs, produced the same kinds of products, and performed the same functions as those of today. The main difference in the plants of the different periods was that different phyla were represented more predominantly during the different times. During the Paleozoic times, the Lepidodendrons, the Calamites, and the Cycadophytes, plants to which the Lycopodiums or club-mosses, the Equisetums or horsetails, and the Cycads, respectively, of today belong, formed the bulk of the coal-forming plants.

During Tertiary times, the Conifers or pines appear to have been the predominant coal-forming plants. Today both the conifers and the dicotyledonous plants, that is, the pines and ordinary leafy plants, are the peat-forming plants, while the horsetails and the club-mosses form but an insignificant part. A chemical study of all the different plants of the different phyla living today has shown that the chemistry of their products is similar. Cellulose, ligno-cellulose, protein, chlorophyl, etc. have a similar composition respectively, no matter in what plant they are found. There is, then, every reason to believe that the sulfur contents of the plants of any period were similar to those of the plants of today. It is also possible that certain plants of ancient times were particularly rich in non-protein sulfur, like the onion and the mustard families of today.

*Source of Sulfur in Plants.*—The sulfur in the higher plants must be taken in through the roots, or organs functioning as roots, as sulfates only. The sulfates may be in the form of calcium, magnesium, and potassium sulfate. They must be reduced by the plant, but how and where this is done is not known. In the plant, sulfur is found chiefly as hydrogen sulfide derivatives, as in the cystein components of proteins and in sulfides, or sulfides combined with other substances in the form of glucosides. The plants must, therefore, carry out a sulfate reduction. The assimilation of hydrogen sulfide must proceed rapidly since it is a violent poison to protoplasm.

*Oxidation of* $H_2S$.—Proteins yield hydrogen sulfide on putrefaction; likewise plants containing sulfides or sulfides in glucosides yield hydrogen sulfide and some mercaptan. Proteins and glucosides are relatively easily decomposed, yet a considerable amount of sulfur may have been retained in the original tissue as organic sulfur, probably still as a sulfide. This would be indicated by the fact that a large amount of vegetable matter escapes putrefaction, as in peat formation and in the large amount of well-preserved plant tissue found in coals and lignites. But neither $H_2S$ nor the sulfur of the plant that has escaped putrefaction is available to plants; in fact $H_2S$ is a deadly poison to them. It is, therefore, of considerable importance to know how sulfur in this compound can again be made available for plants, or in fact, how any form of sulfur other than a sulfate may be made available. This is accomplished through the intervention of so-called sulfur bacteria, of which there are several quite well-known groups.

*Sulfur Bacteria.*—While hydrogen sulfide is a violent poison to higher plants, certain lower forms, both bacteria and fungi, may utilize it as a source of energy. Well-known forms are Beggeatoa, Thiophysa, Thiospirillum, and Thiothrix. Beggeatoa is the best known, due to the studies of Winogradsky, who finds that this organism occurs in all swamps in which the water and soil contain enough sulfates or hydrogen sulfide. The sulfate, however, must be reduced by means of putrefaction to $H_2S$ before it is available; this is accomplished by any plant during its life and decay. It also lives abundantly in sulfur springs. $H_2S$ is first oxidized to elementary sulfur, which is deposited in the protoplasm as amorphous sulfur, seen as highly refractive inclusions in the protoplasm. This sulfur is gradually oxidized in the organism to $H_2SO_4$. There take place, therefore, two reactions within the body of Beggeatoa, which may be expressed thus:

$$2H_2S + O_2 \rightarrow 2H_2O + S_2$$
$$S_2 + 3O_2 + 2H_2O \rightarrow 2H_2SO_4.$$

The sulfuric acid set free combines at once with carbonates to form sulfates. These may be calcium sulfate, sodium sulfate, potassium

sulfate, or magnesium sulfate, and perhaps iron sulfate, all of them available to the plant for another cycle. By this means, Beggeatoa derives its energy. In order that it may thrive, a constant supply of $H_2S$ is necessary, which substance is usually furnished by decomposing vegetable matter. When the supply is plentiful, grains of sulfur are deposited in the protoplasm of the bacteria; when the supply is scarce, they are dissolved and oxidized to $H_2SO_4$. A proper supply of oxygen is necessary for both processes. Thiotrix is a sessile form, whose habits and characters are similar to Beggeatoa.

Other forms are the so-called red sulfur bacteria. These possess a red pigment, called bacteriapurpurin, which possibly has a function similar to that of chlorophyl in green plants. These bacteria can live in water rich in $H_2S$, and, in fact, are not killed in concentrated solutions of it. They are anaerobic. According to Winogradsky, they always live in company with other organisms containing chlorophyll, and hence are able to decompose $CO_2$. The oxygen thus set free is available for the oxidation of $H_2S$ to sulfuric acid.

## Sulfur in Peat

The present state of our knowledge may be summarized as follows: Plants contain sulfur largely in the form of proteins and during putrefaction sulfur is set free as hydrogen sulfide, a form that cannot again be directly utilized by plants, sulfur bacteria, on the other hand, oxidizes it to sulfuric acid, which immediately attacks the carbonates and changes them to sulfates. It is possible that other chemical reactions are going on without the intervention of bacteria. Hydrogen sulfide, in the presence of water in which any of the heavy metals are dissolved, will be precipitated as a sulfide, which may readily be oxidized to a sulfate. Ferric sulfate may be directly reduced by hydrogen sulfide to ferrous sulfate with a simultaneous precipitation of sulfur, and a further reaction of hydrogen sulfide, sulfur, and ferrous sulfate slowly gives rise to ferric disulfide or pyrite. This reaction may be represented by the equation $FeSO_4 + H_2S + S = FeS_2 + H_2SO_4$.[9] But whether such reactions could take place under such strongly reducing conditions as exist in the peat bogs is doubtful.

Sulfates are generally detected in peat, especially in the upper strata, by microchemical tests; in many instances they are present in relatively large amounts. Calcium sulfate appears to be the chief one in peat bogs investigated. When peat is permitted to dry very slowly, calcium sulfate crystallizes out, in certain cases in relatively large quantities.

[9] E. T. Allen: Sulfides of Iron and Their Genesis *Min. & Sci. Pr.* (1913) **103**, 413–414.

Sulfur is present in these bogs, irrespective of the horizon from which taken, whether from near the surface, halfway down, or at the bottom of a bog 10 to 12 ft. deep. Since the plants growing in a bog, in very dense and luxuriant masses, subsist on the residue of the plant growth that preceded them, it is evident that the plants now living in the bog are not able to take in nearly all the sulfates contained in the deposit. There is, therefore, by far more total sulfur in the bog, in both the dead-plant and the living-plant matter than is needed for the cycle. There is thus an accumulation of sulfur. The peats upon which most of these investigations are based contain from 0.89 to 1.63 per cent. of sulfur. Chemical analysis of peats from many sources show all the way from 0.29 to 4.21 per cent. It is, therefore, not far different from the nitrogen content. Iron is also invariably present in peat.

In previous studies on peats, lignites, sub-bituminous, and bituminous coals, it has been shown that there is a continuous loss of cellulosic substances and a relative concentration of resin, resin waxes, and waxes. It is a difficult matter to make fair estimates as to how much of the original peat deposit has disappeared during the transformation from peat to coal. But observations made on the anthraxylon components, that is, components derived from wood in bituminous coals, show that these have been compressed or reduced to one-tenth to one-fortieth of their original mass. Let it be assumed that they have suffered a reduction to only one-tenth of their original mass, which is a very conservative figure. Let it also be assumed that none of the sulfur contained in the peat bog has been removed, which is a fair assumption, since the consensus of opinion is that all the sulfur contained in coal has been carried into the deposit from the outside. There could, therefore, not have been a loss. There are found in peats, according to the figures given above, from 0.29 to 4.21 per cent. of sulfur. If there were then only a reduction to one-tenth of the original mass, and no loss of sulfur, there would be from 2.9 to 42.1 per cent. of sulfur in the resulting coal. There is more than enough sulfur in peat to account for all the sulfur in coal.

*Pyrites in Peat.*—Peats contain pyrite in the same form as found in the coals. As shown in Figs. 9 and 10, the pyrite nodules in peat are lenticular to spherical in form, with a rough surface, and often several are grouped together in a manner similar to the coal pyrite. They break up into minute cubes when the microtome knife strikes them in making the section.

Although the pyrite globules have not been investigated completely, their origin is of enough importance to be briefly considered. In samples of fresh woody peat are found numerous organisms living in the wood fibers or wood cells. They are stained brilliantly red by safranin, and are clearly differentiated from the peat substances, since resinous and dead peat matter do not take this stain well. They are, in general, of a

spherical form and vary greatly in size within certain limits, but are not much larger than 30 or 40 microns. Besides a transition in size, transitions in certain physical characters are noted. In color, they range from light pale yellow, through yellow, light brown, brown, dark brown, to black; simultaneously with the color, they range from transparent to opaque; and parallel to this there is a transition in staining qualities, from a stage in which they take a bright stain through stages in which the staining quality fades away together with an increase in natural color to a point where they are black and opaque, and where they will not take a stain. These stages of the various characters and qualities indicate that they represent different stages in the age of the organisms. Often all the stages may be observed in the same section or preparation. The black opaque stage, and the one that will no longer admit of a stain, is the one that will break into minute cubes, and is the stage recognized as pyrite globules.

The organism is of the plasmodium type, and has been obtained, up to a certain stage, in pure cultures in proper culture solutions under anaerobic conditions. These organisms are evidently either sulfur bacteria or iron bacteria. Their study has not been completed and many points in their history and behavior have not been satisfactorily cleared up. They require further study, and the observations made on them need further verification.

There are then disseminated through the peats pyrite globules of microscopic size similar in appearance, form, and behavior to those in the coals; similar pyrite globules are disseminated through the lignites and the sub-bituminous coals. They can be traced clearly from the peats to the bituminous coals in a continuous chain. On those found in the peats, there appears to be evidence that they are of organic origin. There is, therefore, reason to assume that they are of the same origin in the lignites and the sub-bituminous coals, the next steps above peat in the succession of coal formation. If this is correct, then there is every reason to believe that those in the coals are of the same origin.

## Summary

All the coals that have been examined microscopically contain microscopic grains of pyrite disseminated through them. These are distributed very irregularly and usually occur in colonies. Different coal seams vary in the total content of this form of pyrite; different horizons differ in the total content; and different parts of a section may differ widely in the number of globules present.

The majority of the globules are roughly spherical in form with a rough surface. They readily break into numerous minute cubes. Coals also contain amicroscopic sulfur, probably as organic sulfur.

Although the presence of organic sulfur has been known for a long time its chemical form is not known.

Plants contain sulfur in two forms; as a component of proteins and as non-protein sulfur. The sulfur in proteins is universally present in all plants while the other form occurs only in certain families, but in some of these it occurs in relatively large amounts. On decomposition, the sulfur in plants is set free mainly as hydrogen sulfide.

The sulfur is taken in by the higher plants as a sulfate; hydrogen sulfide is not available and must first be oxidized to a sulfate. Hydrogen sulfide may be oxidized under strongly reducing conditions through the agency of sulfur bacteria, resulting in sulfates. All plant ashes contain some sulfur.

Sulfur is present in peat bogs from 0.29 to 4.21 per cent. Calcium sulfate often crystallizes out when peat is dried slowly. Peats contain pyrite in the form of microscopic grains, similar to those found in the lignites, sub-bituminous and bituminous coals. There is evidence to show that the pyrite grains in peat are of organic origin.

The reasoning on the origin of sulfur in coal applies to microscopic pyrite and organic sulfur; lenses, balls, and sheets of pyrite may have a secondary origin.

ACKNOWLEDGMENT

The writer is deeply indebted to Mr. A. C. Fieldner, Supervising Chemist of the Pittsburgh Experiment Station of the Bureau of Mines, for valuable suggestions and assistance in the work.

## DISCUSSION

R. D. HALL, New York, N. Y.—My point of attack of the problem of how the sulfur entered the coal bed was altogether different from that of Dr. Thiessen. It always seemed to me that it was strange that there was so much sulfur in coal and so little in the measures that included the coal; that is, that the measures above, and the measures below the coal showed very little sulfur. F. W. Clark makes the observation that there is only 0.1 per cent. of sulfur in the lithosphere, that is in the immediate outside crust of the earth; if we could get coal of sulfur content no higher than that in contiguous rock measures, we should be much pleased. Yet there is about 1.1 per cent. of sulfur in the igneous rocks. We must therefore find some place in which that sulfur segregated, and to a considerable extent, it seems to have collected in anything of vegetable or animal character. We find much of the sulfur in the coal. There is from 6 to 10 per cent. of sulfur in Trinidad Asphalt. The natural gas of Ohio and Indiana, which is probably of organic origin, shows 0.17 per cent. The petroleums are also sulfurous, especially those of Texas, California, and Syria.

Sulfur is not often found in even the limestones of the United States. In Pennsylvania, 110 analyses quoted in the Pittsburgh State College report of 1899 and 1900 average 0.08 per cent of sulfur, or less than 0.1 per cent. while 163 analyses ignored the sulfur percentage entirely because it was so small. The highest analysis showed 2.45 per cent. and the next highest 0.314 per cent. E. C. Eckles, in "Building Stones and Clay," shows, in forty-eight slates, an average of 0.202 per cent. and a maximum of 0.92 per cent. with 127 samples without sulfur determination; 628 sandstones with an average of 0.032 per cent. sulfur; six mollusca shells with an average of 0.177 per cent. sulfur and a maximum of 0.324 per cent.; 843 limestones, with an average of 0.102 per cent. sulfur and thirty-nine samples with sulfur undetermined; one marble analysis with sulfur undetermined and twenty-four clays, likewise, with sulfur undetermined. It was evident, from the other cases given, that the sulfur was very low. That was one of the reasons which led me to believe that the sulfur was found in the coal solely because the vegetable substance in the coal had taken it and concentrated it within its substance.

It is interesting to note what Dr. Thiessen has so well said about the large amount of sulfur in vegetation. It is true that some forms of vegetation contain much more sulfur than others. The locoing of horses in the West is said to arise from the action of the barium sulfate in the loco weed, which is sometimes eaten by horses in large quantity. Alfred Dachnowski in his admirable monograph on "Peat Deposits," shows that the sulfur in peat varies from 4.57 per cent. to 0.21 per cent. As the ash in the first sample is 22.14 per cent., the ash- and moisture-free sulfur content would rise to 5.88 per cent. The latter sample has only 7.59 per cent. of ash. So you see there was almost as much sulfur as other ash in the vegetal mass.

The theory appeals to me that the iron in the coal acts as the keeper of the sulfur and that if that iron were absent the sulfur would be driven out. I have always believed that the organic sulfur is probably more easily driven off than the sulfur in pyrite.

Whether we can put a great amount of emphasis in the bacterial theory is, of course, still open to question. Dr. Thiessen thinks that bacteria have a great deal to do with the sulfur in coal. Probably they do, and further evidence of that fact can be adduced from the knowledge that many of the sulfur deposits of the world are not volcanic, but organic, in origin.

Because there are so many volcanoes, it is commonly thought that the large sulfur deposits down in Sicily are volcanic. As they are found in stratified form, there is much reason for believing that bacteria acted as the medium by which these deposits of sulfur were segregated.

It has always seemed to me that when a coke oven was newly charged, it gave off an excessive amount of sulfur. I have been hardly able to

believe that these sulfur fumes were all derived from the pyrite in the coal mass. It is a remarkable thing, also, for us to note that the heating of the coals during their long maturing period cannot have been considerable while the sulfur in the coal was in the form of pyrite, or else we would have found the monosulfide of iron in coal, for this sulfide is very stable. The probabilities are that if the pyrite had been heated to a high temperature, some of the sulfur would have been driven off and we would have found monosulfides of iron or something approaching them, instead of finding all the iron-sulfur compounds in the chemical relation $FeS_2$.

REINHARDT THIESSEN.—In regard to the formation of iron compounds of sulfur, I would like to call your attention to E. C. Harder's *Professional Paper* 113 (1919) of the Geological Survey. He was in the Geological Survey, originally a mining engineer and afterward worked in the Brazilian iron deposits. During his work he made varied observations on the deposits of iron and, in looking for a cause of the origin of the iron, he made observations that these could be due to bacterial action. Lately he found an opportunity in the laboratories of the University of Wisconsin to study iron bacteria. During this time he has made original investigations, as well as a summing up of the knowledge on iron bacteria which also includes, of course, the sulfur bacteria in part. In respect to our problem under consideration, the references to the changes of the sulfur to combine with iron are of great interest to us.

W. H. FULWEILER, Philadelphia, Pa.—Do you consider the formation of the pyrite from the ferrous sulfide in the nature of a secondary reaction after the peat has been subjected to some temperature? Normally, it seems to be rather difficult to form $FeS_2$ in the presence of moisture where the temperature is low, but when the moisture is somewhat driven out and the temperatures caused to rise, then there is considerably more tendency to form the $FeS_2$ than the FeS. That might be the result of secondary transformation after the vegetable matter had begun to be subjected.

R. THIESSEN.—There are four ways in which the sulfur is taken care of in the formation of ferrous sulfate in the presence of decomposing organic matter. (1) Hydrogen sulfide is produced by the decomposition of organic substances by means of bacteria and with a reaction of ferrous salts to ferrous sulfate. (2) Certain sulfate-reducing bacteria in the presence of organic matter take the oxygen from the sulfates, reducing them to sulfides. (3) If sulfides, like calcium sulfide, are present, the carbon dioxide and water transform them into hydrogen sulfide. (4) Bacteria in the presence of decomposing organic matter will reduce free sulfur to hydrogen sulfide.

In the work upon peat, observations have made me believe that bacteria form these little nodules shown in the illustration. There are always present in the woody parts of the peat bacteria that have a gelatinous form, are spherical, and a little larger than these pyritic particles. These organisms ripen into a dull mass and give evidence of a transformation into these pyrite particles. This investigation has not been carried on far enough.

H. C. PORTER, Philadelphia, Pa.—In connection with spontaneous heating of coal, it is of interest to note that Dr. Thiessen calls attention to the fact that there are two physical forms of pyrite that occur in coal. In his reasoning on the origin and occurrence of sulfur, he does not include the lenses, balls and masses of pyrite that occur in large units, but treats rather of the very fine microscopic pyrite. It would be of particular interest to know, approximately, how large a proportion of the total sulfur in ordinary commercial coal brought to market occurs in these balls and lenses. From a practical standpoint, in the spontaneous heating of coal, these large units have very little to do with the heating. But if sulfur in coal has anything to do with the heating, as I think most authorities agree it does in some degree, it is from the finely divided microscopic pyrite rather than from that in the form of balls and lenses.

R. THIESSEN.—Just before I left Pittsburgh, I wanted to look over the original block from which the sections illustrated on p. 914 were made, in order to study the proportion of that part of the coal derived from the woody matter to the attritus. But I found that the block of coal had turned to dust; it is just a powder. I recall another piece of coal that contained this form of sulfur that also had gone entirely into a form of dust.

A. C. FIELDNER, Pittsburgh, Pa.—When the entries of the experimental mine[10] of the U. S. Bureau of Mines were driven in the Pittsburgh Bed at Bruceton, Pa., samples of coal were taken in vertical sections at intervals of 10 to 50 ft. (3 to 15 m.) over a distance of 700 ft. from the outcrop. Analyses of these samples showed that considerable alteration of the coal substance had taken place in the first 40 ft. from the outcrop, the degree of alteration decreasing with distance from the outcrop. The alteration manifested itself in a decrease of calorific value (16.3 per cent. at 5 ft., 6.6 per cent. at 20 ft., etc.), an increase of oxygen (amounting to 10 per cent. of the coal at 5 ft.) with a corresponding decrease of carbon and hydrogen, and a decrease in the total sulfur content in the weathered coal.

[10] H. C. Porter and A. C. Fieldner: Weathering of the Pittsburgh Coal Bed at the Experimental Mine near Bruceton, Pa. U. S. Bureau of Mines *Tech. Paper* 35 (1914) 19–22.

The percentage of sulfur in the weathered parts of the bed was less than half of that in the fresh coal, probably because of oxidation of pyrites to iron sulfate and the leaching effect of surface waters. The fact that half of the original sulfur, however, was still present in the outcrop coal after long exposure and weathering, and that both weathered and unweathered coal contained only traces of sulfate sulfur, indicates either a remarkably slow rate of oxidation of pyrites or the presence of some other form of sulfur, such as organic sulfur, which is more stable toward oxidizing influences. In order to determine the form in which this residual sulfur occurred in the weathered coal, the following experiments were made:

Two samples of weathered and two samples of unweathered coal were carefully crushed to pass through a 40-mesh sieve. All dust finer than 60-mesh was removed. The 40 to 60-mesh material was then subjected to a float-and-sink test in a zinc-chloride solution having a specific gravity of 1.35. The float coal was skimmed off and then stirred into a second zinc-chloride solution of the same specific gravity. In this way most of the pyrite and other free impurities were removed from the nearly pure coal. The float coal was thoroughly washed with a hot 1:6 solution of hydrochloric acid until the washings gave no precipitate or blue coloration with potassium ferrocyanide, and the acid was removed by washing with hot water. After drying the washed coal, determinations of total sulfur, iron, and ash were made. The results are given in Table 1.

TABLE 1.—*Organic Sulfur in Weathered and Unweathered Coal*

| Item | Weathered Coal | | Unweathered Coal | |
|---|---|---|---|---|
| Laboratory number | 12,350 | 12,347 | 12,560 | 12,558 |
| Distance from outcrop, feet | 1 | 5 | 380 | 630 |
| Analysis of dry coal before washing | | | | |
| Ash, per cent | 10 25 | 8 72 | 7 42 | 6 43 |
| Total sulfur, per cent | 0.60 | 0 60 | 2 01 | 1 00 |
| Analysis of float coal, 40 to 60 mesh, dry | | | | |
| Ash, per cent | 1 55 | 2 21 | 4 03 | 3 30 |
| Total sulfur, per cent | 0.67 | 0 68 | 1 21 | 0 90 |
| Iron, per cent | 0 06 | 0 08 | 0 38 | 0 19 |
| Pyritic sulfur, per cent | 0 07 | 0 09 | 0 43 | 0 22 |
| Organic sulfur, per cent | 0.60 | 0 59 | 0 78 | 0.68 |

In the two samples of weathered coal from near the outcrop, little pyrite could be detected with the microscope on examining the residue left after elutriating the powdered coal with water; in the unweathered coal, pyrite was readily collected by the same process, especially in sample 12,560, which contained 2.01 per cent. sulfur. The results of the washing experiments indicate that the sulfur in the weathered coal is

practically all in the form of organic sulfur, as no material lowering of the sulfur content was produced in the specific-gravity separation, and the iron in the ash of the float coal can account for not more than 0.08 per cent. sulfur. The computed figures for organic sulfur in the unweathered coal are of the same magnitude as in the outcrop coal, a relation that may indicate that this form of sulfur is not affected by weathering reactions.

R. D. Hall.—I must protest against the suggestion that you can prove anything about the permanence of sulfur under heat by stating anything about its action under weathering. I am not considering weathering, but heat. I am perfectly willing to believe that the pyrite will weather a great deal faster than the organic sulfur, but I am not so ready to believe that when the coal is heated the organic sulfur will resist reduction. Rather, I surmise that it is first to be driven off and that the pyritic sulfur is driven off later. I cannot feel that my statement is made less strong by anything that is brought up about weathering, which is an altogether different operation.

# Geographic Distribution of Sulfur in West Virginia Coal Beds

BY I. C. WHITE,* PH. D., LL. D., MORGANTOWN, W. VA.

(Chicago Meeting, September, 1919)

ONLY two factors appear to be directly responsible for the geographic distribution of sulfur in the coal fields of West Virginia: these are the

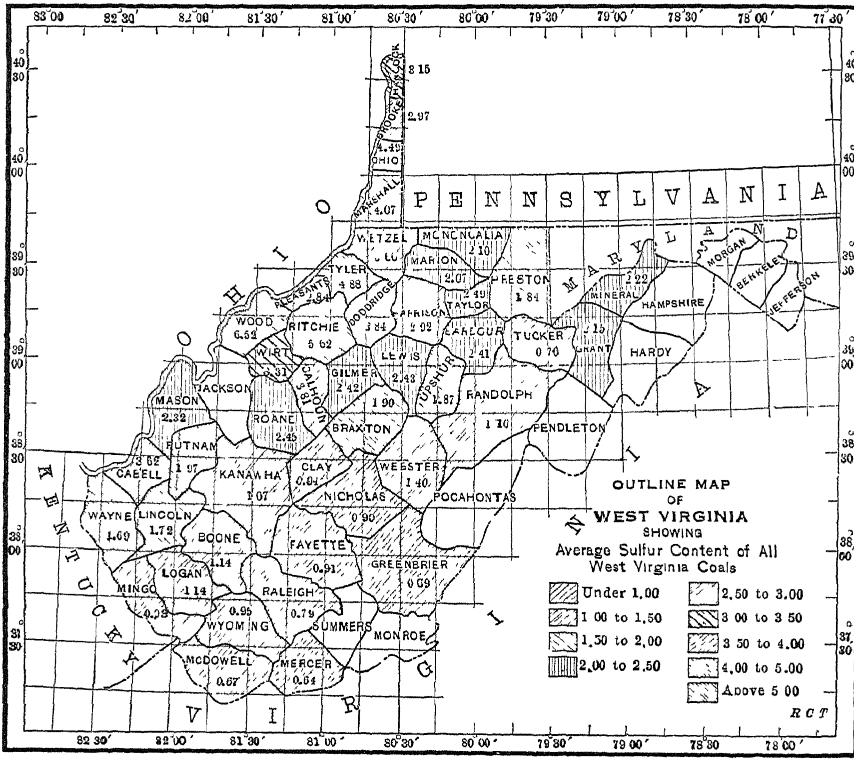


FIG. 1.

stratigraphic position of the coals in question, and the eastward or westward location of the deposits. There appears to be a progressive increase in the sulfur content of West Virginia coals from the oldest Pennsylvanian deposits in the Pocahontas, New River, and Kanawha Groups of the Pottsville Series up through the Allegheny, Conemaugh, and

*State Geologist.

Monongahela, culminating in the thin coals of the Dunkard Series of the Artinsk or Permo-Carboniferous. In these newest coals, a maximum sulfur yield of 8 80 and 9.90 per cent. has been found in the Dunkard Series coal beds of Wood and Ohio Counties along the extreme western boundary of the state. Then, since the lowest and oldest of the Pennsylvanian beds crop out farthest to the east along the Virginia border and successively higher and higher groups make their appearance westward, it follows that there is an increasing amount of sulfur in the West Virginia

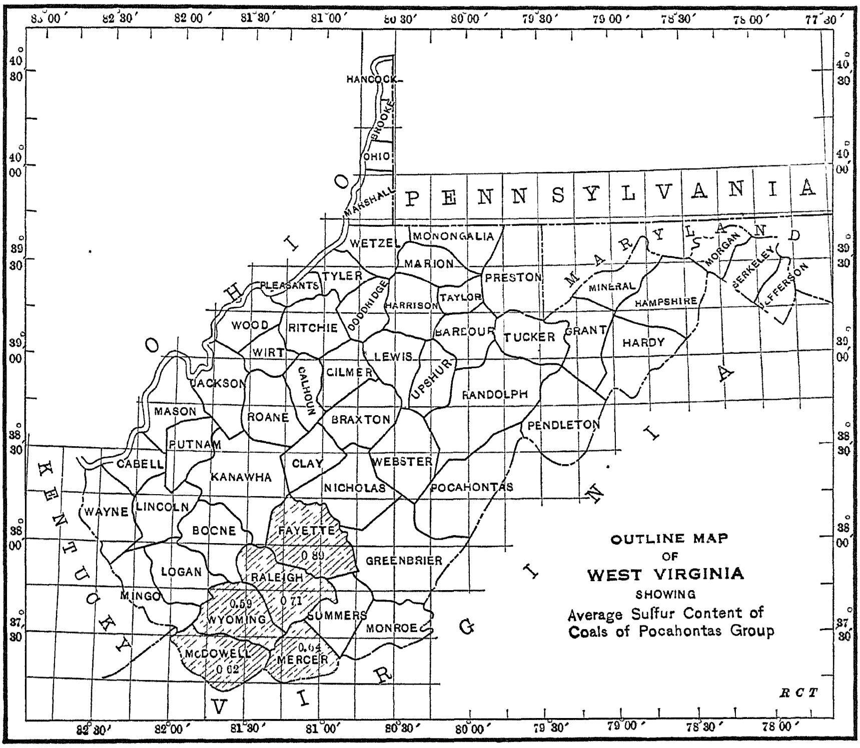


FIG. 2.

coals in going from the east to the west. This law holds good not only for the groups and series, but also for the individual beds where such can be traced over a wide area, like the celebrated Pittsburgh coal. For instance, this coal in Mineral County at the extreme eastern boundary of the Pennsylvanian, contains only 1 to 1½ per cent. of sulfur, and in Maryland often less than 1 per cent., but farther westward this increases to an average of 2.50 in Marion and Monongalia Counties, to 4 per cent. in Marshall and Ohio, and still farther westward, on Captina Creek in the State of Ohio, it is reported to contain over 5 per cent. of sulfur.

Mr. R. C. Tucker of the West Virginia Geological Survey staff has collated all the analyses of West Virginia coals published by the West Virginia Geological Survey and others embodying them into the form of a chart, Table 1, which gives the quantity of sulfur found in each of seventy-two beds of coal in each of the counties of the state where the coals in question occur. This chart also gives the coals by series, and their average for each county, as well as the number of analyses, and the average percentage of sulfur in each bed for the entire state.

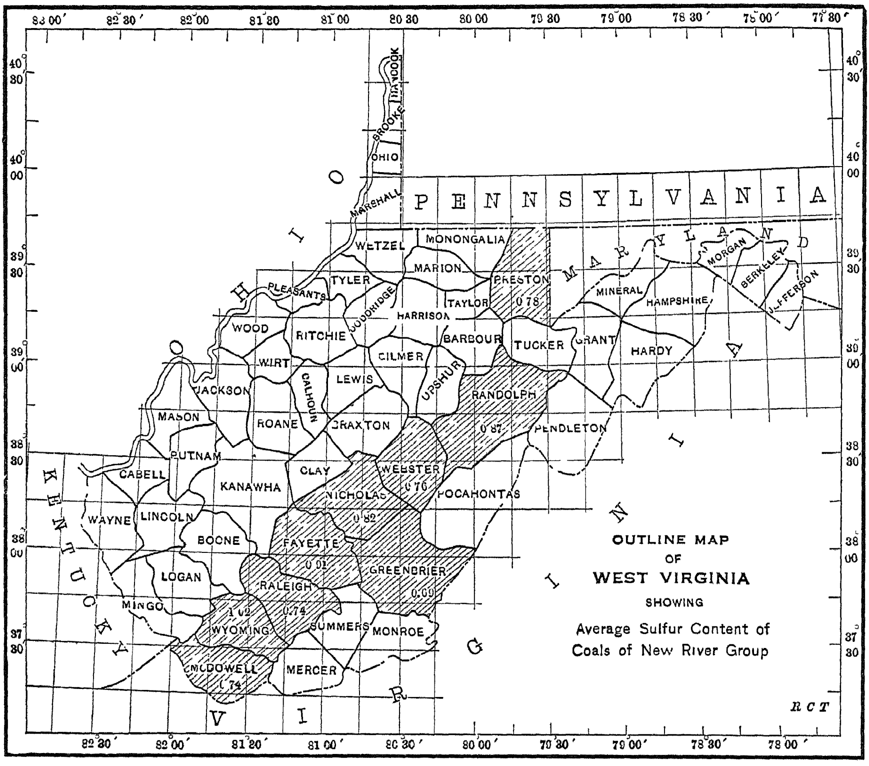


FIG. 3.

Based upon this chart, Mr. Tucker has prepared a series of maps showing how the percentages of sulfur in the West Virginia coals are distributed geographically over the state. Fig. 1 shows the northwestward increase in sulfur content of West Virginia coals by counties. Mr. Tucker has also prepared maps showing the geographical distribution of the several coal groups and series and the average sulfur content by counties for each group or series. These maps, with sulfur legends corresponding to those in Fig. 1, form a very interesting exhibit beginning with the oldest

Pennsylvanian coals, or Pocahontas Group, as shown by Fig. 2, in which the sulfur content is at a minimum in the five counties containing the principal deposits.

The New River Group of coals comes next above the Pocahontas beds; unlike the latter it stretches entirely across the state from northeast to southwest, and, as may be seen from Fig. 3, everywhere averages below 1 per cent. in sulfur except in Wyoming County, where the average (1.02) rises slightly above that figure. It will also be observed that the sulfur

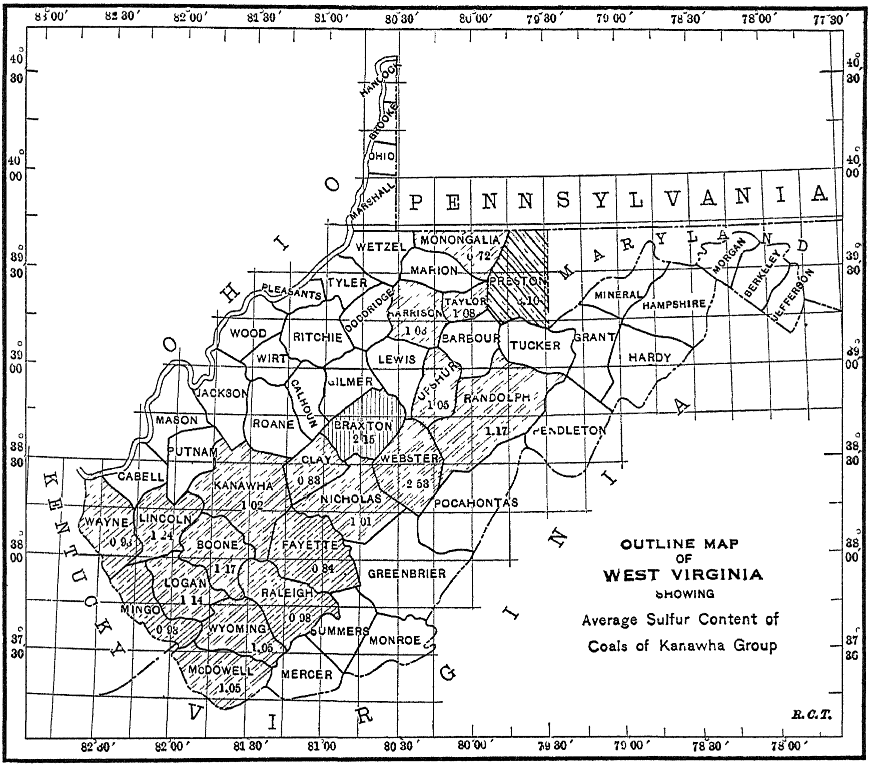


FIG. 4.

in the New River Group averages slightly higher than in the underlying Pocahontas Group of coals. A representative of the New River Group exists in Tucker County, but since no analyses are available from this New River coal in that county the latter is unrepresented on the map.

The Kanawha Group of coals appears to represent mainly the Mercer, Quakertown, and Sharon Coals of western Pennsylvania, but they are thin and of practically very little commercial value in the northern half of West Virginia. They do not attain much value beyond the meridian of Nicholas County and, while still low in sulfur, reveal an increase over

TABLE 1—*Average Percentage of Sulphur in 72 West Virginia Coals by Beds Series and*

| Coal Bed | Barbour | | Boone | | Braxton | | Brooke | | Cabell | | Calhoun | | Clay | | Doddridge | | Fayette | | Gilmer | | Grant | | Greenbrier | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg |
| DUNKARD SERIES | | | | | | | | | | | | | | | | | | | | | | | | |
| Nineveh | | | | | | | | | | | | | | | | | | | | | | | | |
| Hostetter | | | | | | | | | | | | | | | | | | | | | | | | |
| Fish Creek | | | | | | | | | | | | | | | | | | | | | | | | |
| Dunkard | | | | | | | | | | | | | | | | | | | | | | | | |
| Jollytown | | | | | | | | | | | | | | | | | | | | | | | | |
| Washington | | | | | | | | | | | 4 | 1 90 | | | 4 | 3 40 | | | | | | | | |
| Waynesburg "A" | | | | | | | | | | | | | | | | | | | | | | | | |
| Dunkard Series | | | | | | | | | | | 4 | 1 90 | | | 4 | 3 40 | | | | | | | | |
| MONONGAHELA SERIES | | | | | | | | | | | | | | | | | | | | | | | | |
| Waynesburg | | | | | | | | | | | | | | | | | | | | | | | | |
| Uniontown | | | | | 1 | 3 63 | | | | | 3 | 5 99 | | | 2 | 4 71 | | | 2 | 3 69 | | | | |
| Lower Uniontown | | | | | | | | | | | 1 | 5 68 | | | | | | | | | | | | |
| Sewickley | | | | | | | | | | | | | | | | | | | | | | | | |
| Redstone | 7 | 2 23 | | | | | | | | | | | | | | | | | | | | | | |
| Pittsburgh | 13 | 2 67 | | | 16 | 2 17 | 18 | 2 96 | 3 | 3 04 | 2 | 3 41 | | | | | | | 38 | 2 36 | | | | |
| Monongahela Series | 20 | 2 51 | | | 17 | 2 26 | 18 | 2 96 | 3 | 3 04 | 6 | 5 08 | | | 2 | 4 71 | | | 40 | 2 42 | | | | |
| CONEMAUGH SERIES | | | | | | | | | | | | | | | | | | | | | | | | |
| Little Pittsburgh | | | | | | | | | 1 | 5 37 | | | | | | | | | | | | | | |
| Little Clarksburg | | | | | | | | | | | | | | | | | | | | | | | | |
| Elk Lick | 1 | 1 31 | | | | | | | | | | | | | | | | | | | | | | |
| West Milford | | | | | | | | | | | | | | | | | | | | | | | | |
| Harlem | | | | | | | | | | | | | | | | | | | | | | | | |
| Bakerstown | 3 | 2 42 | | | 1 | 1 59 | 1 | 3 15 | | | | | | | | | | | | | | | | |
| Brush Creek | | | | | | | | | | | | | | | | | | | | | | | | |
| Conemaugh Series | 4 | 2 14 | | | 1 | 1 59 | 1 | 3 15 | 1 | 5 37 | | | | | | | | | | | | | | |
| ALLEGHENY SERIES | | | | | | | | | | | | | | | | | | | | | | | | |
| Upper Freeport | 7 | 3 51 | | | 1 | 2 53 | | | | | | | | | | | | | | | 1 | 2 60 | | |
| Lower Freeport | | | | | | | | | | | | | | | | | | | | | | | | |
| Upper Kittanning | 8 | 2 66 | | | 2 | 0 75 | | | | | | | 3 | 1 67 | | | | | | | | | | |
| Middle Kittanning | | | | | 1 | 0 74 | | | | | | | 1 | 0 80 | | | | | | | | | | |
| Lower Kittanning | 29 | 2 02 | 13 | 0 75 | 3 | 1 07 | | | | | | | 7 | 0 86 | | | 8 | 0 72 | | | 2 | 1 93 | | |
| Clarion | 2 | 2 68 | | | 1 | 0 84 | | | | | | | | | | | | | | | | | | |
| Allegheny Series | 46 | 2 39 | 13 | 0 75 | 8 | 1 10 | | | | | | | 11 | 1 07 | | | 8 | 0 72 | | | 3 | 2 15 | | |
| KANAWHA GROUP | | | | | | | | | | | | | | | | | | | | | | | | |
| Upper Mercer | | | | | | | | | | | | | | | | | | | | | | | | |
| Stockton (Lower Mercer) | | | 6 | 0 85 | | | | | | | | | | | | | | | | | | | | |
| Quakertown | | | | | | | | | | | | | | | | | | | | | | | | |
| Coalburg | | | 13 | 1 04 | | | | | | | | | 13 | 0 83 | | | 6 | 0 69 | | | | | | |
| Little Coalburg | | | | | | | | | | | | | | | | | | | | | | | | |
| Buffalo Creek | | | | | | | | | | | | | | | | | | | | | | | | |
| Winifrede | | | 21 | 1 22 | | | | | | | | | | | | | | | | | | | | |
| Chilton "A" | | | | | | | | | | | | | | | | | | | | | | | | |
| Chilton | | | 9 | 0 91 | | | | | | | | | | | | | | | | | | | | |
| Little Chilton | | | 1 | 0 65 | | | | | | | | | | | | | | | | | | | | |
| Hernshaw | | | 15 | 1 27 | | | | | | | | | | | | | | | | | | | | |
| Dingess | | | | | | | | | | | | | | | | | | | | | | | | |
| Williamson | | | 3 | 1 52 | | | | | | | | | | | | | | | | | | | | |
| Cedar Grove (Thacker) | | | 19 | 1 42 | 1 | 0 67 | | | | | | | | | | | | | | | | | | |
| Lower Cedar Grove | | | | | | | | | | | | | | | | | | | | | | | | |
| Alma | | | 38 | 1 32 | | | | | | | | | | | | | | | | | | | | |
| Peerless | | | | | | | | | | | | | | | | | | | | | | | | |
| No 2 Gas | | | 42 | 1 08 | | | | | | | | | | | | | 116 | 0 92 | | | | | | |
| Powellton | | | 2 | 0 61 | | | | | | | | | | | | | 28 | 0 94 | | | | | | |
| Matewan | | | | | | | | | | | | | | | | | | | | | | | | |
| Eagle | | | 10 | 0 79 | 1 | 3 63 | | | | | | | | | | | 39 | 0 83 | | | | | | |
| Little Eagle | | | | | | | | | | | | | | | | | 4 | 0 71 | | | | | | |
| Cedar | | | | | | | | | | | | | | | | | | | | | | | | |
| Lower War Eagle | | | | | | | | | | | | | | | | | | | | | | | | |
| Glenalum Tunnel | | | | | | | | | | | | | | | | | 2 | 0 58 | | | | | | |
| Gilbert | | | | | | | | | | | | | | | | | 2 | 0 93 | | | | | | |
| Douglas | | | | | | | | | | | | | | | | | 1 | 0 97 | | | | | | |
| Kanawha Group | | | 177 | 1 17 | 2 | 2 15 | | | | | | | 13 | 0 83 | | | 198 | 0 84 | | | | | | |
| NEW RIVER GROUP | | | | | | | | | | | | | | | | | | | | | | | | |
| Hughes Ferry (Iaeger) | | | | | | | | | | | | | | | | | 1 | 1 47 | | | | | | |
| Lower Iaeger | | | | | | | | | | | | | | | | | | | | | | | | |
| Castle | | | | | | | | | | | | | | | | | | | | | | | | |
| Sewell "B" | | | | | | | | | | | | | | | | | | | | | | | | |
| Sewell | | | | | | | | | | | | | | | | | 1067 | 0 95 | | | | | 1 | 0 69 |
| Welch | | | | | | | | | | | | | | | | | | | | | | | | |
| Little Raleigh | | | | | | | | | | | | | | | | | | | | | | | | |
| Beckley | | | | | | | | | | | | | | | | | | | | | | | | |
| Fire Creek | | | | | | | | | | | | | | | | | 144 | 0 68 | | | | | | |
| Little Fire Creek | | | | | | | | | | | | | | | | | 1 | 0 66 | | | | | | |
| No 9 Pocahontas | | | | | | | | | | | | | | | | | | | | | | | | |
| No 8 Pocahontas | | | | | | | | | | | | | | | | | | | | | | | | |
| New River Group | | | | | | | | | | | | | | | | | 1213 | 0 91 | | | | | 1 | 0 69 |
| POCAHONTAS GROUP | | | | | | | | | | | | | | | | | | | | | | | | |
| No. 7 Pocahontas | | | | | | | | | | | | | | | | | | | | | | | | |
| No 6 Pocahontas | | | | | | | | | | | | | | | | | 3 | 0 54 | | | | | | |
| No. 4 Pocahontas | | | | | | | | | | | | | | | | | | | | | | | | |
| No. 3 Pocahontas | | | | | | | | | | | | | | | | | 1 | 1 92 | | | | | | |
| No. 2 Pocahontas | | | | | | | | | | | | | | | | | | | | | | | | |
| Pocahontas Group | | | | | | | | | | | | | | | | | 4 | 0 89 | | | | | | |
| MAUCH CHUNK SERIES | | | | | | | | | | | | | | | | | | | | | | | | |
| Pluto | | | | | | | | | | | | | | | | | | | | | | | | |
| Big Spruce Knob | | | | | | | | | | | | | | | | | | | | | | | | |
| Mauch Chunk Series | | | | | | | | | | | | | | | | | | | | | | | | |
| Averages, all coal beds | 70 | 2 41 | 190 | 1 14 | 28 | 1 90 | 19 | 2 97 | 4 | 3 62 | 10 | 3 81 | 24 | 0 94 | 6 | 3 84 | 1423 | 0 91 | 40 | 2 42 | 3 | 2 15 | 1 | 0 69 |

| Coal Bed | Hancock | | Harrison | | Kanawha | | Lewis | | Lincoln | | Logan | | Marion | | Marshall | | Mason | | McDowell | | Mercer | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg |
| DUNKARD SERIES | | | | | | | | | | | | | | | | | | | | | | |
| Nineveh | | | | | | | | | | | | | | | | | | | | | | |
| Hostetter | | | | | | | | | | | | | | | | | | | | | | |
| Fish Creek | | | | | | | | | | | | | | | | | | | | | | |
| Dunkard | | | | | | | | | | | | | | | | | | | | | | |
| Jollytown | | | | | | | | | | | | | | | | | | | | | | |
| Washington | | | | | | | | | | | | | 1 | 2 98 | | | | | | | | |
| Waynesburg "A" | | | | | | | | | | | | | 1 | 4 32 | 1 | 2 88 | | | | | | |
| Dunkard Series | | | | | | | | | | | | | 2 | 3 65 | 1 | 2 88 | | | | | | |
| MONONGAHELA SERIES | | | | | | | | | | | | | | | | | | | | | | |
| Waynesburg | | | | | | | | | | | | | 5 | 2 74 | 4 | 3 73 | | | | | | |
| Uniontown | | | | | | | 1 | 3 31 | | | | | | | | | | | | | | |
| Lower Uniontown | | | | | | | | | | | | | 1 | 1 79 | | | | | | | | |
| Sewickley | | | | | | | | | | | | | 17 | 4 17 | 3 | 3 30 | | | | | | |
| Redstone | | | 7 | 1 92 | | | 18 | 2 80 | | | | | | | | | | | | | | |
| Pittsburgh | 2 | 3 80 | 138 | 2 93 | 14 | 1 93 | 7 | 2 44 | | | | | 87 | 1 59 | 22 | 4 29 | 13 | 2 04 | | | | |
| Monongahela Series | 2 | 3 80 | 145 | 2 88 | 14 | 1 93 | 26 | 2 73 | | | | | 110 | 2 04 | 29 | 4 11 | 13 | 2 04 | | | | |
| CONEMAUGH SERIES | | | | | | | | | | | | | | | | | | | | | | |
| Little Pittsburgh | | | | | | | | | | | | | | | | | 2 | 4 10 | | | | |
| Little Clarksburg | | | 1 | 8 22 | | | | | | | | | | | | | | | | | | |
| Elk Lick | | | | | | | 7 | 2 22 | | | | | | | | | | | | | | |
| West Milford | | | 2 | 4 96 | | | | | | | | | | | | | | | | | | |
| Harlem | | | 5 | 2 82 | | | | | | | | | | | | | | | | | | |
| Bakerstown | | | | | | | 5 | 3 12 | | | | | | | | | | | | | | |
| Brush Creek | 8 | 2 51 | | | | | | | | | | | 1 | 2 41 | | | | | | | | |
| Conemaugh Series | 8 | 2 51 | 8 | 4 03 | | | 12 | 2 59 | | | | | 1 | 2 41 | | | 2 | 4 10 | | | | |
| ALLEGHENY SERIES | | | | | | | | | | | | | | | | | | | | | | |
| Upper Freeport | | | | | | | | | 17 | 3 17 | | | | | | | | | | | | |
| Lower Freeport | 4 | 3 44 | | | | | | | | | | | | | | | | | | | | |
| Upper Kittanning | | | 1 | 2 40 | | | 1 | 3 86 | | | 1 | 0 74 | | | | | | | | | | |
| Middle Kittanning | 1 | 5 86 | | | | | | | | | | | | | | | | | | | | |
| Lower Kittanning | 3 | 3 83 | | | 28 | 1 21 | 7 | 0 85 | 22 | 1 10 | | | 3 | 1 84 | | | | | | | | |
| Clarion | 1 | 1 14 | | | | | | | | | | | | | | | | | | | | |
| Allegheny Series | 9 | 3 58 | 1 | 2 40 | 28 | 1 21 | 8 | 1 23 | 39 | 2 01 | 1 | 0 74 | 3 | 1 84 | | | | | | | | |
| KANAWHA GROUP | | | | | | | | | | | | | | | | | | | | | | |
| Upper Mercer | | | | | | | | | | | | | | | | | | | | | | |
| Stockton (Lower Mercer) | | | 1 | 1 03 | 25 | 0 93 | | | 17 | 1 38 | | | | | | | | | | | | |
| Quakertown | | | | | | | | | | | | | | | | | | | | | | |
| Coalburg | | | | | 51 | 0 77 | | | | | 5 | 0 83 | | | | | | | | | | |
| Little Coalburg | | | | | | | | | | | 1 | 0 88 | | | | | | | | | | |
| Buffalo Creek | | | | | | | | | | | | | | | | | | | | | | |
| Winifrede | | | | | 49 | 0 66 | | | 2 | 0 83 | 2 | 0 74 | | | | | | | | | | |
| Chilton "A" | | | | | | | | | | | | | | | | | | | | | | |
| Chilton | | | | | | | | | 2 | 0 70 | 23 | 0 70 | | | | | | | | | | |
| Little Chilton | | | | | | | | | | | | | | | | | | | | | | |
| Hernshaw | | | | | | | | | | | | | | | | | | | | | | |
| Dingess | | | | | | | | | | | 2 | 0 75 | | | | | | | | | | |
| Williamson | | | | | | | | | | | 1 | 0 62 | | | | | | | | | | |
| Cedar Grove (Thacker) | | | | | 27 | 0 89 | | | | | 28 | 1 32 | | | | | | | | | | |
| Lower Cedar Grove | | | | | | | | | | | | | | | | | | | | | | |
| Alma | | | | | | | | | | | 8 | 1 50 | | | | | | | | | | |
| Peerless | | | | | 6 | 1 12 | | | | | | | | | | | | | | | | |
| No 2 Gas | | | | | 136 | 1 31 | | | 2 | 1 05 | 56 | 1 24 | | | | | | | | | | |
| Powellton | | | | | 5 | 0 64 | | | | | 6 | 1 13 | | | | | | | | | | |
| Matewan | | | | | | | | | | | | | | | | | | | | | | |
| Eagle | | | | | 10 | 0 79 | | | | | 7 | 1 23 | | | | | | | 4 | 1 21 | | |
| Little Eagle | | | | | | | | | | | | | | | | | | | | | | |
| Cedar | | | | | | | | | | | | | | | | | | | | | | |
| Lower War Eagle | | | | | | | | | | | 1 | 0 70 | | | | | | | | | | |
| Glenalum Tunnel | | | | | | | | | | | | | | | | | | | | | | |
| Gilbert | | | | | | | | | | | | | | | | | | | | | | |
| Douglas | | | | | | | | | | | | | | | | | | | 2 | 0 73 | | |
| Kanawha Group | | | 1 | 1 03 | 315 | 1 02 | | | 23 | 1 24 | 140 | 1 14 | | | | | | | 6 | 1 05 | | |
| NEW RIVER GROUP | | | | | | | | | | | | | | | | | | | | | | |
| Hughes Ferry (Iaeger) | | | | | | | | | | | | | | | | | | | 1 | 0 66 | | |
| Lower Iaeger | | | | | | | | | | | | | | | | | | | | | | |
| Castle | | | | | | | | | | | | | | | | | | | | | | |
| Sewell "B" | | | | | | | | | | | | | | | | | | | | | | |
| Sewell | | | | | | | | | | | | | | | | | | | 30 | 0 74 | | |
| Welch | | | | | | | | | | | | | | | | | | | 22 | 0 70 | | |
| Little Raleigh | | | | | | | | | | | | | | | | | | | 1 | 0 77 | | |
| Beckley | | | | | | | | | | | | | | | | | | | 15 | 0 77 | | |
| Fire Creek | | | | | | | | | | | | | | | | | | | 1 | 0 77 | | |
| Little Fire Creek | | | | | | | | | | | | | | | | | | | | | | |
| No 9 Pocahontas | | | | | | | | | | | | | | | | | | | | | | |
| No 8 Pocahontas | | | | | | | | | | | | | | | | | | | | | | |
| New River Group | | | | | | | | | | | | | | | | | | | 79 | 0 74 | | |
| POCAHONTAS GROUP | | | | | | | | | | | | | | | | | | | | | | |
| No. 7 Pocahontas | | | | | | | | | | | | | | | | | | | 1 | 0 70 | | |
| No 6 Pocahontas | | | | | | | | | | | | | | | | | | | | | 13 | 0 58 |
| No. 4 Pocahontas | | | | | | | | | | | | | | | | | | | 25 | 0 64 | | |
| No. 3 Pocahontas | | | | | | | | | | | | | | | | | | | 140 | 0 62 | 12 | 0 65 |
| No. 2 Pocahontas | | | | | | | | | | | | | | | | | | | | | 1 | 1 07 |
| Pocahontas Group | | | | | | | | | | | | | | | | | | | 160 | 0 62 | 56 | 0 61 |
| MAUCH CHUNK SERIES | | | | | | | | | | | | | | | | | | | | | | |
| Pluto | | | | | | | | | | | | | | | | | | | | | | |
| Big Spruce Knob | | | | | | | | | | | | | | | | | | | | | | |
| Mauch Chunk Series | | | | | | | | | | | | | | | | | | | | | | |
| Averages, all coal beds | 19 | 3 15 | 155 | 2 92 | 357 | 1 07 | 46 | 2 43 | 62 | 1 72 | 141 | 1 14 | 116 | 2 07 | 36 | 4 07 | 15 | 2 32 | 251 | 0 67 | 56 | 0 64 |

*Counties from 4193 Analyses by W Va Geological Survey and Others*

| Mineral | | Mingo | | Monongalia | | Nicholas | | Ohio | | Pleasants | | Pocahontas | | Preston | | Putnam | | Raleigh | | Randolph | | Ritchie | | Roane | | Taylor | | Tucker | | Tyler | | Upshur | | Wayne | | Webster | | Wetzel | | Wirt | | Wood | | Wyoming | | State | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No. | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No. | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg | No | Avg |
| | | | | 1 | 3 42 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 1 47 | | | | | | | 2 | 2 45 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 3 52 | | | | | | | 1 | 3 52 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 3 20 | | | | | | | 1 | 3 20 |
| | | | | 1 | 1 83 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 7 24 | | | | | | | 2 | 4 54 |
| | | | | 1 | 2 37 | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 6 39 | | | | | | | | | | | | | | | 2 | 4 38 |
| | | | | 2 | 2 76 | | | 2 | 9 90 | | | | | | | | | | | | | | | 3 | 2 64 | | | | | 3 | 4 38 | | | | | | | 5 | 3 27 | | | 1 | 8 80 | | | 25 | 3 83 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 3 60 |
| | | | | 5 | 2 63 | | | 2 | 2 90 | | | | | | | | | | | | | | | 3 | 2 64 | | | | | 4 | 4 88 | | | | | | | 9 | 3 42 | | | 1 | 8 80 | | | 35 | 3 81 |
| | | | | 4 | 2 06 | | | 7 | 3 02 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 20 | 2 90 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 4 20 | | | | | | | 10 | 4 59 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 3 74 |
| 1 | 1.73 | | | 6 | 2 49 | | | 3 | 3 10 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 30 | 3 55 |
| | | | | 1 | 1 77 | | | | | | | | | | | | | | | | | | | | | | | | | | | 16 | 2 09 | | | | | | | | | | | | | 49 | 2 42 |
| 1 | 1 56 | | | 12 | 2 43 | | | 12 | 3 13 | | | | | 2 | 1 10 | 9 | 1 84 | | | | | 1 | 5 62 | 4 | 2 01 | 17 | 2 60 | | | | | 1 | 3 08 | 2 | 3 57 | | | | | | | | | | | 434 | 2 54 |
| 2 | 1 65 | | | 23 | 2 35 | | | 22 | 3 25 | | | | | 2 | 1 10 | 9 | 1 84 | | | | | 1 | 5 67 | 4 | 2 01 | 17 | 2 60 | | | | | 17 | 2 15 | 2 | 3 57 | | | 1 | 4 20 | | | | | | | 545 | 2 63 |
| | | | | | | | | | | | | | | | | 2 | 2 53 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 5 | 3 72 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 1 05 | | | | | | | | | | | | | | | | | | | 2 | 4 64 |
| | | | | 1 | 3 30 | | | | | | | | | 2 | 3 16 | | | | | | | | | | | | | | | | | 3 | 1 79 | 2 | 1 48 | | | | | | | | | | | 16 | 2 17 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 4 96 |
| | | | | | | | | | | | | | | 3 | 2 06 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 8 | 2 54 |
| 2 | 2 21 | | | | | | | | | | | | | 8 | 2 17 | | | | | | | | | 2 | 1 42 | 2 | 2 57 | | | | | | | | | | | | | 1 | 3 31 | | | | | 25 | 2 42 |
| | | | | | | | | | | | | | | | | | | | | | | | | 1 | 5 68 | | | | | | | | | | | | | | | | | | | | | 10 | 2 81 |
| 2 | 2 21 | | | 1 | 3 30 | | | | | | | | | 13 | 2 30 | 2 | 2 53 | | | | | | | 3 | 2 83 | 3 | 2 07 | | | | | 3 | 1 79 | 2 | 1 48 | | | | | 1 | 3 31 | | | | | 68 | 2 67 |
| 6 | 2 11 | | | 14 | 1 21 | | | | | 3 | 2 84 | | | 65 | 1 74 | | | | | | | | | | | 2 | 1 42 | 3 | 0 49 | | | 6 | 1 55 | | | 1 | 1 65 | | | | | 1 | 4 23 | | | 127 | 2 00 |
| | | | | | | 1 | 0 80 | 2 | 2 13 | | | | | 1 | 3 95 | | | | | | | | | | | | | | | | | 1 | 4 78 | 1 | 2 72 | | | | | | | | | | | 10 | 3 03 |
| | | | | 1 | 1 42 | 6 | 1 11 | | | | | | | 8 | 2 38 | | | | | | | | | | | 3 | 3 67 | | | | | | | | | 3 | 0 95 | | | | | | | | | 37 | 2 05 |
| | | | | | | 1 | 0 80 | | | | | | | | | | | | | | | | | | | | | | | | | 4 | 2 36 | | | 2 | 0 80 | | | | | | | | | 10 | 1 98 |
| 8 | 2 44 | 1 | 0 77 | 4 | 3 28 | 17 | 0 87 | | | | | | | 18 | 1 39 | | | 3 | 0 58 | 11 | 1 55 | | | | | 8 | 2 48 | 6 | 0 80 | | | 7 | 1 76 | 15 | 1 66 | 11 | 1 49 | | | | | | | | | 232 | 1 44 |
| | | | | | | 2 | 1 04 | | | | | | | 6 | 2 19 | | | | | 1 | 1 86 | | | | | | | | | | | | | | | | | | | | | | | | | 13 | 1 88 |
| 14 | 2 30 | 1 | 0 77 | 19 | 1 66 | 27 | 0 93 | 2 | 2 13 | 3 | 2 84 | | | 98 | 1 78 | | | 3 | 0 58 | 12 | 1 57 | | | | | 11 | 2 61 | 9 | 0 76 | | | 18 | 1 99 | 16 | 1 72 | 17 | 1 32 | | | | | 1 | 4 23 | | | 429 | 1 72 |
| | | | | 1 | 0 72 | | | | | | | | | | | | | | | 2 | 2 02 | | | | | 2 | 1 08 | | | | | 2 | 1 21 | | | 1 | 0 96 | | | | | | | | | 8 | 1 29 |
| | | 2 | 0 69 | | | 8 | 1 68 | | | | | | | | | | | | | 1 | 0 73 | | | | | | | | | | | 2 | 0 67 | | | 3 | 0 96 | | | | | | | | | 65 | 1 12 |
| | | | | | | | | | | | | | | 2 | 3 10 | | | | | | | | | | | | | | | | | 1 | 1 66 | | | | | | | | | | | | | 3 | 2 62 |
| | | 17 | 0 77 | | | 17 | 0 94 | | | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 0 68 | 1 | 1 28 | | | | | | | | | 125 | 0 83 |
| | | | | | | 1 | 1 20 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 1 04 |
| | | 11 | 0 75 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 11 | 0 75 |
| | | 2 | 0 63 | | | 2 | 0 62 | | | | | | | | | | | 5 | 0 71 | | | | | | | | | | | | | | | 2 | 0 71 | | | | | | | | | | | 85 | 0 81 |
| | | 1 | 0 66 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 0 66 |
| | | 3 | 1 08 | | | 1 | 0 67 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 38 | 0 78 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 0 65 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 15 | 1 27 |
| | | 2 | 2 11 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 4 | 1 44 |
| | | 4 | 2 12 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 8 | 1 71 |
| | | 36 | 0 91 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 2 25 | | | | | | | 1 | 0 65 | 114 | 1 11 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 0 68 | 2 | 0 68 |
| | | 10 | 0 55 | | | 2 | 2 54 | | | | | | | | | | | 6 | 0 80 | | | | | | | | | | | | | | | | | | | | | | | | | | | 61 | 1 23 |
| | | | | | | 5 | 0 93 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 9 | 0 94 | | | | | | | | | 20 | 0 99 |
| | | 11 | 0 83 | | | 1 | 0 89 | | | | | | | | | | | 8 | 0 81 | 5 | 0 90 | | | | | | | | | | | 3 | 0 95 | 1 | 1 84 | | | | | | | | | 2 | 0 70 | 383 | 1 12 |
| | | | | | | 5 | 1 01 | | | | | | | | | | | 1 | 0 63 | | | | | | | | | | | | | | | | | 1 | 2 78 | | | | | | | | | 48 | 0 96 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 0 91 | 1 | 0 91 |
| | | 8 | 1 34 | | | 28 | 0 72 | | | | | | | | | | | 27 | 1 04 | 4 | 1 18 | | | | | | | | | | | | | | | 12 | 3 91 | | | | | | | 5 | 1 04 | 161 | 1 15 |
| | | | | | | 1 | 0 62 | | | | | | | | | | | 1 | 0 81 | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 1 37 | 8 | 0 88 |
| | | 1 | 1 14 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 1 14 |
| | | 1 | 1 07 | | | | | | | | | | | | | | | 7 | 1 23 | | | | | | | | | | | | | | | | | 1 | 5 52 | | | | | | | 1 | 0 73 | 11 | 1 52 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 0 55 | 3 | 0 86 |
| | | | | | | 9 | 1 35 | | | | | | | | | | | 1 | 0 88 | | | | | | | | | | | | | | | | | 1 | 5 10 | | | | | | | 4 | 1 51 | 17 | 1 54 |
| | | | | | | | | | | | | | | | | | | 1 | 1 87 | | | | | | | | | | | | | | | | | | | | | | | | | 4 | 1 12 | 8 | 1 10 |
| | | 109 | 0 93 | 1 | 0 72 | 80 | 1 01 | | | | | | | 2 | 3 10 | | | 56 | 0 98 | 12 | 1 17 | | | | | 2 | 1 08 | | | | | 8 | 1 05 | 5 | 0 93 | 31 | 2 53 | | | | | | | 23 | 1 05 | 1204 | 1 07 |
| | | | | | | 7 | 1 09 | | | | | | | | | | | | | 2 | 1 19 | | | | | | | | | | | | | | | 3 | 1 38 | | | | | | | 1 | 2 09 | 15 | 1 22 |
| | | | | | | 1 | 0 52 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 0 52 |
| | | | | | | 2 | 0 67 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 0 67 |
| | | | | | | | | | | | | | | | | | | | | 1 | 0 69 | | | | | | | | | | | | | | | 1 | 0 72 | | | | | | | | | 2 | 0 71 |
| | | | | | | 112 | 0 80 | | | | | | | 1 | 0 78 | | | 44 | 0 79 | 25 | 0 85 | | | | | | | | | | | | | | | 38 | 0 75 | | | | | | | 5 | 0 86 | 1332 | 0 92 |
| | | | | | | 1 | 0 84 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 14 | 0 69 | | | | | | | | | 37 | 0 70 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 0 77 |
| | | | | | | | | | | | | | | | | | | 60 | 0 72 | | | | | | | | | | | | | | | | | | | | | | | | | 2 | 1 22 | 77 | 0 74 |
| | | | | | | 1 | 0 84 | | | | | | | | | | | 13 | 0 62 | | | | | | | | | | | | | | | | | 6 | 0 66 | | | | | | | 1 | 0 65 | 166 | 0 68 |
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| | | | | | | | | | | | | | | | | | | 1 | 0 77 | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 0 77 |
| | | | | | | 124 | 0 82 | | | | | | | 1 | 0 78 | | | 118 | 0 74 | 28 | 0 87 | | | | | | | | | | | | | | | 62 | 0 76 | | | | | | | 10 | 1 02 | 1636 | 0 88 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 0 70 |
| | | | | | | | | | | | | | | | | | | 12 | 0 75 | | | | | | | | | | | | | | | | | | | | | | | | | 5 | 0 45 | 33 | 0 62 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 25 | 0 64 |
| | | | | | | | | | | | | | | | | | | 25 | 0 68 | | | | | | | | | | | | | | | | | | | | | | | | | 3 | 0 81 | 211 | 0 64 |
| | | | | | | | | | | | | | | | | | | 3 | 0 73 | | | | | | | | | | | | | | | | | | | | | | | | | | | 4 | 0 82 |
| | | | | | | | | | | | | | | | | | | 40 | 0 71 | | | | | | | | | | | | | | | | | | | | | | | | | 8 | 0 59 | 274 | 0 64 |
| | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 7 06 | | | | | | | | | 1 | 7 06 |
| | | | | | | | | | | | | 1 | 2 78 | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | | 1 | 2 78 |
| | | | | | | | | | | | | 1 | 2 78 | | | | | | | | | | | | | | | | | | | | | | | 1 | 7 06 | | | | | | | | | 2 | 4 92 |
| 18 | 2 22 | 110 | 0 93 | 49 | 2 10 | 251 | 0 90 | 26 | 4 49 | 3 | 2 84 | 1 | 2 78 | 116 | 1 84 | 11 | 1 97 | 217 | 0 79 | 52 | 1 10 | 1 | 5 62 | 10 | 2 45 | 33 | 2 49 | 9 | 0 76 | 4 | 4 88 | 46 | 1 87 | 25 | 1 69 | 111 | 1 40 | 10 | 3 60 | 1 | 3 31 | 2 | 6 52 | 41 | 0 95 | 4193 | 1 29 |

that shown by the coals of the underlying New River. Preston County has no workable coals in this group, and hence the high sulfur (3.10) shown there has no significance, being common to most thin seams in every formation, while the average (2.53) for Webster County will most probably be reduced when more development takes place.

Next above the Kanawha measures comes the Allegheny Series of coals, likewise stretching entirely across the state and extending farther west than the Kanawha Group. Fig. 5 reveals a still higher percentage of

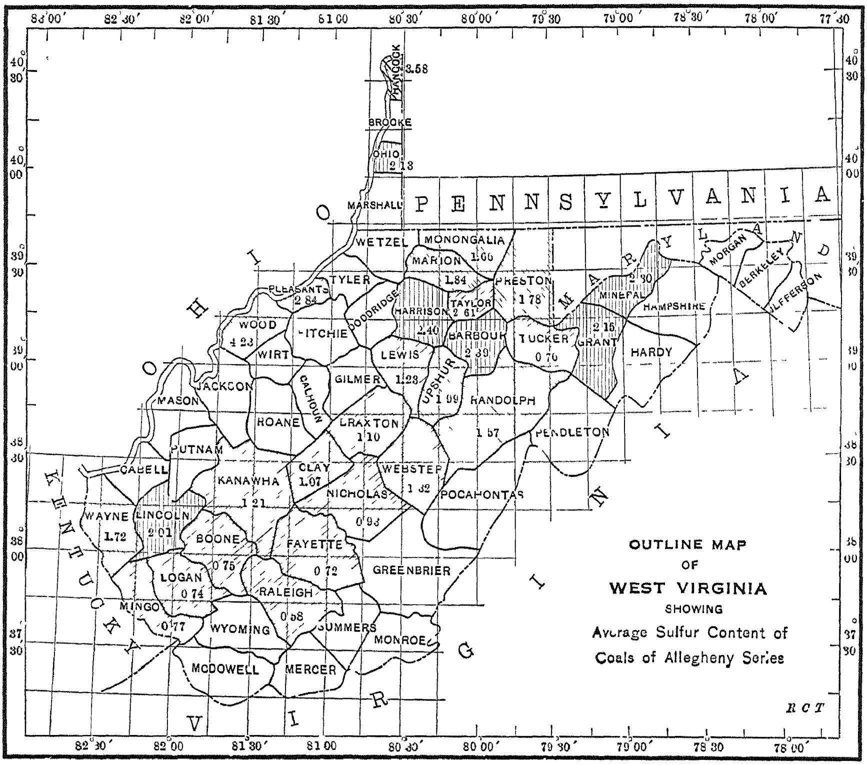


FIG. 5.

sulfur in these coals for the several counties in which samples have been analyzed than is found in the underlying Kanawha Group, as shown by the preceding Fig. 4. It will be noted that, aside from Tucker County, all the very low sulfur counties of the Allegheny Series, like Nicholas, Fayette, Raleigh, Boone, Logan, and Mingo, lie in the southwestern part of the state; also that the extreme northwestern counties like Wood, Pleasants, and Hancock, are highest in sulfur.

The Conemaugh Series of coals overlying the Allegheny beds fall off in their outcrop still farther westward, and reveal an increase in sulfur

content as shown by Fig. 6, over that occurring in the Allegheny coals below, attaining a maximum of 5.37 per cent. in Cabell County at the western boundary of the state.

The Monongahela Series with the great Pittsburgh Coal at its base comes next above the Conemaugh measures and encroaches still farther on the western boundary (Ohio) line of the state. The percentage of sulfur in these coals as a whole, as shown by Fig. 7, is seen to be higher

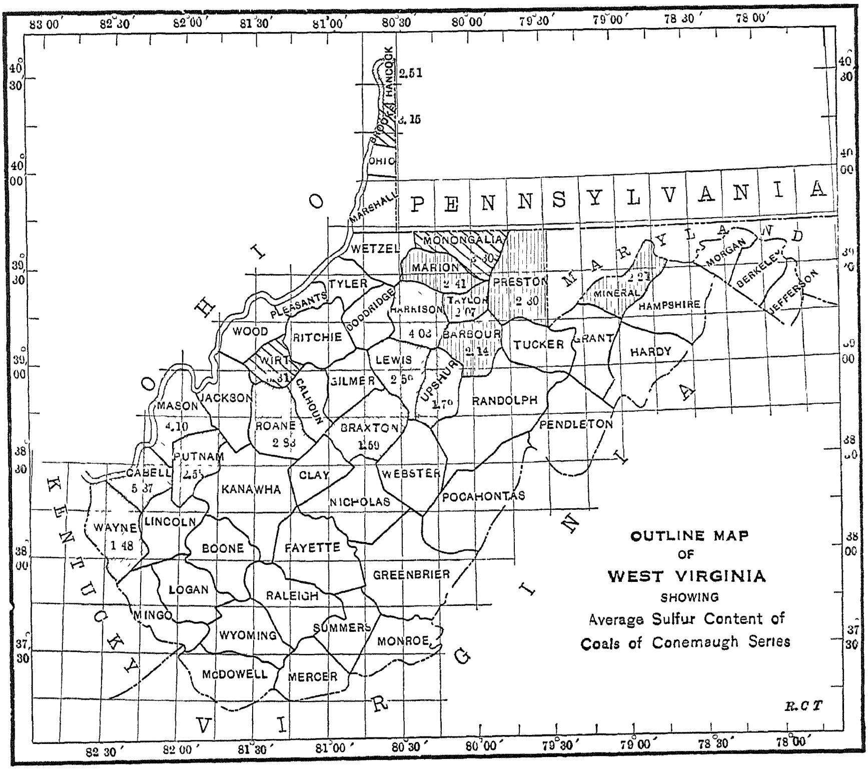


Fig. 6.

than in Fig. 6 which represents the underlying Conemaugh Series of coals.

The Dunkard Series of coals is at the summit of the stratigraphic column of West Virginia, and contains a fossil flora and fauna that assign the beds to the basal Permian, or Artinskian, the coals of which, like those of South Africa and Brazil, are notoriously high in sulfur, those of Brazil averaging 6 per cent. The few analyses of these coals available from West Virginia appear to maintain their reputation for high sulfur, since only one of the West Virginia counties shows less than 2 per cent., while one (Wood) goes to 8 80, and another (Ohio) attains a maximum of

9.90 per cent., the highest sulfur ratio yet found in the Appalachian field and well illustrating the great increase in sulfur content of West Virginia coals in passing from the lowest of the Pennsylvanian (Pocahontas) upward to the summit of the stratigraphic column. The geologic causes that have given origin to this increase of sulfur content with the successively younger and younger age of the several coal groups, the author leaves open for discussion.

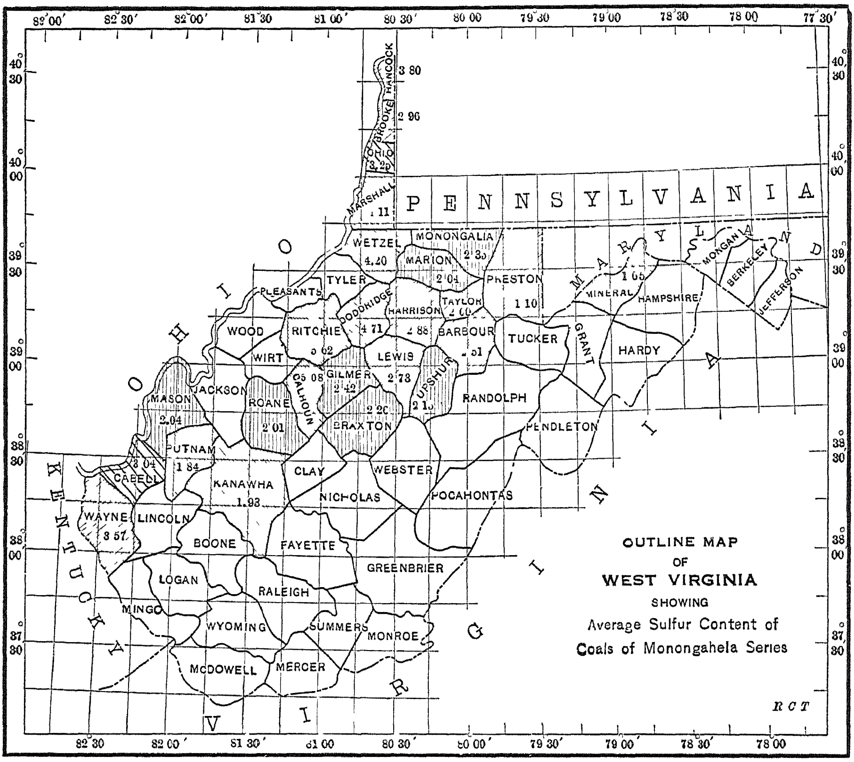


Fig. 7.

## DISCUSSION

J. W. Paul, Pittsburgh, Pa.—The author has not made any contrast with the sulfur content and the other constituents of the coal, and I think it might be well to give that feature some consideration. The coals, as you go in the northwesterly direction in West Virginia, show a gradual increase in sulfur content and I believe that will be true in regard to the volatile content. The southeastern coals, Pocahontas and New River coals, have a much lower volatile content than the coals farther west in the various series.

THE CHAIRMAN (H. H. STOEK, Urbana, Ill.).—The Coal and Coke Committee, in arranging the symposium on sulfur in coal, has tried to secure papers covering the following points: (1) The geological and chemical distribution of the occurrence of sulfur in coal, such as its method of formation, the chemical forms in which sulfur occurs, and the distribution of sulfur through the seam. (2) Geographical distribution of sulfur in the same coal bed in different districts or in the same district. (3) Removal of sulfur by washing or other means. (4) The effect of sulfur in coal upon its use in boilers.

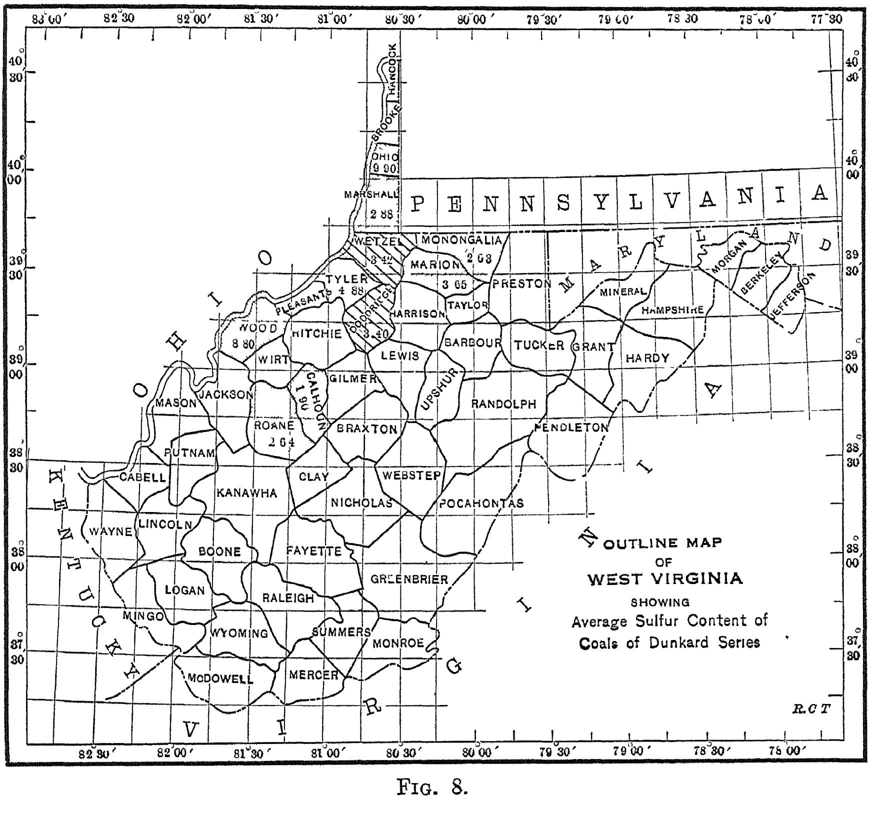


FIG. 8.

S. A. TAYLOR, Pittsburgh, Pa.—I am satisfied from my own investigations that the geographical distribution is only in a general way determinative of the amount of sulfur in coal. Those of you who have gone through a mine taking samples will readily realize that in the same mine the distribution of sulfur is greatly different. In one mine near Fairmount, the sulfur in one part of the mine will run about ½ per cent. and less than ½ mi. away, it will run up to 2 per cent. or more. I have the same information on a number of other mines, so that I doubt very

much whether, as a general rule, it is possible to establish the fact that geographical distribution follows out any such lines as seem to be indicated in this paper. There are a number of phases that present themselves in connection with the sulfur. The one that the Chairman brought out first, as the geographical distribution, seems to me one that ought to carry with it a good deal more investigation than we have had in the past.

The sulfur contents in coal vary so greatly that it is pretty doubtful at the present time whether or not we know exactly how it is formed. Recent investigations have shown that the sulfur is deposited not only in inorganic but in organic forms.

Some thought has been given to the lines of sulfur as following the age of coal. It seems to me that while this is a good clue to start with, yet all we need to do is to think of the conditions that exist in the different coal seams. For example, the older seams in Ohio are the highest in sulfur, while some of the later seams are lower, and the reverse of this is the case in West Virginia; so that the geographical period at which they were laid down cannot form any uniform rule as to the amount of sulfur in coal. Whether or not further investigations will reveal something more specific I am not sure, but I am inclined to think that they will.

Another phase of this, which has been discussed or at least mentioned by some writers, is the relation of the coals to the upheavals, the great upheavals of the Appalachians, for instance. Those nearest the line of rupture have the least sulfur. The heat there generated has apparently taken away some of the sulfur; that, again, cannot be followed fully because we have had examples where the contrary would exist. When we consider the sulfur element and its combinations in varied forms, we are confronted with the fact that there is probably some other condition in the formation of the sulfur in coal, as to its use at least, that needs some further investigation.

Some time ago I visited a mine in the Waynesburg Series. The coal, when used any place except under forced draft burned clearly and gave off good heat, but when put under forced draft the little particles of ash appeared to fuse and form a coating over the coal so that the fire would die out in a very short time. I have seen the same thing in several other seams of coal. In one case, that of a mine I investigated, after we determined the amount of sulfur, which was not so excessively high, we determined the fusion point of the ash; and here we found the difficulty. The fusion point of the ash was so low that, although the sulfur was only medium, we could use that coal only when the fusion point was below that of the ash in other words, not under forced draft.

Some of our research in the future should be along the line of the ash contents, with reference to its combination with the sulfur. Along that line, there is more hope of realizing what we need than by just a

plain statement of so much sulfur. The possibilities of the chemical compounding of the sulfur in the coal with the ash under different degrees of heat, I think, is an item that will require an investigation in the near future, and I am inclined to think will result in more satisfaction to the users of coal, when they know under what condition the sulfur can be best utilized, than the mere statement that coal has a low sulfur content.

A. C. Fieldner, Pittsburgh, Pa.—The Bureau of Mines has made a survey of the fusing, or softening, temperature of the ash of many of the West Virginia coals,[1] especially those from the Pocahontas and New River fields. In general, the softening temperature of coal ash from the various fields of the United States ranges from 1900° to 3100° F. (1038° to 1705° C.). For convenience in discussion, the order of fusibility of an ash may be expressed by subdividing this range of softening temperature into three groups as follows: Class 1, refractory ashes, softening above 2600° F. (1427° C.); class 2, ashes of medium fusibility, softening between 2200° and 2600° F.; class 3, easily fusible ashes, softening below 2200° F. (1205° C.).

The West Virginia coals include representatives of all three classes of fusibility. On the basis of the average of all mines sampled from each coal bed, it is found that with the single exception of the Lower Kittanning bed, all the beds of the Monongahela, Conemaugh, and Alleghany series belong to class 3. There are, of course, individual mines here and there, as, for example, in the Pittsburgh bed in Marion County, which come in the lower part of class 2. Almost all of the mines in the Pocahontas No. 3 bed in McDowell County and half of the mines in Mercer County belong to class 2. The remainder are in the lower part of class 1.

The ash of New River coal is largely in class 1, the Sewell bed averaging over 2580° F. (1412° C.), as compared to 2460° F. (1350° C.) for the Pocahontas No. 3 bed. The other important bed of this field is the Beckley or War Creek, which has a uniformly refractory ash averaging over 2800° F (1536° C.).

In so far as tested, the Kanawha coals comprising the Coalburg, Winifrede, Cedar Grove, No. 2 Gas and Eagle beds show a very refractory ash. Very little if any clinkering should be experienced in using these coals.

G. H. Elmore, Philadelphia, Pa.—Mr. Taylor's remarks bring to mind how sulfur can vary in a single mine. A company in Pennsylvania that owns several coal mines had been selling the product of one of them to a railroad for many years, without any trouble whatsoever, when sud-

[1] W. A. Selvig, W. C. Ratliff, and A C. Fieldner: Fusibility of Coal Ash from the Interior Province Coals as Compared with West Virginia Coals. *Chem. & Met. Eng.* (1919) **20**, 274–276.

denly the coal from this mine was refused because of the high sulfur content. Investigations were made by taking samples at all the headings, all of which were a long distance from the shaft and more or less in a circle. To their amazement, they found the railroad company's assays to be correct and it became necessary to shut down the mine. This case suggests that before large operations are undertaken at any mine, it will be well to find by drilling how the sulfur runs through the field to be operated from that shaft. In this particular instance, a careful investigation of the coal shows that in so far as the sulfur content is concerned, it can be removed to make a good byproduct coke.

# Demonstration Coal Mines*

BY J. J. RUTLEDGE,† PH. D., MCALESTER, OKLA.

(New York Meeting, February, 1920)

THE United States Bureau of Mines established at Bruceton, Pa., in 1909, an experimental mine, for the purpose of testing the means of preventing and limiting mine explosions. During the last ten years numerous explosions have been caused to originate in this mine for investigative purposes and the rate of propagation of the explosion wave, the pressure developed per unit of area by the explosions, and the general results of the explosions have been carefully recorded and studied. Means of preventing mine explosions or of limiting them to the areas in which they originate have been developed. A great deal of valuable information has been derived from the work in this mine and much more useful and valuable information will be obtained in the future. The writer would plead, not for the opening of experimental mines in all the important coal-producing fields in the United States, but for the opening of demonstration mines, or mines in which experiments could be made with the various details of coal mining.

The United States Department of Agriculture has established, in nearly every state, stations that undertake experimental work for the benefit of the agricultural interests. In addition, small demonstration plats are set aside, in suitable locations in the farming districts, where various crops are grown under scientific direction, so that the farmers in the neighborhood can note the results obtained and profit thereby. Something like this should be done for the coal-mining industry. If experimental or demonstration mines were established, methods of working adapted to local conditions could be worked out.

Owing to competition in the same markets, small capitalization, or low profits, it may be an utter impossibility for any one company to try a new method of mining. Labor conditions may prevent the trial of a new method of working. It may be impossible or inadvisable to disturb existing working conditions for fear of causing trouble among the miners through real or fancied changes in the scale of wages.

The first and most important investigation to be undertaken in these demonstration mines would be the trial of various methods of working coal mines until a satisfactory plan had been demonstrated for each

* Published by Permission of the Director, U. S. Bureau of Mines.

† Mining Engineer, U. S. Bureau of Mines.

particular district. It would doubtless be necessary to carry on the work for several years before an acceptable method of working was manifested. The various details of mining associated with such method could then be experimented with and the best methods demonstrated

If a certain method or plan of working had been shown to be the safest and most efficient, public opinion would force coal operators to adopt the new methods of working, if they did not do so voluntarily, and public opinion would also furnish moral support to the operators in overcoming any opposition that the miners and other employees might manifest toward the installation of the new and better method.

If a new method was found to be safer and more economical than the one in use, the authority of the state could be invoked to support any operator who desired to adopt it. Very few coal-mine operators would dare run counter to public opinion, even were they to ignore the financial benefits to be derived from the adoption of the new plan. Compensation insurance companies, through their mine inspectors, would give credit to those mines that adopted the new methods, with the result that their liability insurance would be materially reduced in cost.

It is well known that coal miners flock to new mines and new camps, where the housing and living conditions generally are better than in the older camps, and the miner's working conditions underground can be made satisfactory from the inception of mining operations. A new and better method of mining would not compel the miner to walk long distances to his working place, through poorly brushed haulage ways or manways—perhaps up and down steeply pitching seams. The new plan would take him to his working place, either by means of a safety car or a man-trip, or by a short walk through well-brushed roadways or manways, with a minimum amount of walking. He would then be in condition to perform a good day's work, as he would not be tired by his long walk. The ventilation, also, would be such that the employees would be able to do a good day's work. The operator who adopted the new plan would have his choice of the miners and their work would be efficient.

It seems strange that coal operators will expend large sums of money for improved mining machinery and fail to have conditions, both under and above ground, made such that the employees will be able to perform efficient labor. It is time that technical skill be employed in the underground workings of coal mines to something like the extent to which it has been employed in planning and equipping the surface plant. The most costly surface plant will be useless unless commensurate engineering skill is employed to develop and maintain the underground workings.

No mine can long operate unless a profit is made. Generally speaking, coal-mining companies are of relatively small capitalization and must have an immediate return on the investment—there can be no long wait for dividends, as there is in some other lines of business. No mining

company can afford to abandon the old prevailing methods and try out new methods of mining, no matter how promising, for fear that the new methods will prove to be unsuccessful and there will be no dividends. Again, owing perhaps to close competition, one or more of the mining companies may not be on good terms with the other mining companies in the same field, and hence may hesitate to try out a method of working that may prove to be successful and eventually adopted by the competitor, who will reap the benefits of the new method without having expended either time or money in trying it out. But a demonstration mine, operated under government supervision, supported by government funds —perhaps augmented by financial support from the coal operators of the vicinity—can try out various methods of mining and by experiment find which methods are best adapted to the field. In this way, each method of working would be given an impartial trial under disinterested engineers and miners, whose only purpose would be to learn what method of working was best suited to the region; coal-mining men, including coal miners with a practical working knowledge of improved methods of working as carried out in other fields, could be used as a supervisory consulting board of engineers and miners. Each person on the board would have a voice in determining the plan adopted and in putting it into execution.

Since the mine would be under government supervision and the sole purpose for its operation would be safety and efficiency, it would probably be free from some of the onerous working conditions that frequently are incorporated in the wage agreements in some of the coal fields. The actual true working efficiency of man and machine could be ascertained without the result being subject to doubt or criticism. If results of value were obtained, the entire coal-mining industry in the field where the demonstration mine was located would profit thereby, since the information would be free to all. In like manner, if negative results were obtained the entire coal field would be informed so that others would not try the plan.

Inquiries among coal operators in various coal fields seem to indicate that coal operators will support such demonstration mines both morally and financially. Indeed, it is believed that were such demonstration mines to lack financial support from the government, the various mining companies in the neighborhood of the mine, who would be most interested in the results obtained, could be persuaded to furnish the funds necessary to carry out work in such a mine, the amount necessary being raised by a voluntary assessment on each mining company interested. There would be some revenue from the coal produced in the demonstration mine so that some of the money advanced would be returned to those companies that subscribed to the fund. The coal mined would probably

be of such grade and quality that it would bring a good price since it would be mined under careful supervision.

Since coal seams differ in character and the same seam may greatly change its nature within the distance of a mile and, more over, the nature of the roof or bottom may, and many times does, change within a very short distance so that a method of mining which is successfully followed in one mine may not be at all adapted to the conditions found in a mine a mile distant, it is, in many cases, necessary to develop plans of working suitable for each particular district.

Various methods of mining can be tried out to ascertain whether or not they are adapted to the conditions in the field where they are tried. If the work in the demonstration mine proves successful, from both an engineering and a business standpoint, the methods can, and doubtless will, be adopted by the other mines in the field where the demonstration mine is located.

Among some of the experiments that could be tried are the following:

Some satisfactory method of stopping the slacking of mine roofs during the summer, by the use of the cement-gun or some other form of coating to protect the roof in the mine.

By experiment find what method of mining is best suited to the field: If pillar and room, double entry or entry and air course, or panel system; best width of rooms and pillars, most economical depth of rooms, most satisfactory width and length of room necks, rooms turned off entries at right or at acute angles, concentrated workings. If longwall: Whether advancing or retreating. If advancing: Scotch or 45° system, or face track; by hand mining or by machine. If retreating: angle face or straight face, conveyor or face track.

Better method of timbering entries and air courses; details of longwall advancing: to determine by experiment the angle of break for machine and hand mining and distance between the lines of break in both methods, and the amount of subsidence.

Best methods of timbering roadways and airways in any special coal field: control of squeezes; best methods of causing roof to break; suitable means for combating bottom heaving; proper explosives; most satisfactory mining machines; method for reducing the depreciation of mining properties; improved methods of haulage; better ventilation systems; generation and distribution of power—especially electricity; more efficient pumping arrangements; safer shot-firing methods, mechanical loaders; underground drag-line systems for loading coal into mine cars; a combination method of mining the coal and the overlying or underlying shale and the test of the roof or bottom shale in metallurgical plants or in the manufacture of brick or terra-cotta ware.

## DISCUSSION

H. M. Wilson,* Hartford, Conn. (written discussion†).—Doctor Rutledge makes a most timely suggestion in his paper and one that should have the hearty encouragement of all persons interested in the coal-mine industry. It is unnecessary to tell mining engineers that our coal-mining methods are wasteful, vastly more so than those of some European countries and, in the aggregate, those of a few American mines operated under favorable conditions or with a view to maximum recovery. The visible coal resources of the United States, as with petroleum and a few other resources, are so limited that a few generations will witness their practical exhaustion. Certainly a few generations will necessitate annual reduction in consumption instead of the present great annual increase. The only way in which the life of these fuel supplies can be extended is by more efficient and economic use in consumption, and by more economic, efficient, and complete methods of recovery and preparation in the course of production.

The demonstration mines suggested by Doctor Rutledge should offer the needed medium through which to determine the most efficient and complete methods of extracting and preparing coal for shipment. The cost of operation of these mines would alone be returned to the people of the country as a whole in the economy secured through such demonstrations. Demonstration mines will go a long way toward helping state mine inspectors, insurance inspectors, and mine operators to teach subordinate mine officials and mine workers those means best adapted in each mining district to develop the most safe and healthful method of working. The cost and method of operating such mines should be readily met and determined through one or other of several channels, among which may be mentioned state mining colleges, mine operators associations, special. legislation enactment. The funds should come largely from subscription in stock in the mine by mine owners, and the coordination of the researches and methods to be pursued should be by the Federal Bureau of Mines.

Anything that will add to the safety and welfare of the mine workers will result in a direct profit to the mine operator and a material betterment of employment and labor conditions. In all states where there are workmen's compensation laws, the premium rate for insured mines is reduced proportionately to the improvement of the safety measures adopted to protect the insured employee. In the case of self-insured mines, the same plan is, in some measure, adopted with a view toward reducing compensation cost to the owner. Furthermore, there can be no question of the direct return from expenditures of this nature, since they result

* General Manager The Associated Companies.

† Received Feb. 16, 1920.

in keeping the men on active payroll a larger part of the time than is possible where injuries are more frequent or not promptly given medical care. Then there is the return for increased efficiency of the human machine due to good health and good working and living conditions.

J. A. Ede, La Salle, Ill. (written discussion*)—The writer pleads not "for the opening of the experimental mines in all the important coal-producing fields of the United States, but for the opening of demonstration mines, or mines in which experiments could be made with the various details of coal mining." On page 947, he refers to a demonstration mine operated under government supervision and supported by government funds to learn what method of working was best suited to the region. The writer evidently recognizes the fact that in the same district the *modus operandi* in one mine would be of little service in determining the best methods to be employed in another within a distance of a mile. In order to satisfy the requirements of such a field, it would be necessary to open more than one demonstration mine.

It cannot be logically disputed that experimental or demonstration mines would be a valuable object lesson to the particular part of the field in which they were located and that after carrying the work on "for several years" an "acceptable method of working would be manifested." It should, however, be recognized that the intelligence that would ultimately evolve the presumed acceptable method is at the present time active in the field and if supported by such authority as referred to, on page 946, would go far toward, if not pass, the objective pointed out by the writer.

The project as proposed covers a wide field and embraces so many diverse interests that it should be studied with care. The carrying out of such a scheme is a very different matter from the experimental mine at Bruceton, Pa., and the analogy between such a program and that carried on by the United States Department of Agriculture, while agreeing in substance, differs materially in application.

We believe that a closer relationship between the government or state and the mines now in operation would prove of greater value at the present than the scheme proposed. The writer introduces in his plea some matters that call for urgent consideration. He says that labor conditions may prevent the trial of a new method of mining. It may be impossible or inadvisable to disturb existing working conditions for fear of causing trouble among the miners through real or fancied changes in the scale of wages. The writer is evidently familiar with the economic conditions of the Middle West coal fields; but if the state of affairs he refers to is possible, is this not a condition that the government should see is removed? It can be done without adding to the force now in the

* Received Feb. 14, 1920.

field engaged by the federal and state departments. In one of the middle states the coal-cutting machine is being introduced in longwall mining. One of the operators of such a mine was told that he must assign a certain place or room to each miner. As the plea of the operator to the executive board of the miners is pertinent to the statement Mr. Rutledge has made, the operators' objection is quoted:

"In the competitive field of Illinois, some new method or system must be adopted other than is now in vogue, and I cannot but believe that you will encourage and support any attempt made along lines with this object in view in so far as such anticipated change does not lessen the earning power of the miner. To predetermine our future operations by arbitrary designated positions would seriously interfere with any plan or innovation on our present mode of operation. We therefore object to having to assign a special room to every miner along the machine-cut face. We are entering upon a new phase of mining that has shown how it can favor the earning power of the miner. There is no doubt that the efficient application of the mining machine is the only presentable salvation for this district; and though in its experimental stage, its application should not be hampered by terms foreign to its use and applicable only to the old method of mining. In this case there is no attempt made to interfere with the earning power of the miner, unless it is to increase his wages. We object to having to place our men where we do not want them and our scheme of operation interfered with. We consider the demand made on us to be an infringement of the inalienable rights of the operator to exploit his mine in the way he thinks best."

This, I presume, is only one case of many where labor conditions may prevent the trial of a new method of mining. We hope, however, to learn that in this particular case the action of the person who demanded the claim set forth will not be sustained. For the last few years, there has been a pleasing approachment between the operator and miner.

Reference is made to the importance of conserving the man power and to the general improvement of the roadways and general conditions, which would be a part of the study in the demonstration mines. The writer again shows his appreciation of an important factor, but at no time in the history of coal mining has this factor been so analytically studied by the different operators as during the last few years. In some districts, vocational training is being introduced and the men are being conveyed to their working places in cars along illuminated roadways. I agree with Mr. Rutledge "that it seems strange that coal operators will expend large sums of money for improved mining machinery and fail to have conditions under and above ground made such that the employees will be able to perform efficient labor."

The falls from the roof in a mine in this district were materially reduced after an improvement in the system of regulating the air-current,

more attention being given to the pressure, humidity, and temperature of the circulating current. To facilitate the measurements, standard areas representing the minimum area of the airways were placed in different parts of the mine with a thermometer showing the temperature where the velocity of the air was taken. The mine inspector was able to check some of his measurements from the same location. A study of the humidity and constituent of the air has been given considerable attention by the Bureau of Mines.

Some of the operators, I am informed, are today standardizing on their books the heads of their operating cost so that a closer estimate can be made of the comparative cost of operation, also comparing the result obtained from the different cut of the machines. All the matters mentioned by Mr. Rutledge are the subject of the daily study of the engineer and no better place for their practice can be found than in the mines now in active operation.

Another such mine as at Bruceton might serve a useful purpose in the Middle West and, perhaps, in the extreme West for matters of research, but for the determination of the best method for mining the coal and the study of the various details of mining, there are plenty of mines —the mines are already opened up.

## Low-temperature Carbonization of Coal*

BY S. W. PARR† AND T. E LAYNG,‡ URBANA, ILL.

(New York Meeting, February, 1920)

THE low-temperature carbonization of coal involves the carrying out of the coking process under conditions wherein neither the coal mass nor any of the passageways through which the volatile products pass are heated above 700° or 800° C. For convenience in this discussion, the single number 750° will be used to designate the maximum range. This temperature is not selected arbitrarily; it is the result of certain natural conditions that are inherent in the substances involved. Two of these conditions are sufficiently pronounced to suggest a line of demarcation at this point as follows: (1) Below 750°, all the heavy hydrocarbons are expelled, which means that, at these lower temperatures, the illuminants, the gases of high calorific value, and the condensible oils are discharged; above 750°, there are given off the lean, non-illuminating gases consisting for the most part of hydrogen and marsh gas and having no condensible constituents present. (2) Below 750°, there is substantially no secondary decomposition; above 750°, the volatile products are readily decomposed, forming tars, naphthalene, free carbon, etc.

It is not intended to maintain that no secondary decompositions occur below 750°. Many recent studies have demonstrated the practicability, especially in the presence of catalytic substances, of cracking certain of the hydrocarbon compounds; but at these lower temperatures the step is a moderate one, as, for example, from xylene to toluene or from toluene to an anthracene. These changes are moderate in amount. Not only do the reactions proceed slowly but they are subjected to the decomposing conditions for only a short time. This is evident when it is recalled that at these initial temperatures the decomposition of the coal is very rapid and, if anywhere near a neutral pressure is maintained, the movement of the evolved gases is lively and reduces, correspondingly, the time for the retention of the gases in the passageways where these milder decomposing conditions exist.

It is acknowledged that the maintenance of these temperature conditions at the present time has only an ideal and not a practical status. Actual operations under these conditions, as an industrial accomplish-

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* From material in preparation as a bulletin of the University of Illinois Engineering Experiment Station, by permission of the Director.

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ment, is still in the experimental stage. However, many tendencies and many experimenters, not to say promoters, are working toward this goal and the topic is certain to be one of great interest until we have come much nearer the ideal in practice. It is not the purpose of this paper to discuss possible methods whereby this end may be attained. Any industrial process has in the main only negative results to report when it is in the development stage; otherwise it would not be in that stage. There is a definite value, however, in setting forth in quantitative terms, so far as they may have been determined, the factors that represent the normal yield to be expected if conditions are maintained as planned. An added reason for offering such data is the tendency to make extravagant and unwarranted statements regarding the value and quantities that accompany the low-temperature process.

The apparatus employed in these experiments is capable of handling from 25 to 35 lb. of coal at a charge. The heat is applied by means of an electric current, the amount of resistance wire being so adjusted as to produce a temperature not over 800° C. The cross-section of the retort is 7¾ in. Pyrometer readings are taken at the center of the mass and next to the wall of the retort. The coals employed were mainly from Illinois but the experiments were extended to include samples from Indiana, Kentucky, West Virginia, and Pennsylvania. As already noted, it is the purpose of this paper to give as nearly as possible what may be looked upon as a normal value for the different products obtainable from the various coal samples employed. In this particular, therefore, it is supplementary to bulletins already published by the Engineering Experiment Station of the University of Illinois, wherein chief attention was given to methods of manipulation and where quantitative data as to byproducts were meager and occasionally in error.[1]

The results given in the tables are sufficiently specific and a further discussion must be reserved for the bulletin wherein it is hoped that additional data will be available concerning the composition of certain of the byproducts, especially the tars. Attention may be called to certain items as follows:

First, the temperature conditions were maintained consistently throughout so that uncertainty on that point is eliminated.

Second, the yield of byproducts from a given type of coal is sufficient in form to afford strong presumption as to the fact that these are the normal values that may reasonably be expected under low-temperature carbonization conditions.

Third, the tars are of unusual interest and require further study to arrive at full information concerning this product. The high content of

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[1] S. W. Parr and H. L. Olin: The Coking of Coal at Low Temperature, *Bull* 60 and *Bull.* 79 (1913, 1915).

TABLE 1.—*Analysis of Coal*

| No | Sample | Proximate | | | | Ultimate | | | | | Heat Value in B t u |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Moisture, Per Cent | Volatile Matter, Per Cent | Fixed Carbon, Per Cent | Ash, Per Cent | Sulfur, Per Cent | Hydrogen, Per Cent | Carbon, Per Cent | Nitrogen, Per Cent | Oxygen, Per Cent | |
| 125 | Herron, Williamson Co, Ill . | 6 07 | 33 60 | 50 23 | 10 10 | 2 79 | 4 74 | 69 26 | 1 47 | 5 78 | 12,663 |
| 128 | Harrisburg, Saline Co, Ill. . | 4 83 | 35 32 | 52 87 | 7 00 | 2 11 | 5 01 | 70 94 | 1 59 | 8 52 | 12,840 |
| 129 | Harrisburg, Saline Co, Ill ... | 4 02 | 35 33 | 54 31 | 6 34 | 2 20 | 5 1 | 71 20 | 1 59 | 8 55 | 12,839 |
| 140 | Georgetown, Vermilion Co, Ill | 15 09 | 32 76 | 42 65 | 9 50 | 1 61 | 4 18 | 59 83 | 1 26 | 8 53 | 10,782 |
| 130 | Fairmont, W. Va, Pittsburgh Seam, high volatile ... | 3 38 | 35 34 | 54 01 | 7 27 | 1 28 | 4 94 | 75 84 | 1 50 | 5 59 | 13,624 |
| 131 | Fairmont, W Va, Pittsburgh Seam, high volatile . | 1 32 | 35 62 | 55 78 | 7 28 | 1 40 | 4 73 | 78 15 | 1 49 | 7 03 | 13,916 |
| 132 | Fairmont, W Va., Pittsburgh Seam, high volatile . .... | 3 14 | 35 30 | 54 41 | 7 15 | 1 0 | 4 88 | 77 95 | 1 51 | 4 37 | 13,698 |
| 135 | "Jenner Coal," Somerset Co, Pa, *C* prime seam, low volatile. | 1 84 | 15 81 | 75 55 | 6 80 | 0 78 | 4 10 | 82 24 | 1 41 | 3 83 | 14,243 |
| 136 | "Jenner Coal," Somerset Co, Pa, *C* prime seam, low volatile | 1 43 | 16 21 | 75 61 | 6 75 | 0 72 | 4 21 | 82 06 | 1 39 | 3 44 | 14,283 |

TABLE 2.—*Temperature Readings at Center of Mass*

| No | Sample | End of First Hour, Degrees C | End of Second Hour, Degrees C. | End of Third Hour, Degrees C. | End of Fourth Hour, Degrees C. | Total Time of Carbonization |
|---|---|---|---|---|---|---|
| 125 | Herron, Williamson Co., Ill | 300 | 420 | 590 | 760 | 4 hr. 30 min. |
| 128 | Harrisburg, Saline Co, Ill. | 300 | 430 | 660 | 790 | 4 hr. 15 min. |
| 129 | Harrisburg, Saline Co, Ill | 275 | 390 | 590 | 770 | 4 hr. 30 min. |
| 140 | Georgetown, Vermilion Co, Ill. . . . . . . . . . . . . | 330 | 375 | 580 | 800 | 3 hr. 45 min. |
| 130 | Fairmont, W. Va, Pittsburgh seam, high volatile | 330 | 430 | 590 | 750 | 4 hr. 45 min. |
| 131 | Fairmont, W. Va., Pittsburgh seam, high volatile | 290 | 390 | 610 | 750 | 4 hr. 45 min. |
| 132 | Fairmont, W. Va, Pittsburgh seam, high volatile | 320 | 395 | 550 | 730 | 5 hr. |
| 135 | "Jenner Coal," Somerset Co., Pa, "C prime" seam, low volatile .... ... | 320 | 420 | 505 | 610 | 6 hr. |
| 136 | "Jenner Coal," Somerset Co, Pa, "C prime" seam, low volatile ... . | 320 | 450 | 570 | 730 | 5 hr. |

TABLE 3.—*Coke from Low-temperature Carbonization*

| No. | Sample | Moisture, Per Cent | Ash, Per Cent | Volatile Matter, Per Cent. | Fixed Carbon, Per Cent | Sulfur, Per Cent | B.t u |
|---|---|---|---|---|---|---|---|
| 125 | Herron, Williamson Co, Ill... | 0 11 | 13 48 | 6 01 | 80 40 | 1 89 | 12,627 |
| 128 | Harrisburg, Saline Co, Ill ... | 0 55 | 10 65 | 6 15 | 82 65 | 1 78 | 13,154 |
| 129 | Harrisburg, Saline Co., Ill ... | 0 25 | 9 50 | 11 70 | 78 55 | 1.94 | 13,267 |
| 140 | Georgetown, Vermilion Co, Ill. | 0 22 | 15 59 | 5 02 | 79 17 | 1 47 | |
| 130 | Fairmont, W. Va., Pittsburgh seam, high volatile .... .... | 0 63 | 11 11 | 11 64 | 76 62 | 1 17 | 13,624 |
| 131 | Fairmont, W. Va, Pittsburgh seam, high volatile.... . ... | 0 61 | 10 31 | 5 52 | 83 36 | 1 00 | 13,916 |
| 132 | Fairmont, W. Va., Pittsburgh seam, high volatile...... . | 0 31 | 10 03 | 4 12 | 85.78 | 0.90 | 13,851 |
| 135 | "Jenner Coal," Somerset Co. Pa., C prime seam, low volatile.......... ....... . | 0 29 | 6.75 | 3.83 | 87.69 | 0.77 | 14,430 |
| 136 | "Jenner Coal," Somerset Co; Pa., C prime seam, low volatile...................... | 0 18 | 8 00 | 4.15 | 87.67 | 0.75 | 14,281 |

TABLE 4.—*Tars from Low-temperature Carbonization of Coals*

| No. | Sample of Coal | Yield per Ton, Gallons | Specific Gravity, 15 5° C. | Free Carbon Per Cent |
|---|---|---|---|---|
| 125 | Herron, Williamson Co, Ill. | 19 75 | 1 065 | 1 8 |
| 128 | Harrisburg, Saline Co, Ill. | 22 00 | 1.059 | 0 5 |
| 129 | Harrisburg, Saline Co., Ill. | 23 56 | 1 057 | 0 5 |
| 140 | Georgetown, Vermilion Co, Ill | 13 85 | 1 07 | 0 5 |
| 130 | Fairmont, W. Va., Pittsburgh seam, high volatile. | 28 33 | 1.061 | 0.5 |
| 131 | Fairmont, W. Va., Pittsburgh seam, high volatile. | 25 00 | 1 06 | 0.5 |
| 132 | Fairmont, W. Va, Pittsburgh seam, high volatile. | 29 25 | 1.06 | 0 5 |
| 135 | "Jenner Coal," Somerset Co, Pa, C prime seam, low volatile | 7 15 | 1.1 | 5 5 |
| 136 | "Jenner Coal," Somerset Co., Pa., C prime seam low volatile | 7 0 | 1.1 | 10.7 |

TABLE 5.—*Gas from Low-temperature Carbonization*

| No | Coal Sample | Yield, in Cubic Feet per Pound of Coal | Heat Value in Each Foot, B t u. | | | Sulfur in Each Foot-grain per 100 Ft. | | |
|---|---|---|---|---|---|---|---|---|
| | | | A | B | C | A | B | C |
| 125 | Herron, Williamson Co., Ill. | 3 0 | 967 | 685 | 435 | 244 | 44 | 12 |
| 128 | Harrisburg, Saline Co., Ill. | 3.2 | 900 | 628 | 428 | 391 | 200 | 96 |
| 129 | Harrisburg, Saline Co., Ill | 3.2 | 892 | 676 | 443 | 303 | 206 | 93 |
| 140 | Georgetown, Vermilion Co., Ill. | 3.4 | 845 | 541 | 465 | 198 | 122 | 28 |
| 130 | Fairmont, W. Va., Pittsburgh seam, high volatile. | 3.4 | 995 | 685 | 462 | 404 | 254 | 34 |
| 131 | Fairmont, W. Va., Pittsburgh seam, high volatile. | 3.3 | 950 | 631 | 430 | 444 | 235 | 34 |
| 132 | Fairmont, W. Va., Pittsburgh seam, high volatile. | 3 3 | 946 | 678 | 450 | 318 | 59 | 52 |
| 135 | "Jenner Coal," Somerset Co., Pa., C prime seam, low volatile. | 3.2 | 632 | 421 | 362 | 11 | 2 | 0 |
| 136 | "Jenner Coal," Somerset Co., Pa, C prime seam, low volatile. | 3 7 | 745 | 564 | 410 | 19 | 13 | 0 |

TABLE 6.—*Type of Tars From Low-temperature Carbonization*

| No. | Coal Sample | Specific Gravity 15 5° C. | Free Carbon, Per Cent | Per Cent, Distillation | | | | Per Cent, Fractions up to 300° C. | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | Up to 190° C | 190°–300° | 300°–360° | Pitch | Tar Acids | Amines | Paraffins |
| 125 | Herron, Williams Co., Ill ...... | 1 065 | 1 8 | 2 8 | 33 8 | 24 4 | 38 9 | 45 0 | 3 6 | 8 0 |
| 128 | Harrisburg, Saline Co., Ill.... | 1 059 | 0 5 | 1.4 | 41 3 | 32 8 | 24 5 | 45 0 | 3 0 | 10 0 |
| 129 | Harrisburg, Saline Co., Ill ... | 1 057 | 0 5 | 1 5 | 44 9 | 33 8 | 19 8 | 47 0 | 4 0 | 10 0 |
| 130 | Fairmont, W. Va., high volatile . | 1 061 | 0 5 | 2 0 | 41 0 | 31 0 | 26 0 | 40 0 | 4 0 | 12 5 |
| 131 | Fairmont, W. Va., high volatile . | 1 06 | 0 5 | 2 0 | 36 5 | 31 0 | 30.5 | 36.0 | 4.0 | 12 4 |
| 135 | "Jenner Coal," Somerset Co., Pa., C prime seam, low volatile . | 1 093 | 5 5 | 0 0 | 29 4 | 30 9 | 39 7 | 20 0 | 8 0 | 12 0 |
| 136 | "Jenner Coal," Somerset Co., Pa., C prime seam, low volatile. .... | 1 148 | 10 7 | 0 0 | 24 0 | 33 2 | 42 8 | 20 0 | 6 0 | 10 0 |

free carbon in the last two samples of Table 4 is due to dust mechanically carried over and not to secondary decomposition.

Fourth, the gas yield represents unusually high calorific values. The columns A, B and C, Table 5, represent the first, second, and third foot of gas discharged per pound of coal.

Fifth, the behavior of the sulfur is, in some respects, the most important of all the data. It will receive more detailed discussion in the larger publication.

Sixth, the coke is, in many respects, the most interesting product of all and will be discussed more fully in the larger publication. It is sufficient at this time to say that the so-called non-coking coals of Illinois give promise of being advanced into the class of coking coals.

TABLE 7.—*Type of Gases Produced from Low-temperature Carbonization*

| No. | Coal Sample | $CO_2$ | $O_2$ | $C_2H_4$ | $C_6H_6$ | $H_2$ | CO | $CH_4$ | $C_2H_6$ | N |
|---|---|---|---|---|---|---|---|---|---|---|
| 125 | Herron, Williamson Co., Ill...... | 5.0 | 1.1 | 1.4 | 1.1 | 44.1 | 6.6 | 38.2 | | 2.0 |
| 128 | Harrisburg, Saline Co., Ill........ | 4.7 | 0.7 | 1.9 | 1.4 | 48.1 | 4.5 | 33.0 | 3.1 | 2.1 |
| 129 | Harrisburg, Saline Co., Ill........ | 4.6 | 0.7 | 1.6 | 1.4 | 47.8 | 4.7 | 29.5 | 4.7 | 5.0 |
| 130 | Fairmont, W. Va., Vermilion Co., Ill.......................... | 3.9 | 0.8 | 1.5 | 1.6 | 47.3 | 4.3 | 33.2 | 4.5 | 5.0 |
| 131 | Fairmont, W. Va., Vermilion Co., Ill.......................... | 4.4 | 0.8 | 1.7 | 1.8 | 44.0 | 5.1 | 32.8 | 5.4 | 4.0 |
| 132 | Fairmont, W. Va., Vermilion Co., Ill.......................... | 3.2 | 0.7 | 1.3 | 1.7 | 37.5 | 4.8 | 29.5 | 6.8 | 4.5 |
| 135 | "Jenner Coal," Somerset Co., Pa., C prime seam, low volatile..... | 1.1 | 0.6 | 0.5 | 0.8 | 58.5 | 2.4 | 29.0 | 2.8 | 4.3 |
| 136 | "Jenner Coal," Somerset Co., Pa., C prime seam, low volatile..... | 0.6 | 0.6 | 0.3 | 0.7 | 65.1 | 0.2 | 28.4 | 0.2 | 3.9 |

## Anthracite Mining Costs

Discussion of the paper by R. V. NORRIS, presented at the New York Meeting, February, 1919, and printed in Vol. LXI, p. 323.

EDWARD W. PARKER,* Philadelphia, Pa.—At the New York meeting of the Institute a year ago, Mr. R. V. Norris presented a paper on anthracite mining costs, in which he gave the results of an intensive study of this subject by the Engineers' Committee of the United States Fuel Administration. Within the last 30 days, the Federal Trade Commission has issued an official report on the same subject, which covers a longer period of time and comprises statistics of realization values and margins that were not comprehended in the paper by Mr. Norris. As, however, the data utilized by Mr. Norris formed also a part of the basis for the report of the Federal Trade Commission, a brief digest of the latter may not be out of place at this meeting.

The report of the Federal Trade Commission covers primarily the two calendar years 1917 and 1918, divided into five periods which furnish opportunity for the study of how the influence of certain definite factors affected either the cost of production or the price, or both, during the 24 months. The first period, January to April, 1917, which may be considered the basic period, comprised the 4 months in 1917 during which the wage agreement of May, 1916, without modification, was in effect. During this period, the labor cost on fresh-mined coal is placed by the Commission's report at $1.79 per gross ton, the total cost at $2.66, the average realization at $3.29, and the "margin" (sales realization over total f.o.b. mine cost) at $0.63.

The second period covers the four months from May 1 to August 31, during which time the first of the supplemental wage agreements (supplemental to the agreement of 1916) was in effect. It was also the period during which the summer discounts were in effect and when prices were practically fixed by a voluntary agreement between the operators and the Federal Trade Commission. It was, further, the period in which the 75 cents differential on prepared sizes permitted to the individual operators by the Federal Trade Commission began; at that time, also, on account of war conditions an unusually large quantity of steam sizes were marketed at exceptionally good prices. On account of the two latter conditions, not explained in the Commission's report, the "margin"[1] on operating

* Director, Anthracite Bureau of Information.

[1] "Margin" here is placed in quotation marks for reasons that will be explained later.

cost increased to 79 cents, the total cost having advanced to $2.86, and the average realization price to $3.65.

The third period covers the 3 months, September to November, 1917, directly following the Executive Order of August 23, 1917, fixing the maximum prices of anthracite domestic sizes, and when the supplemental wage agreement of the previous May was still in effect. The prices fixed by the Executive Order were, with one exception (that of pea coal), the same as those agreed upon by the operators and the Federal Trade Commission, and retained the differential of 75 cents a ton permitted to the individual operators. During this period, there was a decrease in the operating "margin" of 2 cents a gross ton, the cost having increased 19 cents a ton and the realization 17 cents.

The fourth period, from December, 1917, to October, 1918, is the longest of the five. It constitutes the eleven months during which another supplemental agreement (that of November 17, 1917) was in effect and prices were still fixed by the Fuel Administration. A part of this period (from November, 1917, to May, 1918) is the time covered by Mr. Norris in his paper on anthracite mining costs. He showed that as a result of the supplemental agreement of November, 1917, the cost of production had increased 76.3 cents per ton, while an advance in price on the domestic sizes of only 35 cents a ton had been permitted by the Fuel Administration. The Trade Commission's report shows that for this entire period the "margin" on the operating cost was 30 cents a ton less than during the preceding period, the f.o.b. cost having increased 66 cents while the average realization had advanced only 36 cents, or 1 cent more than the advance allowed by the Fuel Administration on domestic sizes alone.

The last period covered by the Commission's report is for the two months of November and December, 1918, when the third supplemental agreement, to the 1916 agreement, was in effect. It was estimated that the advances given to labor by this agreement would increase the labor cost about 73 cents a ton on the entire production, or $1.05 if applied to the prepared sizes alone, and this time the Fuel Administration permitted an advance of $1.05 a ton on the prepared sizes. The Commission's report shows that the cost of labor advanced 84 cents a ton instead of 73 cents, the cost of supplies increased 20 cents a ton, and general expenses 9 cents a ton, making a total of $1.13 a ton, while the average realization advanced $1.02, or 11 cents less than the increase in cost, the "margin" dropping to 36 cents, which was 41 cents less than in the period from September to November, 1917, and 27 cents, or 43 per cent., less than in the first period covered by the Commission's report. Moreover, it must be remembered that in the first period the 63-cent margin was on the turn-over of $3.29, or a little less than 20 cents on the dollar. The high-margin period of May–August, 1917, was 79 cents on the turnover of $3.65, or 21.6 cents on the dollar. The 36 cent-margin in the

final period was on a turn-over of $5.20 or a little less than 7 cents on the dollar.

The Commission, in its report, says, "From this 'margin' would have to be paid any sales expenses, interest, and Federal taxes, the remainder being available for surplus and dividends." The words "if any" might well be added after "remainder." The Federal Trade Commission does not permit, as indicated, the inclusion of the items mentioned above in the statement of cost. And yet they are just as much a part of the expense of placing the product upon the market as is the cost of labor. It is, of course, explained that these must be paid out of the margins, and the trained student of economics will so understand, but the ordinary layman, in fact the ordinary legislator in Congress or in a state assembly and the ordinary newspaper writer will pass over that statement, and upon his mind the impression is made that "margin" means profits. It is far from being so.

It is generally conceded that interest on investment and borrowed capital, income and excess profits taxes, and gross or net profits are not properly chargeable to cost of production, but are items that should be deductible from income, though with the exception of "gross or net profit on investment" they are factors that must be considered in estimating the total cost to the producer of putting his product into the hands of the consumer. It is a moot question, however, whether or not selling expenses, non-insurable risks, and extra costs of development work under abnormal conditions should be excluded from the cost statements. In the opinion of many, they are legitimately chargeable to producing cost, as they have the effect of making the margins appear larger than they really are. But the Commission is the court of last resort, as well as the first, in such matters and while its decision is accepted due consideration must be given to the facts in a study of the margins as determined by it. The difference between mine or shop cost and the actual cost of placing a commodity on the market may be, and frequently is, very wide.

The average margin above the f.o.b. mine cost for the 24 months covered by the Commission's report was 58 cents.[2] Mr. Norris, in his paper, states that the capital invested in anthracite mining ranges from $5 to $11 per ton of annual output, and that from $7.50 to $8 might probably be considered a fair average. The normal annual production of anthracite is about 80,000,000 gross tons, which would mean a total investment, at the lower figures given by Mr. Norris, of approximately $600,000,000.

Earnings on capital invested in such a hazardous industry as anthra-

[2] In his paper Mr. Norris figures that the average realization for the 6 months covered by the investigation conducted by the Engineers' Committee of the Fuel Administration was 50 cents a ton, which is in close agreement to that shown by the more extended period covered by the report of the Federal Trade Commission.

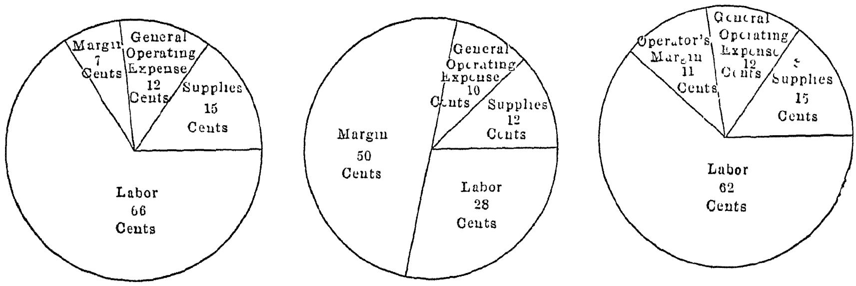
cite mining should not be less than 10 per cent.; in fact, such a percentage of earnings in any manufacturing industry possessing ordinary hazards would be considered very moderate expectation. But estimating the return at only 6 per cent. (money could not be borrowed at that rate to engage in anthracite mining) the interest on the investment, accepting the lower figure given by Mr. Norris ($7.50 per ton of output) would be 45 cents a ton. Selling expenses, according to Mr. S. D. Warriner, in his discussion of Mr. Norris' paper, amount to at least 10 cents a ton, which should be applied to about 80 per cent. of the tonnage included in the Commission's report, or say about 8 cents a ton for the average. These two items alone make up 53 cents of the margin, which leaves only 5 cents a ton as available for the payment of Federal income and excess profits taxes, non-insurable risks, and profits.

As previously stated, the Commission's report is based primarily on the statistics of cost and realization for the two years 1917–1918; these cover operations producing about 99 per cent. of the total output. It also presents, for comparison, data covering the operations of seven companies producing about 42,000,000 tons annually, or more than 50 per cent. of the total tonnage for a period of 6 years, or from January 1, 1913, to December, 1918. During this period the labor cost ranged from $1 56 to $3.31 per ton; the cost of supplies, from 29 to 80 cents; general expenses, from 33 to 61 cents; and the total mine cost from $2.23 to $4.72. The average sales realization varied from $2.59 to $5.11, the highest figures for all being, of course, in the 2 months of November and December, 1918. The "margins" above mine cost were as low as 19 cents in the period from January to March, 1915, and as high as 72 cents from May to August, 1917. The reason for this has already been explained—the differentials on individual coal and exceptionally good prices for steam fuel. From December, 1917, to December, 1918, the margins were less than 40 cents; the mean average margin for the entire period was 42 8 cents.

The foregoing observations have been based entirely on fresh-mined coal, for under normal conditions washery production is an inconsiderable factor. During the 2 years of the war, however, which are the 2 years covered by the Commission's report, washery coal made up a significant portion of the tonnage. For a number of years preceding the breaking out of the European war, washery production had been declining steadily from a maximum of 4,300,000 tons, in 1907, to 1,736,500 tons, in 1914. From this low point, it jumped to 6,370,000 tons, in 1917, and to 7,735,000 tons, in 1918. The low cost of this production and the exceptionally high prices it commanded in 1917 and 1918 were a saving grace for a large part of the anthracite industry. It increased the "margin," in the first period, from 63 cents a ton of fresh-mined coal to 64 cents on the total output; in the second period, from 79 cents to 83 cents; in the third

period, from 77 cents to 82 cents; in the fourth period, from 47 cents to 58 cents; and in the fifth period, from 36 cents to 54 cents.

The average margin on combined fresh-mined and washery coal




The Dollar Paid for Anthracite Fresh Mined Coal

FIG 1.

The Dollar Paid for Anthracite Washery Coal

FIG. 2.

The Dollar Paid for Anthracite Total Production (Fresh Mined and Washery)

FIG 3.

for the 2 years was 65.8 cents. Deduct from this the interest on the investment, 45 cents, and sales expenses, 8 cents (total 53 cents) and there remains the average profits of 12.8 cents, from which to pay Federal income and excess profits taxes, non-insurable risks, etc.

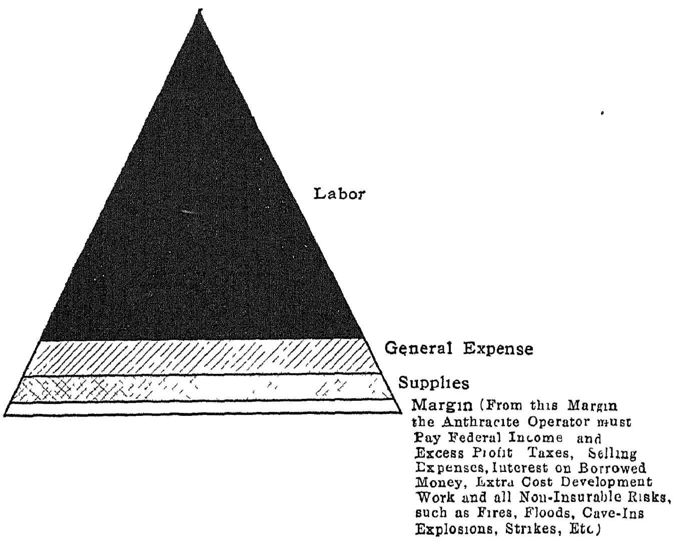



FIG. 4.—WHAT ONE TON OF ANTHRACITE COSTS TO TAKE FROM THE GROUND, PREPARE AND LOAD FOR SHIPMENT (AS OF DECEMBER, 1918, LATEST REPORT) AS REPORTED BY THE FEDERAL TRADE COMMISSION, JUNE 30, 1919.

The turn-over of the dollar on the total production was 11 cents in November and December, 1918, as compared with 7 cents for fresh-mined coal. The accompanying diagrams, Figs. 1 to 3, illustrate how

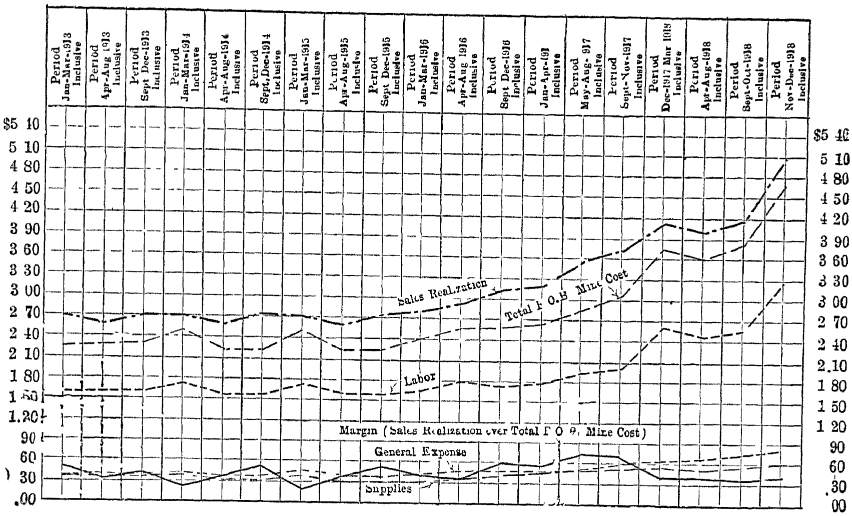



FIG. 5.—GROSS TON BASIS AS REPORTED BY FEDERAL TRADE COMMISSION, JUNE 30, 1919.


FIG 6.—MARGIN, PER CENT. SALES REALIZATION OVER TOTAL F. O. B. MINE COST AS REPORTED BY FEDERAL TRADE COMMISSION, JUNE 30, 1919.

the dollar spent for anthracite at the mines is divided up. The fourth diagram, Fig. 4, shows how the margins are squeezed down by the burden of labor, general expenses, and supplies. Fig. 5 shows the curves of costs, realizations, and margins, and Fig. 6 illustrates the rather violent fluctuations in the percentage of margins that have obtained during the 6 years covered by this portion of the Commission's report.

G. H. Ashley, Harrisburg, Pa.—An interesting thing has come up in recent years bearing on this question of costs. In former times, the prepared sizes had to carry a large part of the costs of the smaller sizes. During the past few years, coal which has been carried down the Susquehanna, Schuylkill, and other streams, has been recovered on a large scale, the recovery during the past year amounting to about 2,000,000 tons. The companies are encouraging this recovery because it has induced the use of furnace grates suited to this fine coal, and they feel that when all the river coal has been recovered, former users of that coal will turn to the small sizes from the mines and thus the latter will benefit.

# INDEX

[NOTE —In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics Casual notices, giving but little information, are usually indicated by bracketed page numbers The titles of papers presented, but not printed in this volume, are followed by bracketed page numbers only.]

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